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THE MINERAL INDUSTRY

ITS

STATISTICS, TECHNOLOGY & TRADE

1900

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ITS STATISTICS, TECHNOLOGY AND TRADE

IN THE

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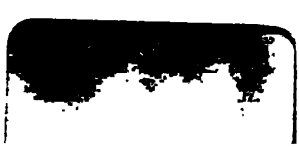

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RICHARD P. ROTHWELL,

Editor of the Engineering and Mining Journal; Ex-President American Institute of Mining Engineers; Member American Society of Civil Engineers; Fellow Royal Statistical Society, London; Member of American Statistical Association; etc.

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Page 119, last line, for "1898" read "1897."
Page 158, 12th line, for "3.86c." read "3.41c."
Page 164, in table for 1899, for "5,775,716" read "5,275,716."
Page 269, 4th line, for "29.76 tons" read "29.76c. per ton."
Page 482, 29th line, for "thiocyanate" read "thiocyanide."
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Inches to Millimeters.	Feet to Meters.	Yards to Meters.	Miles to Kilometers.		Drams to Cubic Centimeters.	Ounces to Milliliters.	Quarts to Liters.	Gallons to Liters.	Cubic Inches to Cubic Centimeters.	Cubic Feet to Cubic Meters.	Cubic Yards to Cubic Meters.	Bushels to Hectoliters.
25.4000	0.304801	0.914402	1.60935	= 1 =	3.70	29.57	0.94636	3.78544	16.387	0.02832	0.765	0.35242
50.8001	0.609601	1.828804	3.21869	= 2 =	7.39	59.15	1.89272	7.57088	32.774	0.05663	1.529	0.70485
76.2001	0.914402	2.743205	4.82804	= 3 =	11.09	88.72	2.83908	11.35632	49.161	0.08495	2.294	1.05727
101.6002	1.219202	3.657607	6.43739	= 4 =	14.79	118.30	3.78544	15.14176	65.549	0.11327	3.058	1.40969
127.0002	1.524003	4.572009	8.04674	= 5 =	18.48	147.87	4.73180	18.92720	81.936	0.14158	3.823	1.76211
152.4003	1.828804	5.486411	9.65608	= 6 =	22.18	177.44	5.67816	22.71264	98.323	0.16990	4.587	2.11454
177.8003	2.133604	6.400813	11.26543	= 7 =	25.88	207.02	6.62452	26.49808	114.710	0.19822	5.352	2.46696
203.2004	2.438405	7.315215	12.87478	= 8 =	29.57	236.59	7.57088	30.28352	131.097	0.22654	6.116	2.81938
228.6004	2.743205	8.229616	14.48412	= 9 =	33.28	266.16	8.51724	34.06896	147.484	0.25485	6.881	3.17181

Square.					Weight.			
Square Inches to Square Centimeters.	Square Feet to Square Decimeters.	Square Yards to Square Meters.	Acres to Hectaers.		Grains to Milligrams.	Avoirdupois Ounces to Grams.	Avoirdupois Pounds to Kilograms.	Troy Ounces to Grams.
6.452	9.290	0.836	0.4047	= 1 =	64.7989	28.3495	0.45359	31.10348
12.903	18.581	1.672	0.8094	= 2 =	129.5978	56.6991	0.90719	62.20696
19.355	27.871	2.508	1.2141	= 3 =	194.3968	85.0486	1.36078	93.31044
25.807	37.161	3.344	1.6187	= 4 =	259.1957	113.3981	1.81437	124.41392
32.258	46.452	4.181	2.0234	= 5 =	323.9946	141.7476	2.26796	155.51740
38.710	55.742	5.017	2.4281	= 6 =	388.7935	170.0972	2.72156	186.62089
45.161	65.032	5.853	2.8328	= 7 =	453.5924	198.4467	3.17515	217.72437
51.613	74.323	6.689	3.2375	= 8 =	518.3914	226.7962	3.62874	248.82785
58.065	83.613	7.525	3.6422	= 9 =	583.1903	255.1457	4.08233	279.93133

1 chain	=	20.1169 meters.
1 square mile	=	259 hectaers.
1 fathom	=	1.829 meters.
1 nautical mile	=	1853.27 meters.
1 foot=0.304801 meter,		9.4840158 log.
1 avoir. pound	=	453.5924277 gram.
15432.35639 grains	=	1 kilogram.

TABLES FOR CONVERTING METRIC TO UNITED STATES WEIGHTS AND MEASURES.

Linear.					Capacity.							
Meters to Inches.	Meters to Feet.	Meters to Yards.	Kilometers to Miles.		Milliliters or Cubic Centiliters to Fluid Drams.	Centiliters to Fluid Ounces.	Liters to Quarts.	Dekaliters to Gallons.	Hektoliters to Bushels.	Cubic Centimeters to Cubic Inches.	Cubic Meters to Cubic Feet.	Cubic Meters to Cubic Yards.
39.3700	3.28083	1.093611	0.62137	= 1 =	0.27	0.338	1.0567	2.6417	2.8375	0.0610	35.314	1.308
78.7400	6.56167	2.187222	1.24274	= 2 =	0.54	0.676	2.1134	5.2834	5.6750	0.1220	70.629	2.616
118.1100	9.84250	3.280833	1.86411	= 3 =	0.81	1.014	3.1700	7.9251	8.5125	0.1831	105.943	3.924
157.4800	13.12333	4.374444	2.48548	= 4 =	1.08	1.353	4.2267	10.5668	11.3500	0.2441	141.258	5.232
196.8500	16.40117	5.468056	3.10685	= 5 =	1.35	1.691	5.2834	13.2085	14.1875	0.3051	176.572	6.540
236.2200	19.68500	6.561667	3.72822	= 6 =	1.62	2.029	6.3401	15.8502	17.0250	0.3661	211.887	7.848
275.5900	22.96583	7.655278	4.34959	= 7 =	1.89	2.368	7.3968	18.4919	19.8625	0.4272	247.201	9.156
314.9600	26.24667	8.748889	4.97096	= 8 =	2.16	2.706	8.4534	21.1336	22.7000	0.4882	282.516	10.464
354.3300	29.52750	9.842500	5.59233	= 9 =	2.43	3.043	9.5101	23.7753	25.5375	0.5492	317.830	11.771

Square.					Weight.			
Square Centimeters to Square Inches.	Square Meters to Square Feet.	Square Meters to Square Yards.	Hectares to Acres.		Kilograms to Grains.	Hectograms to Ounces Avoirdupois.	Kilograms to Pounds Avoirdupois.	Grams to Ounces Troy.
0.1550	10.764	1.196	2.471	= 1 =	15432.36	3.5274	2.20462	0.03215
0.3100	21.528	2.392	4.942	= 2 =	30864.71	7.0548	4.40924	0.06430
0.4650	32.292	3.588	7.413	= 3 =	46297.07	10.5822	6.61386	0.09645
0.6200	43.055	4.784	9.884	= 4 =	61729.43	14.1096	8.81849	0.12860
0.7750	53.819	5.980	12.355	= 5 =	77161.78	17.6370	11.02311	0.16075
0.9300	64.583	7.176	14.826	= 6 =	92594.14	21.1644	13.22773	0.19290
1.0850	75.347	8.372	17.297	= 7 =	108026.49	24.6918	15.43235	0.22505
1.2400	86.111	9.568	19.768	= 8 =	123458.85	28.2192	17.63697	0.25721
1.3950	96.874	10.764	22.2	= 9 =	138891.21	31.7466	19.84159	0.28936

The only material standard of customary length authorized by the U. S. Government is the Troughton scale, whose length at 59°.63 Fahr. conforms to the British standard. The yard in use in the United States is therefore equal to the British yard.

The only authorized material standard of customary weight is the Troy pound of the Mint. It is of brass

of unknown density, and therefore not suitable for a standard of mass. It was derived from the British standard Troy pound of 1758 by direct comparison. The British avoirdupois pound was also derived from the latter, and contains 7,000 grains troy.

The grain Troy is therefore the same as the grain avoirdupois, and the pound avoirdupois in use in the United States is equal to the British pound avoirdupois.

The British gallon = 4·54346 liters.

The British bushel = 36·3477 liters.

By the concurrent action of the principal Governments of the world an International Bureau of Weights and Measures has been established near Paris. Under the direction of the International Committee, two ingots were cast of pure platinum-iridium in the proportion of 9 parts of the former to 1 of the latter metal. From one of these a certain number of kilograms were prepared, from the other a definite number of meter bars. These standards of weight and length were intercompared, without preference, and certain ones were selected as International prototype standards. The others were distributed by lot to the different Governments and are called National prototype standards.

The metric system was legalized in the United States in 1866.

The International Standard Meter is derived from the Metre des Archives, and its length is defined by the distance between two lines at 0° Centigrade, on a platinum-iridium bar deposited at the International Bureau of Weights and Measures.

The International Standard Kilogram is a mass of platinum-iridium deposited at the same place, and its weight *in vacuo* is the same as that of the Kilogramme des Archives.

The liter is equal to a cubic decimeter of water, and it is measured by the quantity of distilled water which, at its maximum density, will counterpoise the standard kilogram in a vacuum, the volume of such a quantity of water being, as nearly as has been ascertained, equal to a cubic decimeter.

Long ton: 2240 lb. avoirdupois	= 1016 kilogram.	Barrel of petroleum	= 42 gal.	= 1·59	hectoliter.
Short ton: 2000 " "	= 907·2 "	" " salt	= 280 lb.	= 127	kilogram.
Pound avoirdupois	= 453·6 grams.	" " lime	= 200 "	= 90·720	"
Flask of Mercury = 76½ lb. avoir.	= 34·700 kilograms.	" " natural cement,	= 300 "	= 136·080	"
Troy ounce	= 31·104 grams.	" " Portland cement.	= 400 "	= 181·440	"
Gallon	= 3·785 litres.	Gold coining value per oz. Troy	$20·6718	= $0·6646	per gram.
		Silver " " " Troy	$1·2929	= $0·04157	"

OFFICIAL UNITED STATES VALUES OF FOREIGN COINS, APRIL 1, 1900.

Country.	Standard.	Unit.	Value in U. S. Gold.	Coins.
			cts.	
Argentina	Both	Peso	96·5	Gold: argentine ($4·824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	20·3	Gold: former system—4 florins ($1·929), 8 florins ($3·858), ducat ($2·287), and 4 ducats ($9·149). Silver: 1 and 2 florins. Present system—Gold: 20 crowns ($4·052) and 10 crowns ($2·026).
Belgium	Both	Franc	19·3	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Silver	Boliviano	43·6	Silver: boliviano and divisions.
Brazil	Gold	Milreis	54·6	Gold: 5, 10, and 20 milreis. Silver: ½, 1, and 2 milreis.
Canada	Gold	Dollar	100·0	
Central America—				
Costa Rica	Gold	Colon	46·5	Gold: 2, 5, 10, and 20 colons ($9·307). Silver: 5, 10, 25, and 50 centimos.
British Honduras	Gold	Dollar	100·0	
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	43·6	Silver: peso and divisions.
Chile	Gold	Peso	36·5	Gold: escudo ($1·825), doubloon ($3·650), and condor ($7·300). Silver: peso and divisions.
China	Silver	Tael*	71·7	
Colombia	Silver	Peso	43·6	Gold: condor ($9·647) and double condor. Silver: peso.
Cuba	Both	Peso	92·6	Gold: centen ($5·017). Silver: peso.
Denmark	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Ecuador	Silver	Sucre	43·6	Gold: condor ($9·647) and double condor. Silver: sucre and divisions.
Egypt	Gold	Pound	494·3	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters. Silver: 1, 2, 5, 10 and 20 piasters.
Finland	Gold	Mark	19·3	Gold: 20 marks ($3·859), 10 marks ($1·93).
France	Both	Franc	19·3	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	23·8	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound	486·65	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Both	Drachma	19·3	Gold: 5, 10, 20, 50 and 100 drachmas. Silver: 5 drachmas.
Haiti	Both	Gourde	96·5	Silver: gourde.
India	Silver	Rupee	32·7	Gold: mohur ($7·105). Silver: rupee and divisions.
Italy	Both	Lira	19·3	Gold: 5, 10, 20, 50 and 100 lire. Silver: 5 lire.
Japan	Gold	Yen	49·8	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.
Liberia	Gold	Dollar	100·0	
Mexico	Silver	Dollar	47·3	Gold: dollar ($0·983), 2½, 5, 10, and 20 dollars. Silver: dollar (or peso) and divisions.
Netherlands	Both	Florin	40·2	Gold: 10 florins. Silver: ½, 1, and 2½ florins.
Newfoundland	Gold	Dollar	101·4	Gold: 2 dollars ($2·027))
Norway	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Peru	Silver	Sol	43·6	Silver: sol and divisions
Portugal	Gold	Milreis	108·0	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	51·5	Gold: imperial 15 rubles ($7·718) and ½ imperial 7½ rubles ($3·859). Silver: ¼, ½, and 1 ruble.
Spain	Both	Peseta	19·3	Gold: 25 pesetas. Silver: 5 pesetas.
Sweden	Gold	Crown	26·8	Gold: 10 and 20 crowns.
Switzerland	Both	Franc	19·3	Gold: 5, 10, 20, 50 and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	04·4	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	103·4	Gold: peso. Silver: peso and divisions.
Venezuela	Both	Bolivar	19·3	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

*Haikwan (Customs).

RICHARD P. ROTHWELL.

JOSEPH STRUTHERS.

CONTRIBUTORS.

IT is impossible to name here all who have aided us in the collection of statistics and other information for the present volume, but we give in the following pages brief biographies of most of those who have contributed special articles, in order that readers may appreciate the high professional standing of those who have assisted in the work.

Besides the contributors of special articles, however, the preparation of this volume has been aided by the courteous co-operation of many thousands of producers who have furnished statistics of their output, and by many persons prominent in various branches of the mineral industry who have given special information. Exceedingly valuable assistance has been furnished also by the officials of many railways in the United States and Mexico, and by the State geologists, commissioners of mines, and inspectors of mines in most of the States of the Union. The statisticians of foreign countries have been extremely courteous in their co-operation, by furnishing copies of their latest publications, often in manuscripts. Professional men and experts of the whole world have rendered exceedingly valuable assistance, as have also the officials of the United States Government at Washington and abroad, and have added greatly to the value of this work. Among the thousands who have thus aided us, and by their assistance made possible the publication of this volume, as well as its predecessors, it would be invidious to select names, and in making such an attempt we should not know where to draw the line, since the contributions of almost all have been indispensable. Consequently we have decided to limit ourselves to this general acknowledgment, relying upon the belief that each of our friends will feel amply repaid for his work in the knowledge that he has contributed to the preparation of a volume which is everywhere recognized to be of the highest value to the mineral industry of the world. This high appreciation has been generously and delicately expressed by the French "Société d'Encouragement pour l'Industrie Nationale," which, since the appearance of Vol. VI., has granted to THE MINERAL INDUSTRY and its editor, the magnificent gold medal of the society, which is voted to the work or the author of the work, which, during the six preceding years, has contributed most to the cause of the national industry.

ACHESON, EDWARD G., born in 1856, at Washington, Pa., studied in the academy at Bellefonte, Pa. From the age of 18 he was engaged for several years in railway construction. In 1880 he joined the staff of Thomas A. Edison, at Menlo Park, N. J., with which he remained about three years. During that time he had supervision of the construction of the Edison lamp factories at Paris, and installed and started the first incandescent electric lighting plants in Holland, Belgium and Italy. Returning to the United States in 1884 he spent two years in experimental work, and then became electrician to the Standard Underground Cable Co., of Pittsburg, Pa. In 1890 he organized the Monongahela Electric Light Co., and in his laboratory connected therewith produced, in March of the following year, the new substance carborundum. In the fall of 1891 the Carborundum Co. was incorporated, with Mr. Acheson as its president. In 1898, he completed the organization and incorporation of the Acheson Graphite Co., formed for the manufacture of graphite from coke. He contributes to this volume the notes on artificial graphite.

AICHINO, GIOVANNI, was born in 1860 at Rapallo. He received his diploma as engineer at Turin in 1882. Having entered the "Corps Royal des Mines," he spent some years in Belgium, and made a study of certain mining and metallurgical subjects at the School of Mines at Liege. He also went to England in order to visit the industrial establishments of that country, and to become acquainted with the English Geeological Survey. Returning to Italy, he was soon engaged by the Geological Bureau at Rome, and has devoted his attention particularly to mineral analysis. For many years he has been a collaborator on the *"Annuaire Geologique Universel de Dagincourt,"* and has written many articles on metallurgy for the *"Enciclopedia delle Arti e Industrie de Pareto et Sacheri"*

and for the *"Rassegna Mineraria."* He has contributed to this volume the thorough article on "Sulphur in Italy."

BEALS, WILLIAM, JR., graduated from Harvard in 1888 with degree of *magna cum laude*, and for several years following was a member of the Archæan division of the United States Geological Survey, working mainly in Vermont and eastern New York. Mr. Beals was manager of granite and marble quarries for one year, and since 1894, has been established professionally in Boston as geologist and mining engineer. For the present volume Mr. Beals has assisted Mr. Whitt in the preparation of the article on "Stone."

BORCHERS, WILHELM, PH.D., was born in 1856, in Goslar, Hanover, and studied at the universities of Griefswald (1875-76), Erlargen (1876-78), and Munich (1878-79). In 1879 he became connected with the chemical works of E. de Haën in Hanover. In 1882 he went to the United States, and built the works of the Colonial Chemical Co. in West Medford, Mass., which he managed until 1887, when he returned to the works of De Haën in Hanover. In 1891 he left this position to make some special studies in the Bergakademie at Clausthal and to devote himself to electro-metallurgy, but in the following year he was made instructor in metallurgy and technical chemistry in the Royal Engineering and Metallurgical School at Duisburg. In 1897 he became professor of metallurgy at the Royal College of Technology at Aachen in Prussia. Dr. Borchers has written and published *Elektrometallurgie* (1891), *Anorganische Chemie* (1893), and in connection with Dr. W. Nernst, *Jahrbuch der Elektrochemie* (1895 to 1899). He is now one of the editors of *Zeitschrift für Elcktrochemie.* He contributes to the present volume the article on the "Progress of Electro-Chemistry in 1899."

BRANNER, JOHN CASPER, was born July 4, 1850, at New Market, Tenn., and was educated at Cornell University, graduating in 1874. Directly after his graduation he became a member of the Imperial Geological Survey of Brazil, and remained in that country until 1882. From 1882 to 1885 Mr. Branner served as assistant to the Second Geological Survey of Pennsylvania, and in 1885 was appointed the professor of geology at the University of Indiana, holding that position until 1891, when he accepted the professorship of geology in the Leland Stanford, Jr., University, and was appointed to the vice-presidency of the University in February, 1899. From 1887 to 1892 he was State Geologist of Arkansas. Prof. Branner is a member of several scientific societies, and has contributed to the present volume notes on the occurrence of diamonds and manganese ores in Brazil.

CARACRISTI, C. F. L. ZEILINGER (Count Caracristi), was born in 1856, and received his early education in Austria. He graduated from the Royal Italian Polytechnic in 1876, and served as Lieut. of Engineers in the Italian army, being detailed to report on the world's production of marble. In 1885 and 1886 he was professor of engineering and physics in the Maryland Military and Naval Academy, and served as secretary to Gen. Joseph Wheeler during 1887 and 1888. In 1890-91 he investigated the fuel supply of the coal and petroleum fields of South America, and reported on the fitness of these deposits as a base of fuel supply for the proposed isthmian canals, and for the navy in case of war. Mr. Caracristi studied law at Columbian University, Washington, D. C., and was admitted to the bar. At present he is consulting engineer for a number of companies interested in oil fields. He has written numerous scientific articles, and has contributed to the present volume the paper on "The World's Production of Petroleum."

CLAREMONT, LEOPOLD, born in London, was articled at an early age to Richard Cook Nockhold, precious-stone merchant, of London, and during his apprenticeship devoted himself to the study of the sciences which pertain to the business, so that by the end of his apprenticeship he had become an expert mineralogist as well as a practical gem cutter. For a while he acted as curator of a collection of rare gems exhibited at the Imperial Institute, and was afterward appointed manager of the gem-cutting establishment of Streeter & Co., Ltd., Glasshouse Street, London, which business he finally purchased and is now carrying on at the same address. Mr. Claremont has gained a high reputation as a gem expert through the application of scientific methods to this industry. He has been

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

WILHELM BORCHERS.

WILLIAM BEALS, JR.

JOHN C. BRANNER.

C. F. Z. CARACRISTI.

LEOPOLD CLAREMONT.

HANS A. FRASCH.

JAMES DOUGLAS.

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

AUGUST HECKSCHER.

ALBERT GROTHE.

H. O. HOFMAN.

OTTOKAR HOFMANN.

HENRY MARION HOWE.

WALTER RENTON INGALLS.

WILLIAM KENT.

a contributor to numerous technical periodicals, and for this volume has written the paper on "The Cutting and Polishing of Precious Stones."

DE KALB, COURTENAY, was born in London County, Va., in 1861, and was educated at Syracuse University. His early professional career was followed in various positions connected with mining and metallurgy in the West and South, and from 1889 until 1893 his attention was given to gold mining in South and Central America. In 1894 he was appointed professor of mining and metallurgy at the Missouri School of Mines, subsequently resigning to accept a similar position in the School of Mining, Kingston, Ontario, Can. Mr. De Kalb is also established in New York City as a consulting engineer, and is connected in that capacity with several coal mining companies in the United States, and with the Cocheño Mining Co., Mexico. For the present volume he has contributed notes on "Corundum in Ontario, and Abrasive Tests."

DOUGLAS, JAMES, is a Canadian, who has made his home in the United States since 1875. His first experience in mining and metallurgy was acquired in trying to unravel the complicated affairs of an unsuccessful Canadian mining enterprise. He came to the States in order to take charge of copper works established in Phœnixville, Pa., for the utilization of local copper ores, whose supply, however, proved deficient; but he is best known through his connection with the copper industry of Arizona, with which he has been intimately associated almost since its initiation. He is president of the American Institute of Mining Engineers; is also president of the Copper Queen Consolidated Mining Co., and of other Arizona concerns, and is also president of the Arizona and Southern Railroad Co. Such original work as he has done was chiefly in connection with the late eminent chemist, Dr. T. Sterry Hunt, in the field of the hydrometallurgy of copper. Mr. Douglas contributes to this volume the notes on the copper industry in Arizona.

FALDING, F. J., was born in England and educated at Amersham Hall, London, and at the Bergakademie at Freiberg in Saxony, but did not graduate from the latter. After emigrating to America he spent a year and a half in the Bank of Montreal, and about three years in the office of the Attorney-General of Canada. In 1878-79 he made a study of the Canadian apatite deposits, and in 1880 returned to Europe, and studied the manufacture of sulphuric acid and fertilizers in England and Germany. In 1881 he returned to Canada and unwatered the Capelton pyrites mines, now worked by G. H. Nichols & Co. From 1882 to 1886 he practiced as a mining engineer, with headquarters in New York, making a specialty of pyrites and phosphate mining, and made several visits to Europe to study mines and chemical and metallurgical works there. In 1888 he entered the employ of the Grasselli Chemical Co. as engineer, in charge of its mines, and in 1890 became chief engineer for that company with charge of all its works. During this time he designed the company's new works at East Chicago, Ill. In 1895 he established himself in New York as a consulting chemical engineer, since which time he has constructed and rebuilt many sulphuric acid plants in various parts of the United States. He contributes to this volume the paper on the "Manufacture of Chamber Sulphuric Acid Stronger than Chamber Acid."

FERRY, CHARLES, was born in 1868 at Peterborough, N. H. He received his early education at Williston Seminary, Easthampton, Mass., and graduated at the Worcester Polytechnic Institute, Worcester, Mass., in 1888. For two years after leaving Worcester he was assistant to Dr. T. M. Drown, who was then director of the chemical department of the Massachusetts Institute of Technology in Boston. There he gathered much valuable professional experience, and acquired an interest in scientific research, which finally became concentrated upon refractory materials. For seven years Mr. Ferry has been chemist of the Ostrander Fire Brick Co., the main office and laboratories of which are located at Troy, N. Y. In the interests of this company Mr. Ferry's practical and theoretical investigations have been along the lines of fire clays and their proper application to various classes of work. He has written for this volume the paper on "Sand for Rolling Mill Reheating Furnace Bottoms."

FRASCH, H. A., was born and educated in Germany, coming in 1876 to the United

States, where he first was engaged in the sugar business in Louisiana, remaining there until 1882, when he joined his brother, who was then erecting ammonia soda works at Bay City, Mich. In 1886 he accepted the technical management of a cement works at Kansas City, and established a mineral paint factory. From 1888 to 1891 he was engaged in the development of the salt and soda ash works at Cleveland, Ohio, and then took charge of the department of technical research of the Grasselli Chemical Co. at Cleveland. In this capacity he discovered the presence of valuable organic acids in petroleum refuse, and devised methods for their separation and utilization in the production of dye stuffs. While with the Grasselli Chemical Co. he also constructed a plant for the manufacture of artificial asphalt from petroleum refuse, and an apparatus for the production of aqua ammonia direct from gas liquor. He left the Grasselli Co. in 1896 in order to construct an asphaltum refining works in Texas. Lately Mr. Frasch has been established as consulting chemical engineer in New York. He contributes to this volume the article on "Lithia Salts and their Manufacture."

GROTHE, ALBERT, was born at Hagen, Prussia, in 1841, and was educated as a civil engineer at the technical college at Utrecht, Holland. In 1862 he visited Java, and was employed on the construction of the railroad from Samarang to the interior. In 1867 he was appointed chief engineer of the Batavia Tramway Co., and after constructing 26 miles of road, he returned to Europe, and in 1870 superintended the construction of a number of bridges on the railway from Tamboff to Saratoff on the Volga in southeastern Russia. In 1871 he was in charge of the construction of the bridge across the Tay in Scotland, at that time the longest bridge in the world, having a length of two miles, and a height of 98 ft. above water. In 1879 Mr. Grothe was appointed general manager of the mines and works in Spain controlled by the Tharsis Sulphur & Copper Co. and after four years in their employ he developed several mines in the south of Portugal. Since 1884 he has established a business as consulting engineer, and has reported on many properties in Mexico, New Zealand, and the United States. Mr. Grothe has experimented with the Snake River gravels, and has contributed to the present volume the article on "Gold Dredging in the United States." At present Mr. Grothe is in charge of the Tepezala mine, a large copper deposit near Aguas Calientas, Mexico.

HECKSCHER, AUGUST, was born at Hamburg, Germany, and educated in Germany and Switzerland. He was actively connected with the anthracite coal mining industry in the United States from 1868 to 1882. Since 1881 he has been engaged in the zinc mining and manufacturing industry in New Jersey, Pennsylvania, Illinois, Wisconsin and Missouri. At present he is general manager of the New Jersey Zinc Co.

HOFMAN, H. O., was born in 1852 at Heidelberg, Germany. He studied at the Bergakademie at Clausthal, where he graduated in 1877 in mining engineering and metallurgy. He was then appointed chemist and assistant at the smelting and refining works at Lautenthal in the Harz. In 1881 he came to the United States and was employed successively at Mine La Motte in Missouri, at the Argentine smelting and refining works of the Consolidated Kansas City Smelting and Refining Co., and as metallurgist of the Delaware Lead Co., in Philadelphia. When the last named works were closed he went to Colorado, and after running the Rico smeltery for a short time went to Park City, Utah, to study the amalgamation and lixiviation of silver ores at the Ontario mill. After a short time spent in charge of a smeltery in Mexico he was appointed assistant to Prof. Richards at the Massachusetts Institute of Technology in Boston; from there he went to the School of Mines of South Dakota as professor of metallurgy and assaying, where he remained until called back to the Massachusetts Institute of Technology to the professorship of metallurgy, which he now holds. Dr. Hofman has made numerous contributions to technical literature, his most important work being the admirable treatise on *The Metallurgy of Metal.* For his paper on the "Dry Assay of Tin Ores" the degree of Ph.D was conferred on him by the University of Ohio. For the present volume he has reviewed progress in metallurgy of lead, continuing the series of articles begun in the first volume.

HOFMANN, OTTOKAR, was born in 1843 at Rusberg in Austria-Hungary. He was educated at the Realschule and Polytechnic Institute in Vienna, and the Bergakademie at Freiberg, graduating from the last in 1866. In 1867 he came to the United States, and first entered the assay office of Guido Küstel in San Francisco as a partner. Mr. Hofmann has been chiefly identified with leaching processes for the extraction of gold and silver from ores. In 1868 he erected, at La Dura, Sonora, Mex., the first lixiviation works operated in North America, following its success with the construction of other works at La Trinidad and San Marcial, Sonora, and at Trinufo, Lower California. He has served as metallurgist with various amalgamation and chlorination mills in Nevada and California, and, since 1880, he has been occupied almost exclusively in hydrometallurgy, and has been engaged successively at St. Louis, Mo. (1882-83); Cusihuiriachic, Chihuahua (1885); Yedras, Sinaloa (1886-87); Cusihuiriachic (1887); Parral, Chihuahua (1888); Cusihuiriachic (1890-94); Sombrerete and Parral (1894). In 1895 Mr. Hofmann took charge of the leaching department of the Consolidated Smelting and Refining Co. at Argentine, Kan., and in 1897 he displaced the Hunt & Douglas process, which had been used until that time, by a process, that would free in one operation copper sulphate solutions from iron, arsenic, antimony and other impurities. He is still in charge of these works. Mr. Hofmann has contributed frequently to metallurgical literature, chiefly through the *Engineering and Mining Journal*, and the *Transactions of the American Institute of Mining Engineers*. For the present volume he has written the paper on "Hofmann's Method for the Manufacture of Blue Vitriol."

HOWE, HENRY M., graduated at Harvard College in 1869, and at the Massachusetts Institute of Technology in Boston in 1871, paying especial attention while at the latter institution to the metallurgy of iron and steel. In 1872 he became superintendent of the Bessemer steel works at Joliet, Ill., and, until 1883, he was actively engaged in metallurgical manufactures, chiefly of iron and steel. From then until the present time he has been a consulting metallurgist, attending almost exclusively to the metallurgy of iron and steel. For a number of years Mr. Howe was lecturer at the Massachusetts Institute of Technology, and is now professor of metallurgy at Columbia University School of Mines. He is one of the most eminent authorities on the metallurgy of iron and steel, and his close and careful studies, together with his thorough and scientific experimental work have been generally recognized. In 1895 the Franklin Institute of Philadelphia awarded him the Elliot Cresson gold medal for his researches on the nature and properties of iron and steel, and in the same year, the Iron and Steel Institute of Great Britain awarded him its Bessemer gold medal in recognition of his scientific contributions to metallurgical literature; the French Société d'Encouragement pour l'Industrie Nationale awarded him a prize of 2,500 francs for his monumental treatise on the *Metallurgy of Steel*, which, besides running through several American editions, has been translated and published in French. In addition, the Verein zur Befoerderung des Gewerbfleisses of Berlin awarded him its great gold medal for scientific research. Mr. Howe is a member of many scientific and technical societies in the United States and Europe, and is past president of the American Institute of Mining Engineers. He was also president of the jury on "Mines and Mining" at the World's Columbian Exposition in Chicago in 1893. For the present volume he has written the paper on the "Progress in Iron and Steel Metallurgy in 1899."

ILES, MALVERN W., graduated from the School of Mines, Columbia College (now Columbia University), in 1875, and after two years of scientific work at Columbia and at Johns Hopkins University, he received the degree of Ph.D. from the former. Mr. Iles has been prominent in the engineering profession for his special work in the metallurgy of lead; he was identified in many capacities with the Omaha & Grant Smelting and Refining Co. from 1879-1886, when he resigned and became the metallurgist of the Globe Smelting and Refining Co., and its superintendent from 1887 until 1899. He is now traveling in Europe. Mr. Iles is a member of numerous chemical and engineering societies, and has written many articles for various scientific publications. He has contributed for the present volume the paper on "Notes on the Moebius Electrolytic Process."

INGALLS, WALTER RENTON, a well-known mining engineer and metallurgist, was born at Lynn, Mass., in 1865, and graduated from the Massachusetts Institute of Technology, in 1886. In 1886-90 he was engaged in mining at Leadville and elsewhere in Colorado. In 1890-92 he was assistant editor of the *Engineering and Mining Journal*, resigning that position to go to Mexico to open tin mines in the State of Durango for the Pittsburg & Mexican Tin Mining Co. In 1893 and 1894 he established himself in New York, and visited professionally various mining districts in the United States, Canada, Belgium, Germany and Poland, devoting himself especially to the metallurgy of zinc. During a part of 1894 he had charge of the operations of the Illinois Phosphate Co. in Florida, and later in the year became connected with the Gold and Silver Extraction Co. of America, Ltd., as metallurgist. In 1895 he was manager of a cyanide works at Cripple Creek, Colo., and in 1896 of copper-matte smelting works in Durango, Mexico, returning to New York in 1897. He was assistant editor of THE MINERAL INDUSTRY, Vols. V., VI. and VII., and is now located in Boston, Mass., as consulting engineer. For the present volume he has contributed the paper on "The Progress in the Metallurgy of Zinc in the United States during 1899."

JANIN, LOUIS, JR., is a mining engineer and metallurgist of wide and varied experience in various portions of the world. Born in Virginia City, Nev., he attended the University of California, and later entered into active work in the gold mines of that State. Later he was metallurgist of the Ontario Silver Mining Co., at Park City, Utah, where he made the first recorded experiments on any scale with potassium cyanide as a means of extracting gold and silver. Considerable Mexican experience followed this, and for many years he was engaged professionally in examining mines in various parts of the United States, Mexico, Central and South America. In 1894 he became connected with the various companies controlling the MacArthur Forrest cyanide patents, and in this connection made a long visit to Australia. In that country he became interested in mining, and acted as consulting engineer or general manager for various companies, notably copper-gold mines at Cobar, N. S. W. Mr. Janin has been a frequent contributor to the *Engineering and Mining Journal*, and at one period was assistant editor of that journal. To Vol. I. of THE MINERAL INDUSTRY he contributed an article on "The Cyanide Process," which was the first comprehensive one published on that subject, and to Vol. II. a thorough article on "The Amalgamation of Free-Milling Gold Ores." For the present volume, in addition to the signed papers on "Progress in the Cyanide Process," and "Copper Mining in New South Wales," he has contributed numerous unsigned articles.

KENT, WILLIAM, well known to the engineering profession as the author of "The Mechanical Engineers' Hand-Book," was born in 1851 in Philadelphia. He studied at Stevens Institute, where he graduated in 1876 in mechanical engineering, and spent an additional year making a research on the alloys of copper, tin and zinc for the United States Testing Board under the direction of Prof. R. H. Thurston. He went to Pittsburg in 1877, where he became editor of *The American Manufacturer and Iron World*, and shortly after he was appointed superintendent of the open-hearth steel department of Shoenberger & Co. He was one of the organizers of the Pittsburg Testing Laboratory, and in 1882 he became manager of the Pittsburg office of the Babcock & Wilcox Co., removing to New York City in the following year to take the position of superintendent of the sales department and engineer of tests. From 1885 to 1890 Mr. Kent was general manager of the Springer Torsion Balance Co., and by his efforts did much to make the work of the company a success. From 1890 until the present time he has been engaged in professional practice, and since 1895 he has been associate editor of *The Engineering News*. He is a member of numerous engineering societies, and has frequently contributed to their proceedings. For the present volume he has written the paper on "Fuel and Its Economical Utilization."

KERSHAW, JOHN B. C., was born at Southport, Eng., in 1862, and was educated at Bickerton House School, Southport, and at Owens College, Manchester. In 1879 Mr. Kershaw entered the Sutton Lodge Chemical Works, St. Helens, Eng., and remained

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

JOHN B C. KERSHAW.

VICTOR LENHER.

ROBERT LINTON.

FRANZ MEYER.

J. OBALSKI.

WILTON P. RIX.

ROBERT H. RICHARDS.

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.

WILBUR E. SANDERS.

ALBERT SAUVEUR.

T. O'CONOR SLOANE.

HENRY E. SOUTHER.

TITUS ULKE.

ALFRED T. WEIGHTMAN.

CHARLES L. WHITTLE.

there for twelve years, rising in this period to the position of chief chemist and assistant manager. In 1892 Mr. Kershaw went to Germany and pursued the studies of chemistry and allied sciences at Bonn University. Since 1896 he has been engaged in practice as consulting chemist and as a technical journalist in London, and has devoted himself especially to work relating to electro-chemical processes and industries. He is a member of several chemical societies, and is also on the staff of abstractors for *The Journal of the Society of Chemical Industry* and for *Science Abstracts*. Mr. Kershaw has written numerous articles in recent years upon electro-chemical and electro-metallurgical subjects, and is the translator and editor of Dr. Neumann's German work on *Electrolytic Methods of Analysis*. To the present volume he has contributed special articles on "Aluminum," "Calcium Carbide," "Chlorates" and "Sulphuric Acid."

LENHER, VICTOR, graduated in chemistry in 1893 from the University of Pennsylvania, and for three years following he was assistant to Prof. W. B. Rising in the chemical department of the University of California. From 1896 to 1898 he was engaged in pursuing special studies in chemistry, mineralogy and geology at the University of Pennsylvania, receiving the degree of Ph.D. from that institution in 1898. Dr. Lenher has been a regular contributor to various chemical journals, and is now engaged in translating Moissan's "Le Four Electrique." He is on the staff of instruction in chemistry at Columbia University, and has contributed to the present volume the article on "Rare Earths."

LEWIS, FREDERICK H., studied civil engineering at the University of Pennsylvania, graduating in 1878. He then became heliotroper on the United States Coast Survey, serving during the summer of 1878, and for three years afterward was assistant engineer of the construction department of the Pennsylvania Railroad Co.'s lines west of Pittsburg. From 1882 to 1885 he was superintendent of bridges and buildings of the Northern Pacific Railway, being situated at St. Paul, Minn., and was also in charge of the location of the company's terminal lines between St. Paul and Minneapolis. In 1885 and 1886 he was resident engineer of the South Pennsylvania Railroad, at Sideling Hill tunnel, Fulton County, Pa. From 1886 to 1893 he was an Eastern manager of the Pittsburg Testing Laboratory at Philadelphia. Since 1893 he has been practicing as consulting engineer at Philadelphia, and has also been consulting engineer for the firm of Booth, Garrett & Blair, in their department of physical tests and inspection. He is now manager and chief engineer of the Virginia Portland Cement Co., and contributes to this volume the paper on "The Hydraulic Cement Industry in the United States in 1899."

LINTON, ROBERT, was born at Hudson, O., in 1870. He was educated at Washington and Jefferson College, Washington, Pa., and at the Royal Polytechnic School, Berlin. After graduation he was engaged for a time in mining engineering in the Pennsylvania bituminous region near Pittsburg, Pa., and, at a later date, entered the employ of the R. C. Schmertz Glass Co., continuing with them until the formation of the American Window Glass Co. At present Mr. Linton is manager of their works at Belle Vernon, Pa. He has made numerous contributions to several technical publications, and for the present volume, he has contributed the paper on "Glass."

LOUIS, HENRY, was born in London in 1855. He first attended school in Bavaria, then the City of London School, and in 1873 entered the Royal School of Mines, the associateship of which he obtained in 1876. He was then engaged for some time in Dr. Percy's private laboratory, making some of the researches for Dr. Percy's treatise on *Silver*. Thence he was sent by Sir William Siemens to Nova Scotia, for the Steel Co. of Canada. Later he went to South America, where he was employed in gold mining in Colombia and Ecuador; thence to the Gold Coast, and next to South Africa, where he spent over three years in gold and diamond mining. He then spent three years managing and inspecting mines and exploring in the far East, including the Malay Peninsula (gold and tin), Siam, Tonkin and Bengal. Upon his return to Europe he devoted himself to consulting work and the management of an iron mine in Spain. In 1895 he was appointed professor of mining in the Durham College of Science, Newcastle-upon-Tyne, and in 1896 received from the University of Durham the honorary degree of M.A. Mr. Louis is the

author of a *Handbook of Gold Milling* (1894), a revised and enlarged edition of Phillips' *Treatise on Ore Deposits* (1896), second edition, 1899; translations of Schnabel's *Handbook of Metallurgy* (1898), and Schmeisser's *Gold Field of Australia* (1898). He contributes to this volume the notes on the metallurgy of tin.

MEYER, FRANZ, was born in Loitz, Germany, in 1865, and studied for five years at the University of Berlin and Greifswald, receiving from the latter institution, in 1891, the degree of Ph.D. For three years after graduation the analytical and technical laboratory of the Actien Gesellschaft für Zink Industrie at Hamborn-am-Rhein, was under his charge, and from 1895 until 1897 he was superintendent with the Chemische Fabriken at Herdingen-am-Rhein. In 1897 Dr. Meyer came to the United States to superintend the construction of the works of the Somerset Chemical Co., at Bound Brook, N. J., and was its manager until March 1, 1900, when he resigned to accept the position of manager of the chemical department with the Actien Gesellschaft für Zink Industrie. Dr. Meyer has contributed to the present volume the paper on "Liquid Anhydrous Sulphurous Acid."

NICHOLSON, FRANK, was born in Dallas, Texas, in 1860. He graduated, in 1880, from the School of Mines, Washington University, St. Louis, in mining engineering, and in 1883 received the degree of Master of Science from that institution. Mr. Nicholson has been identified in various capacities with numerous mining companies in the West, and in Mexico, and during 1899 he was consulting engineer for ten of the principal producing companies in the Joplin district of southwestern Missouri. Mr. Nicholson has done a large amount of expert work, covering properties in the United States, British Columbia and Mexico, and is a member of several engineering and technical societies. For the present volume he has contributed notes on the mining and milling of zinc ores in the Galena-Joplin district.

OBALSKI, J., was born in France in 1852, and studied at the Ecole des Mines at Paris; after graduation he occupied several positions in connection with the mining industry in France and Spain, and visited various mining districts in those countries and Belgium and Portugal. In 1881 he was called by the government of the Province of Quebec to fill the position of mining engineer and inspector of mines, which post he still occupies. He has contributed to the present volume the notes on asbestos, chrome ore, copper ore, and mica mining in Quebec.

RICHARDS, ROBERT HALLOWELL, professor of mining and metallurgy at the Massachusetts Institute of Technology, was born in 1844 at Gardiner, Me. He graduated in 1868 from the Massachusetts Institute, being a member of its first class, and became assistant in chemistry in the corps of instruction, passing successively to the posts of instructor, assistant professor of chemistry, professor of mineralogy and assaying, professor of mining engineering, and in 1884 to his present professorship of mining and metallurgy. Under his administration the mining and metallurgical laboratory, which was the first of its kind in an educational institution, has been developed to a high degree of excellence, and has been a model for similar laboratories in other colleges. In addition to his professional duties, Prof. Richards has been actively engaged as a consulting engineer in mining and metallurgical work, and has been the inventor of several ingenious devices, which have found extended use in practice. He has contributed many valuable papers to the technical press, and to the transactions of various scientific societies. His most recent work on the principles of ore dressing is the admiration of the entire body of engineers engaged in that branch of metallurgy. To the several volumes of THE MINERAL INDUSTRY he has contributed the comprehensive reviews of "Progress in Ore Dressing," and in addition, he has given to the present volume the summary of his forthcoming elaborate treatise on "Ore Dressing."

RIX, WILTON P., has been identified with the Doulton Art Potteries at Lambeth from their establishment, 30 years ago, until 1898, and during his long and satisfactory connection with that company, he made many important researches on the manufacture of colored glazes which would be sufficiently durable to withstand the heat of the Lambeth kilns. During his experimental work he produced many new articles of ceramic ware,

that brought great credit to the company. His extensive knowledge of clays and other raw materials for the great variety of pottery and china manufactures has especially qualified him for the expert and research work to which he is now devoting his attention. Mr. Rix contributes to the present volume the paper on "Notes on the Coarse Pottery Production of Great Britain."

SANDERS, WILBUR E., studied at the School of Mines, Columbia College (now Columbia University), graduating as a mining engineer in 1885. With the view of obtaining practical experience, he then spent several years in the mines of Butte, Mont., and in general milling, amalgamating, concentrating and smelting practice. He also prospected for several seasons in the mountains of Montana. Mr. Sanders has filled various positions with mining companies in the West, and has managed several properties in that section, and in the zinc regions of Missouri. Mr. Sanders has reported on many mines throughout the West, and is engaged at present as a consulting mining engineer, with headquarters at Butte, Mont. For the present volume he has contributed the article on "Mine Timbering."

SAUVEUR, ALBERT, studied four years at the Ecole des Mines at Liége, Belgium, and then at the Massachusetts Institute of Technology, where he was graduated in 1889. After one year in the chemical laboratory of the Pennsylvania Steel Co., at Steelton, Pa., he became connected with the South Chicago works of the Illinois Steel Co., with which he remained six years, most of the time in charge of the experimental laboratory. Since February, 1897, he has been proprietor and manager of the Boston Testing Laboratories at Boston, and is now lecturer at Harvard University, and at the Massachusetts Institute of Technology, and editor of *The Metallographist*. Mr. Sauveur is a member of several scientific societies, and has been a contributor to numerous technical publications. He has a world-wide reputation as an expert in his specialty, the investigation of the microstructure of metals, and its bearing upon their technology. Mr. Sauveur contributes to this volume a paper upon that subject.

SLOANE, T. O'CONOR, was born in 1851. He graduated from St. Francis Xavier's College, and later, in 1872, from the School of Mines, Columbia College (now Columbia University), in mining engineering. He subsequently obtained the degree of Ph.D. from the latter institution. Mr. Sloane served as assistant to the U. S. Geological Survey through the Yellowstone National Park in 1872, and later was chemist to the New York Gas Light Co. While in this latter position he did considerable work on the analysis of gas and coal, and published numerous papers, in scientific journals, on these subjects. In 1877 he became chief engineer of the Citizen's Gas Light Co., of Brooklyn, N. Y. From 1885 until 1897 he was on the editorial staff of *The Scientific American*, and in recent years he has devoted himself to expert work in patent cases. Mr. Sloane has written many works on popular subjects, and has contributed largely to scientific and technical publications. For the present volume he has written the paper on "Liquid Air."

SOUTHER, HENRY, was born in Boston, Mass., in 1865, and received his early education in the public schools of that city. He graduated from the Massachusetts Institute of Technology, in 1887, in mining and metallurgy, and spent the following year at Clausthal, Germany, returning to the United States in 1888, to accept a position in the Bessemer and open-hearth departments of the Pennsylvania Steel Co., at Steelton, Pa. In 1893 he entered the employ of the Pope Mfg. Co., at Hartford, and built the laboratory for chemical and physical tests, resigning in 1899 to establish himself as a consulting engineer at Hartford, Conn. Mr. Souther has contributed to the present volume the paper on "Nickel and Iron Alloys."

STRUTHERS, JOSEPH, was born in New York City, and attended the School of Mines, Columbia College (now Columbia University), graduating therefrom in chemistry in 1885, and later, in 1895, he received the degree of Ph.D. from that institution. For three years after his graduation he was assistant in the department of mineralogy at Columbia University, and from thence until 1890 he devoted his time to metallurgy, being on the staff of instructors of the department of metallurgy, first assisting Dr. Thomas Egleston, and later Prof. Henry M. Howe. He has recently been appointed honorary lecturer in

metallurgy at Columbia University. In 1896 he organized and conducted the first summer school in practical metallurgy of Columbia University, which was held at Butte, Mont., and, in the following year, the summer school at Chicago and Pittsburg was under his charge. In 1898 and 1899 he assisted at the summer schools held in Pennsylvania and Maryland. Aside from the duties attendant on instruction work, he has devoted the greater part of his summers to visiting metallurgical plants in the United States and in Europe, and with other professional work, he has carried on scientific investigations in calorimity, measurement of furnace temperatures, microscopical examination of metals and slags, and work of like character. Dr. Struthers has contributed numerous articles to the *School of Mines Quarterly*, and from 1892 to the present time has been on its Board of Editors, acting for most of this period as business manager. As assistant editor of THE MINERAL INDUSTRY, Vol. VIII., he has had charge of its preparation, and has contributed many unsigned articles.

ULKE, TITUS, was born in 1866, at Washington, D. C. In 1889 he graduated from the Royal School of Mines at Freiberg, Saxony, as metallurgical engineer. After spending some time in visiting the various mines and metallurgical works in Europe, Mr. Ulke returned to this country and was engaged as chemist to the Harney Peak Tin Co., in South Dakota. In 1891 he became assayer for the United Smelting Co., and afterward was engaged by the Anaconda Mining Co., as chemist at its electrolytic copper refining works. In 1893 Mr. Ulke acted as metallurgist to the Mines and Mining Department of the Chicago Columbian Fair, and was later employed at the Guggenheim works at Perth Amboy, N. J. As triangulator for the U. S. Geological Survey in 1897, he had charge of a party to survey the Montana timber reserves, from which he returned to continue work in his specialty—the electrolytic production of copper, nickel, gold and silver. Soon after the declaration of war with Spain, Mr. Ulke was appointed Assistant Inspector of Ordnance, U. S. A., which position he has just resigned to become chemical engineer to the Lake Superior Power Co., Sault Ste. Marie, Ont. Mr. Ulke has contributed the article on "Progress in the Electrolytic Refining of Copper during 1899" to the present volume.

WEIGHTMAN, ALFRED T., was born near London, Eng. He received his training at the City Guilds Central Technical College, and was honor medalist in electro-chemistry. After graduating he took charge of the electrolytic department, with H. H. Vivian & Co., Swansea, Wales, and was subsequently appointed chief electro-chemist to the Chloride Electrical Storage Syndicate, Manchester, Eng. He resigned this position for the present post of electro-chemist of the Lake Superior Power Co., Sault Ste. Marie, Ontario, Can. Mr. Weightman has contributed the paper on the "Electrolytic Production of Chlorine and Caustic Soda" for the present volume.

WEINBERG, E. A., is a mining engineer and metallurgist of wide and varied experience. Educated in Germany, he took up his residence in San Francisco, and later entered into the employment of the Jocuistita Mining Co., of Mexico, becoming eventually its general manager. On the closing down of these mines, Mr. Weinberg became metallurgist of the Anaconda Mining Co., leaving this company to conduct the smelters of the Deadwood & Delaware Co., at Deadwood, S. D. His Australian experience has been extensive, for many years he was general manager of the Queensland Smelting Co,. Ltd., at Aldershot, Queensland, and in 1898 became general manager of the Smelting Company of Australia, at Illawarra, N. S. W. Mr. Weinberg has contributed to the present volume notes on copper and nickel mining in New Caledonia.

WHITTLE, CHARLES L., studied at the Lawrence Scientific School of Harvard University, and upon completion of his course spent six years with the Archæan and Atlantic coast divisions of the U. S. Geological Survey, working mainly in Vermont and western Massachusetts. After the disruption of the Archæan division of the Survey, in 1892, he gave a course in assaying at Harvard University, and assisted in teaching mineralogy and petrography. Mr. Whittle has devoted much time to economic geology, and since leaving Harvard has been established professionally at Boston as a geologist and mining engineer. He has contributed to this volume, the article on "Stone," a subject in which he is well known as an expert.

INTRODUCTION.

THE total value at the place of production of the mineral and metal output of the United States in 1899 was $1,211,361,861, as against $861,751,017 in 1898. These amounts are quite unexampled in the history of the mineral industry.

Of this vast sum ores and minerals contributed $580,836,032 in 1899 and $440,997,150 in 1898; metals, $496,057,320 in 1899 and $312,650,857 in 1898; secondary products $63,996,969 in 1899 and $49,095,010 in 1898; while the value of metals smelted or refined from foreign material was $70,471,540 in 1899 and $59,008,000 in 1898. In these gross totals of value are included certain duplications, such as those of the manganese and iron ore used in making ferromanganese and pig iron; bauxite used in making aluminum and alum; coal used in making coke; lead used in making white and red lead and litharge and a few other duplications, the whole amounting in 1899 to $92,581,031, and in 1898 to $62,232,984. The values of the crude foreign ores or metals smelted or refined here should also be deducted in order to arrive at the net value of the mineral industry of the United States.

In the preparation of the statistics for this volume, the figures previously reported for 1898 have been revised in the light of later and more minute investigation, in accordance with our practice, wherefore it is important for students to observe the caution to use always the figures in the latest volume of THE MINERAL INDUSTRY. There are no statistical reports of this nature which are absolutely correct, owing to the practical impossibility of obtaining accurate reports from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers which prevents them from making returns, the unwillingness of a few to give their figures, and confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our own statistics, but also in the statistics reported by various governments. In our own work, however, we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation.

We have made great use of the reports of several State geological surveys,

especially those of Alabama, Kansas, Iowa and Montana, and the State mining bureaus of California and Colorado. We have generally credited these figures to the proper sources in the subsequent pages, but this acknowledgment may stand for any unintentional oversights.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES. (FIRST PRODUCTS.)

Number.	Products.	Customary Measures.	1898. Quantity. Customary Measures.	1898. Quantity. Metric Tons.	1898. Value at Place of Production. a Totals.	1898. Value at Place of Production. a Per M. Ton.	1899. Quantity. Customary Measures.	1899. Quantity. Metric Tons.	1899. Value at Place of Production. a Totals.	1899. Value at Place of Production. a Per M. Ton.
					$	$			$	$
1	Asbestos	Sh. T.	885	803	13,425	16·72	912	827	13,960	16·76
2	Asphaltum	Sh. T.	25,690	23,306	482,175	20·69	15,060	13,662	308,130	22·55
3	Asphaltic limestone	Sh. T.	14,099	12,791	70,495	5·51	10,378	9,415	54,422	5·78
4	Bitumin's sandstone	Sh. T.	49,536	44,938	146,275	3 26	42,104	40,646	123,229	3·03
5	Barytes	Sh. T.	28,247	25,626	112,988	4·41	32,636	29,607	137,071	4·63
6	Bauxite	L. T.	26,791	27,220	66,978	2·46	36,813	37,402	101,235	2·71
7	Borate of calcium. c	Sh. T.	15,335	13,911	306,700	n22·05	24,068	21,834	505,428	n23·15
8	Bromine	Lb.	486,978	221	196,354	617·19	433,008	196	125,571	640·67
9	Cement, nat. hydraul.	g Bbls	8,161,078	1,110,552	3,819,995	3·44	10,186,447	1,386,163	5,183,500	3·74
10	Cement, Portland	h Bbls	3,584,586	650,383	6,168,106	9·48	5,805,620	1,053,365	10,441,431	9·91
11	Chrome ore	L. T.	100	102	1,000	9·84	100	102	1,000	9·84
12	Clay products				59,379,874				74,064,628	
13	Coal, anthracite	Sh. T.	52,848,605	47,943,940	81,445,937	1·70	60,577,398	54,955,455	103,648,780	1·89
14	Coal, bituminous	Sh. T.	165,435,389	150,082,000	129,039,154	0·86	191,501,350	173,728,885	172,406,679	0·99
15	Coal, cannel	Sh. T.	49,899	45,259	134,700	2·98	36,639	33,239	91,597	2·76
16	Cobalt oxide	Lb.	9,640	k 4,373	15,424	k 3·53	10,200	k 4,627	15,810	k 3·42
17	Copper sulphate. f	Lb.	55,119,361	25,002	m 1,879,570	75·18	67,908,370	30,801	m 3,580,975	114·64
18	Corundum	Sh. T.	786	713	63,630	89·24	970	880	78,570	89·28
19	Emery	Sh. T.	2,956	2,682	143,800	53·62	3,000	2,722	150,000	55·11
20	Feldspar	L. T.	21,350	21,692	107,147	4·94	26,968	27,399	137,866	5·03
21	Fluorspar	Sh. T.	12,145	11,018	86,985	7·89	24,030	21,800	152,655	7·00
22	Fullers earth	Sh. T.	15,553	14,110	87,365	6·19	13,620	12,356	81,900	6·63
23	Garnet	Sh. T.	2,882	2,615	82,930	31·71	2,565	2,327	72,672	31·23
24	Grahamite	Sh. T.	2,675	2,427	80,250	n33·07	3,150	2,858	97,650	n34·17
25	Graphite, crystalline	Lb.	1,647,679	k 747,382	82,385	k 0·11	3,632,608	k 1,647,740	145,304	k 0·09
26	Graphite, amorphous	Sh. T.	1,200	1,089	11,400	10·47	1,030	934	8,240	8·82
27	Gypsum	Sh. T.	314,865	285,644	864,415	3·03	372,285	337,735	1,109,286	3·28
28	Iron ore	L. T.	20,655,865	20,986,359	37,593,674	1·79	25,841,000	25,746,456	58,284,300	2·26
29	Lepidolite	Sh. T.	5	5	181	36·20	124	112	4,600	41·07
30	Magnesite. c	Sh. T.	2,355	2,136	9,420	4·41	2,000	1,814	7,600	4·19
31	Manganese ore. i	L. T.	187,782	190,787	416,627	2·18	143,256	145,548	306,476	2·10
32	Mica, scrap	Sh. T.	3,529	3,201	o 39,837	12·44	6,917	6,275	o 50,956	8·12
33	Mica, sheet	Lb.	110,918	k 50,317	92,232	k 1·83	97,586	k 44,265	76,926	k 1·74
34	Molybdenite	Sh. T.	8	7	400	57·14	18	16	900	56·25
35	Monazite	Lb.	150,000	68	7,500	110·23	330,000	150	18,480	123·20
36	Natural gas				e 10,500,000				e 11,500,000	
37	Ocher. p	Sh. T.	41,950	38,057	461,450	n12·13	42,286	38,362	465,146	n12·13
38	Petroleum, crude	y Bbls	55,499,875	7,764,713	44,542,906	5·74	57,234,304	8,007,368	64,143,890	8·01
39	Phosphate rock	L. T.	1,257,645	1,277,767	4,355,025	3·41	1,633,391	1,852,565	7,031,785	3·80
40	Precious stones				e 125,000				e 175,000	
41	Pyrites	L. T.	191,160	194,219	589,329	3·03	178,408	181,263	583,323	3·22
42	Salt. q	Bbls.	18,756,394	2,382,197	4,753,664	2·00	19,861,948	2,522,610	5,437,941	2·16
43	Silica, brick	M.	33,449		590,254		45,000		850,000	
44	Diatom. earth	Sh. T.	1,392	1,263	11,002	8·71	1,196	1,031	8,216	7·27
45	Flint	Sh. T.	20,593	18,682	47,333	2·53	24,805	22,049	56,481	2·56
46	Sand, etc	L. T.	e 800,000	812,800	e 1,178,560	1·45	e 850,000	863,600	e 1,338,580	1·55
47	Pumice	Sh. T.	144	131	720	n 5·51	340	308	1,700	n 5·52
48	Grindstones	Sh. T.	38,859	35,253	488,675	12·44	43,959	39,879	501,985	12·59
49	Whetstones				107,990				115,220	
50	Tripoli	Sh. T.	1,865	1,692	3,367	1·99	2,102	1,907	4,729	2·48
51	Slate, roofing	Sq'es.	1,041,532	314,888	2,777,996	j 2·67	1,098,374	332,146	3,055,988	j 2·78
52	Manufactures				528,856				540,434	
53	Pigment	Sh. T.	7,886	7,154	70,671	9·88	4,911	4,455	56,338	11·80
54	Soapstone	Sh. T.	18,862	17,111	158,635	9·27	20,011	18,153	189,504	10·44
55	Soda, natural. r	Sh. T.	7,150	6,486	94,650	14·59	10,500	9,526	152,250	15·98
56	Stone, for building				e 35,404,900				e 33,797,487	
57	Stone, limestone (flux)	L. T.	5,275,819	5,360,232	2,304,900	0·43	6,707,435	6,814,754	3,475,525	0·51
58	Stone, lithographic	Sh. T.	112	102	2,240	n 21·96	40	36	2,000	55·56
59	Strontium sulphate	Sh. T.	2	2	10	n 5·55	*Nil.*	*Nil.*	*Nil.*	*Nil.*
60	Sulphur	L. T.	2,726	2,770	59,754	m 21·57	1,565	1,590	33,585	m 21·12
61	Sulphuric acid. f. l	Sh. T.	47,558	43,144	970,183	m 22·49	59,572	54,043	1,338,456	m 24·59
62	Talc, common	Sh. T.	9,112	8,266	78,645	9·51	6,671	6,052	51,763	8·55
63	Talc, fibrous	Sh. T.	54,807	49,721	285,759	5·75	57,120	51,819	272,595	5·26
64	Tungsten ore	L. T.	78·865	80·13	17,398	217·12	168	171	n 33,600	196·49
65	Uranium ore	Sh. T.	38	30	16,500	550·00	43	39	21,500	551·28
66	Wulfenite	Sh. T.	12	11	144	13·09	(b)	(b)	(b)	(b)
67	Zinc sulphate	Sh. T.	145	132	4,640	35·15	528	479	21,648	n 45·19
68	Zinc ore, exported	Sh. T.	11,782	10,688	299,870	28·06	28,220	25,601	725,944	28·36
69	Zinc white. s	Sh. T.	32,747	29,708	2 226,796	74·96	39,663	35,982	3,331,692	92·04
70	Est. prod. unspecified				5,000,000				5,000,000	
	Totals				440,997,150				580,886,039	

PRODUCTION OF METALS IN THE UNITED STATES.

Number.	Products.	Customary Measures.	1898. Quantity. Custom'y Measures.	1898. Quantity. Metric Tons.	1898. Value at Place of Production. a Totals.	1898. Value at Place of Production. a Per M. Ton.	1899. Quantity. Custom'ry Measures.	1899. Quantity. Metric Tons.	1899. Value at Place of Production. a Totals.	1899. Value at Place of Production. a Per M. Ton.
					$	$			$	$
71	Aluminum	Lb.	5,200,000	*k*2,358,705	*m*1,690,000	*k*0·72	6,500,000	*k*2,948,881	*m*2,112,500	*k*0·72
72	Antimony	Lb.	2,000,000	907	165,000	181·92	2,500,000	1,137	241,250	212·18
73	Copper	Lb.	535,900,292	243,083	*t*63,129,047	259·70	581,319,091	263,685	100,916,994	*t*382·72
74	Ferromang'nese.*u*	L. T.	218,769	217,189	*v*10,474,681	48·23	219,768	223,284	*v*18,350,626	82·19
75	Ferromolybden'm	Lb.	2,100	.....	1,050	1,102·30	6,000	2·7	3,000	1,102·30
76	Gold	Oz.*w*.	2,148,642	*k*97,988	65,082,490	*k*664·60	3,391,196	*k*105,471	76,096,021	*k*664·60
77	Iron, pig	L. T.	11,560,165	11,745,128	*v*110,168,372	9·38	13,400,735	13,615,350	*v*234,725,754	17·24
78	Iridium	Oz.*w*.	8·5	.....	255	.....	5·6	.....	165	.....
79	Lead	Sh. T.	228,475	207,271	*m*17,272,710	83·33	217,085	196,938	*m*19,407,399	98·55
80	Molybdenum	Lb.	9,550	*k*4,332	11,937	*k*2·76	80,000	*k*13,608	37,500	*k*2·76
81	Nickel	Lb.	11,145	*k*5,055·3	*m*3,845	*k*0·76	22,500	*k*10,205·9	*m*8,156	*k*0·80
82	Platinum	Oz.*w*.	300	*k*9·3	3,687	*k*411·25	*Nil.*	*Nil.*	*Nil.*	*Nil.*
83	Quicksilver.*x*	Fl'sks	30,493	1,058	1,109,945	1,049·09	28,879	993	1,155,160	1,163·30
84	Silver	Oz.	56,755,032	*k*1,765,265	38,065,482	*mk*18·73	57,126,834	1,776,829	34,636,168	*mk*19·16
85	Tungsten	Lb.	33,200	*k*15,059	43,160	*k*2·87	*e*45,000	*k*20,412	*n*54,000	*k*2·87
86	Zinc	Sh. T.	114,104	103,514	10,429,106	*m*100·75	129,675	117,644	14,912,625	*m*126·76
	Totals				312,650,857				496,057,320	

SECONDARY MINERAL AND CHEMICAL PRODUCTS OF THE UNITED STATES.

Number.	Products.	Customary Measures.	1898. Quantity. Custom'ry Measures.	1898. Quantity. Metric Tons.	1898. Value at Place of Production. a Totals.	1898. Value at Place of Production. a Per M. Ton.	1899. Quantity. Customary Measures.	1899. Quantity. Metric Tons.	1899. Value at Place of Production. a Totals.	1899. Value at Place of Production. a Per M. Ton.
					$	$			$	$
87	Alum	Sh. T.	18,791	17,047	563,730	*m*33·07	27,276	24,745	845,556	*m*34·17
88	Aluminum sulphate	Sh. T.	56,663	51,404	1,416,575	*m*27·56	81,805	74,213	2,106,479	*m*28·38
89	Ammonium sulphate	Sh. T.	3,589	3,256	179,450	*m*55·11	(*b*)	(*b*)	(*b*)	(*b*)
90	Carborundum	Lb.	1,504,152	728	151,444	209·47	1,741,245	790	150,712	198 37
91	Cement, slag	ABbls	157,662	28,606	235,721	8·24	244,757	44,408	360,800	8·10
92	Coke	Sh. T.	15,938,556	14,459,363	31,109,680	2·15	18,025,256	16,352,405	42,081,082	2·57
93	Copperas	Sh. T.	11,295	10,238	58,105	5·68	13,770	12,492	108,508	8·69
94	Crushed steel	Sh. T.	330	299	46,200	154·53	337	306	47,250	154·41
95	Graphite, artificial	Lb.	185,647	*k*84,209	11,608	*k*0·14	405,870	*k*18,410	32,475	*k*0.18
96	Lead, white. *z*	Sh. T.	98,172	84,525	9,391,738	111·11	103,466	93,864	10,812,197	115·20
97	Lead, red	Sh. T.	9,160	8,310	916,000	110·23	10,199	9,252	1,070,895	115·07
98	Lead, orange miner'l	Sh. T.	653	592	88,937	150·23	998	842	139,200	165·32
99	Litharge	Sh. T.	7,460	6,768	710,192	104·93	10,020	9,090	103,206	113·53
100	Mineral wool	Sh. T.	6,560	5,951	70,314	11·82	7,448	6,757	85,899	12·72
101	Soda, manufact'ed. *r*	M. T.	.....	340,022	4,080,651	11·98	.....	387,020	5,925,276	15·31
102	Venetian red	Sh. T.	3,270	2,967	64,570	21·76	6,575	5,965	121,519	20·37
	Totals				49,095,010				63,996,909	

METALS PRODUCED FROM FOREIGN ORES AND BULLION. (*aa*)

Metals.	Customary Measures.	1898. Quantities. Customary Measures.	1898. Quantities. Kg.	1898. Values.	1899. Quantities. Customary Measures.	1899. Quantities. Kg.	1899. Values.
				$			$
Copper	Pounds.	36,055,352	16,354,600	4,247,320	40,659,868	18,443,195	7,058,533
Gold	Troy oz.	1,065,552	33,142	22,024,960	1,423,439	44,274	29,422,691
Lead	Short tons	89,209	80,929,874	6,744,200	76,423	69,330,491	6,832,216
Nickel	Pounds.	7,127,784	3,233,142	2,459,085	8,048,343	3,650,708	2,917,525
Silver	Troy oz.	40,392,095	1,256,325	23,532,435	40,542,858	1,261,014	24,240,575
Total values				59,008,000			70,471,540
Total ores and minerals				$440,99?,150			$580,826,089
Total metals				312,650,857			496,057,320
Total secondary products				49,095,010			63,996,909
Grand total, values				$861,751,017			$1,211,361,861

In using the statistics in the foregoing tables reference should also be made to the detailed tables under the respective captions further on in this volume, where many explanatory notes as to the statistics will be found. The following notes refer to the four preceding tables: (*a*) Except where otherwise specified. (*b*) Not enumerated. (*c*) Amount or value of crude mineral. (*d*) Partly estimated. (*e*) Estimated. (*f*) Amount recovered as a by-product. (*g*) Barrels of 300 lb. (*h*) Barrels of 400 lb. (*i*) Includes manganiferous iron ore; this is not duplicated in the report of iron ore. (*j*) Value per square, *i.e.*, 100 sq. ft., lapped and laid; the weights are calculated on the basis 3 squares = 2,000 lb., but these figures are only approximately correct. (*k*)

Kilograms or per kilogram. (*l*) Reduced to a basis of 66° B. (*m*) Average market price at New York. (*n*) Nominal. (*o*) Value before grinding. (*p*) Includes ocher, umber, sienna, and oxide of iron. (*q*) Includes salt used for the manufacture of alkali; the barrel of salt weighs 280 lb. (*r*) Reduced to a basis of 58% ash. (*s*) Includes a small amount made from spelter. (*t*) Average value of Lake copper at New York, less 0·25c. per lb. (*u*) Includes spiegeleisen, though the total value is reckoned as if the whole product were ferromanganese. (*v*) Average market price at Pittsburg. (*w*) Troy oz. (*x*) Flasks of 76·5 lb. (*y*) Barrels of 42 gal. (*z*) Includes a comparatively small amount made directly from ores. (*aa*) Not included in the preceding tables.
Abbreviations: Sh. T., short tons (2,000 lb.); L. T., long tons (2,240 lb.); M. T., metric tons (2,204·6 lb).

METALS AND ALLOYS.

Aluminum.—Production of aluminum in the United States in 1899 was 6,500,000 lb. ($2,112,000) against 5,200,000 lb. ($1,690,000) in 1898. The increased price of copper during 1899 caused an increased consumption of aluminum wire for electrical conductors.

Antimony.—The production of metallic antimony in 1899 was 2,500,000 lb. (241,250) against 2,000,000 lb. ($165,000) in 1898.

Copper.—The production increased from 535,900,232 lb. ($63,129,047) to 581,319,091 lb. ($100,916,994). The main increase was in Arizona. Montana showed a slight increase, while Lake Superior fell off somewhat.

Ferromanganese.—The production of ferromanganese, including spiegeleisen was 219,768 long tons ($18,350,628) against 213,769 long tons ($10,474,681) in 1898.

Ferromolybdenum.—The production in 1899 was 6,000 lb. ($3,000) against 2,100 lb. ($1,050) in 1898.

Gold and Silver.—The domestic production of gold in 1899 was 3,391,196 troy oz. ($70,096,021) against 3,148,642 troy oz. ($65,082,430) in 1898. The production of silver was 57,126,834 troy oz. ($34,036,168) against 56,755,032 troy oz. ($33,065,482). Colorado's production of gold increased, principally due to Cripple Creek. South Dakota and Alaska, including Cape Nome and the American Yukon also contributed to the increased production. California showed an unexpected falling off in the production of both gold and silver. Besides the production reported, 1,423,349 oz. of gold and 40,542,858 oz. of silver were smelted in the United States from imported ores. The average value of silver in the United States in 1899 was 59·58c. per oz. against 58·26c. per oz. in 1898.

Iron.—The production of pig iron in 1899, exclusive of ferromanganese and spiegeleisen was 13,400,735 long tons ($234,725,754) against 11,560,165 long tons ($110,168,732) in 1898. Of the production in 1899, 8,202,778 long tons were Bessemer pig against 7,337,384 long tons in 1898. Basic pig increased from 785,444 to 985,033 long tons. The remainder of the output is classed as foundry and forge iron.

Lead.—The domestic production in 1899 was 217,085 short tons ($19,407,399) against 228,475 short tons ($17,272,710) in 1898. The greater portion of the falling off was due to suspension of operations in Idaho owing to labor difficulties. The average price of lead at New York was 4·47c. against 3·78c. per lb. in 1898. Besides the above, the American smelters in 1899 recovered 76,423 tons of lead from foreign ore and base bullion, against 89,209 tons in 1898.

Molybdenum.—Some 30,000 lb. of molybdenum ($37,500) were produced in 1899, against 9,550 lb. ($11,937) in 1898. A considerable portion of the ore came from Arizona.

Nickel.—The domestic production of nickel was 22,500 lb. ($8,156) against 11,145 lb. ($3,845) in 1898.

Platinum.—There was no production of platinum in 1899 against 300 troy oz. ($3,837) in 1898. The value of bar platinum at New York varied from $15 to $18 per oz. against an average of $12·79 in 1898.

Quicksilver.—The production of quicksilver decreased to 28,879 flasks ($1,155,160) from 30,493 flasks ($1,109,945) in 1898. The increased price of quicksilver has caused the reopening of many old California mines, and it is probable that 1900 will show a marked increase in production.

Tungsten.—Forty-five thousand lb. ($54,000) were produced in 1899 against 33,200 lb. ($43,160) in 1898. The exact statistics have not been available for the production of ferrotungsten.

Zinc.—The production in 1899 was 129,675 short tons ($14,912,625) against 114,104 short tons ($10,429,106) in 1898. Missouri and Kansas increased their production, while the average price of spelter in New York in 1899 increased to 00c. per lb. against 4·57c. per lb. in 1898.

ORES, MINERALS AND CHEMICAL PRODUCTS.

Alum and Aluminum Sulphate.—The production of crystallized alum in the United States in 1899 was 27,276 short tons ($845,556) against 18,791 ($563,730) in 1898. The production of aluminum sulphate in 1899 was 81,805 short tons ($2,106,479) against 56,665 ($1,416,675) in 1898.

Ammonium Sulphate.—Statistics for the amount of ammonium sulphate recovered by coke works in 1899 are not available at the time of publication, but the production in 1898 was 3,589 short tons. A larger amount of this substance is obtained from the ammoniacal liquor of illuminating gas works, but there are no statistics as to that production. The value of sulphate, basis 25%, was $58·76 per short ton at New York in 1899, against $50 in 1898.

Asbestos.—The domestic production was 912 short tons ($13,860) in 1899, against 885 short tons ($13,425) in 1898. In each year the production was made almost entirely by one mine in Georgia.

Asphaltum and Asphaltum Products.—The production of asphaltum, liquid and solid, in 1899 was 15,060 short tons ($308,130), against 25,690 ($482,175) in 1898, the entire output each year having been made in California. California and Kentucky, chiefly the former, produced 42,104 short tons ($123,229) of bituminous rock in 1899, against 49,536 ($146,275) in 1898. Utah and the Indian Territory produced 10,378 short tons ($54,422) of asphaltic limestone, against 14,099 ($70,495) in 1898. The output of Utah increased in 1899 and that of the Indian Territory decreased somewhat. The production of grahamite or gilsonite in 1899 was 3,150 short tons, against 2,675 in 1898, the product in 1899 being valued nominally at $31 (in 1898 at $30) per ton at the mines. Colorado produced 150 tons in 1899, the remainder being derived from Utah.

Barytes.—The production in 1899 was 32,636 short tons ($137,071) against 26,791 ($66,978) in 1898. Of the production in 1899 Missouri furnished

17,836 tons, the remainder being obtained in Virginia, North Carolina and Tennessee.

Bauxite.—The production in 1899 was 36,813 long tons ($101,235) against 26,791 ($66,978) in 1897. Of the production in 1899 Georgia furnished 19,619 tons, the remainder being mined in Alabama.

Borate of Calcium.—The production in 1899 was 24,068 short tons, against 15,335 in 1898. Most of this product is colemanite, mined in California.

Bromine.—The production in 1899 was 433,003 lb. ($125,571), against 486,978 ($136,354) in 1898. Of the production in 1899 liquid bromine constituted 307,003 lb., against 356,978 in 1898, the remainder of the output being the bromine equivalent of potassium bromide produced in Michigan.

Calcium Carbide.—The production in 1897 was 1,925 short tons ($134,750). It was probably larger in 1898, and still more so in 1899, but statistics for these years could not be obtained.

Carborundum.—The production reported by the sole producer was 1,741,245 lb. ($156,712) in 1899, against 1,594,152 lb. ($151,444) in 1898. The decline in value was due to the increased proportion of the cheaper grades marketed.

Cement.—The total production of Portland cement in 1899 was 5,805,620 bbl. of 400 lb., valued at $10,441,431, against 3,584,586 ($6,168,106) in 1898. The Lehigh district of Pennsylvania and New Jersey has maintained its supremacy as a center of production, and Michigan, Ohio and other States also showed important gains. There was remarkable prosperity in this industry on account of the increased demand and higher prices. The production of natural rock cement in 1899 was 10,186,447 bbl. of 300 lb., valued at $5,183,500, against 8,161,078 ($3,819,995) in 1898. The increase in the production of natural rock cement was due chiefly to the Indiana-Kentucky district, where the remarkably low prices brought about by excessive competition among the producers undoubtedly stimulated consumption. Other parts of the United States showed an increase, owing to the large amount of construction work in 1899. The production of slag cement in 1898 was 244,751 bbl. of 400 lb., valued at $360,800, against 157,662 ($235,721) in 1898.

Chrome Ore.—The production in 1899 was 100 long tons ($1,000) an amount the same as in the previous year. Chrome mining in California ceased in 1898 and the entire output was derived from Pennsylvania and Maryland.

Clay.—The value of brick and other clay products made in the United States in 1899 was $74,064,628, against $59,379,874 in the previous year.

Coal and Coke.—The total production of coal in the United States in 1899 was 252,115,387 short tons ($276,147,056) against 218,333,833 ($210,619,791) in 1898. The production of anthracite, all of it from Pennsylvania with the exception of an insignificant amount from Colorado, was 60,577,398 short tons ($103,648,780) in 1899, against 52,848,605 ($81,445,937) in 1898. Kentucky produced 36,639 short tons ($91,597) of cannel coal against 49,889 ($134,700) in 1898. The remainder of the output each year was bituminous coal, of which Pennsylvania and the Central States are the largest producers, their relative rank in 1899 having been Pennsylvania, Illinois, West Virginia

and Ohio in the order named. There was an increase in the output of all of the important coal-producing States in 1899. The total production of coke in 1899 was 18,025,256 short tons ($42,081,002) against 15,938,556 ($31,109,680) in 1898 Pennsylvania furnished about two-thirds of the output each year.

Coal Tar.—No statistics were compiled of the production of coal tar by by-product coke oven plants in 1899. In 1898 they turned out 16,021 short tons and 13,304 in 1897. There are no statistics as to the large production of this substance by illuminating gas works.

Cobalt Oxide.—The production of cobalt oxide in 1899 was 10,200 lb., against 9,640 in 1898. The value was $1·55 per lb. in 1899, against $1·60 in 1898.

Copperas.—The production in 1899 was 13,770 short tons ($108,508) against 11,285 ($58,105) in 1898. The chief producer in this country is the American Steel & Wire Co., which controls all the wire and rod mills recovering copperas as a by-product. The above statistics do not include copperas converted into Venetian and Indian reds at the works of original production.

Copper Sulphate.—The production in 1899 was 67,903,370 lb., against 55,119,361 in 1898. Of this the amount recovered as a by-product, chiefly by gold and silver refiners, was 37,285,870 lb. in 1899 and 28,061,501 in 1898. The remainder of the output each year was made from metallic copper previously reported in the production of that metal. The average value of copper sulphate at New York per 100 lb. was $5·20 in 1899, against $3·41 in 1898.

Corundum and Emery.—The production of corundum in 1899, all of it from North Carolina and Georgia, was 970 short tons ($78,570) against 786 ($63,630) in 1898. The production of emery, most of it from Massachusetts, was 3,000 short tons ($150,000) against 2,956 ($143,800) in 1898. The resources of the United States in both corundum and emery of the highest quality are large and an increasing production is to be looked for. The production of steel emery or crushed steel in 1899 was 674,000 lb. ($47,250) against 660,000 ($46,200) in 1898, the entire make each year being supplied by the Pittsburg Crushed Steel Co.

Feldspar.—The production in 1899 was 26,968 long tons ($137,866) against 21,350 ($107,147) in 1898. Pennsylvania being the chief producer each year.

Fluorspar.—The production in 1899 was 24,030 short tons ($152,655) against 12,145 ($86,985) in 1898, Illinois and Kentucky furnishing the entire output.

Fullers Earth.—The output in 1899 was 13,620 short tons ($81,900) against 15,553 ($87,365) in 1898. The most part each year was mined in the vicinity of Quincy, Fla.

Garnet.—The production in 1899 was 2,565 short tons ($72,672) against 2,882 ($82,930) in 1898, the output each year being furnished by New York, Pennsylvania and Connecticut. The domestic resources of this mineral are large, but the demand for it is limited.

Graphite.—The production of crystalline graphite in 1899 was 3,632,608 lb. ($145,304) against 1,647,679 ($82,385) in 1898. The change in the valuation of the 1898 product from $148,291 (as reported in Vol. VII. of THE MINERAL INDUSTRY) to $82,385 is due to the change in valuation from New York prices

to prices at the mine. The proportionately large increase in value was due to the increased demand for crucibles for the manufacture of projectiles during the Spanish war. The larger part of the product in 1899 was obtained from Ticonderoga, N. Y., but a considerable quantity was mined in Pennsylvania, where some of the old mines were reopened. The production of amorphous graphite in 1899 was 1,030 tons ($8,240) against 1,200 tons ($11,400) in the previous year. One company produced 405,870 lb. ($32,475) of artificial graphite against 185,647 ($11,603) in 1898.

Gypsum.—The production increased from 314,865 short tons in 1898 to 372,285 in 1899. These figures represent the amount of crude rock quarried. The more part of the production is marketed as stucco or plaster of Paris.

Iron Ore.—The production in 1899 was 25,341,000 long tons against 20,655,865 in 1898, these figures being exclusive of the production of manganiferous iron ore, reported separately under manganese. The increase in the production was due chiefly to the Lake Superior ranges, the Southern States showing only a comparatively small gain.

Lead White, Red Lead and Litharge.—The productionof white lead in 1899 was 103,466 short tons ($10,812,197) against 93,172 ($9,391,738); of red lead 10,199 ($1,070,895) against 9,160 ($916,000); of litharge 10,020 ($103,206) against 7,460 ($710,192); of orange mineral 928 ($139,200) against 653 ($88,937) in 1898. The most of these products is obtained by the corrosion of pig lead, but a small part of the white lead product is made directly from ores.

Limestone for Iron Flux.—Iron smelters consumed 6,707,435 long tons in 1899 against 5,275,819 in 1898, the increase being caused by the greater make of pig iron.

Lithographic Stone.—Utah produced 40 short tons in 1899, against 112 in 1898. There was no output in previous years.

Magnesite.—California produced 2,000 short tons ($7,600) in 1899 against 2,355 ($9,420) in the previous year.

Manganese Ore.—The production of manganese ore, including manganiferous iron ore, was 143,256 long tons in 1899 against 187,782 in 1898. Included in these figures are 53,762 tons of manganiferous iron ore from Michigan and Wisconsin in 1899, against 112,318 in 1898, and 53,921 tons of franklinite residuum from New Jersey in 1899, against 47,470 in 1898.

Mica.—The production of sheet mica in 1899 was 97,586 lb. ($76,926) against 110,918 ($92,232) in 1898. The production of scrap mica was 6,917 short tons ($50,956) against 3,529 ($39,837), the values of scrap mica being reckoned for the product before grinding. Practically the entire production of mica is made in South Dakota, New Hampshire and North Carolina.

Mineral Wool.—The production in 1899 was 7,448 short tons ($85,899) against 6,560 ($70,314) in 1898. A part of this product was made from slag and a part by the fusion of natural rock, the latter being the more valuable.

Molybdenite.—This mineral was produced in the United States in 1899 to the amount of 18 short tons, valued nominally at $50 per ton, against 8 short tons of the same value per ton in 1898.

Monazite.—North Carolina produced 330,000 lb. ($18,480) in 1899 against

150,000 ($7,500) in 1898. Notwithstanding the increase no important future is to be looked for in this industry on account of the vast supplies of cheaper mineral available in Brazil.

Ocher and Oxide of Iron Pigments.—The production of ocher, umber, sienna and natural oxide of iron ground pigment, the last being known commonly as "metallic paint," was 42,286 short tons ($465,146) in 1899 against 41,950 ($461,450) in 1898. The values are reckoned nominally at $11 per ton each year. These products merge into one another so that no satisfactory division of the statistics is possible. Pennsylvania is the largest producer.

Petroleum.—The total output in 1899 was 57,234,304 bbl. ($64,143,890) against 55,499,879 ($44,542,906) in 1898. The increase was due to the Appalachian and the Lima fields. California and Texas made increased outputs, while Colorado and Kansas showed a decrease. The above values are calculated for crude oil at the wells. The average was considerably higher in 1899 than in the previous year.

Phosphate Rock.—The production in 1899 was 1,823,391 long tons ($7,031,-785) against 1,257,645 ($4,355,025) in 1898. The increase was due especially to Tennesee and the deposits of Florida and South Carolina. Phosphate mining was generally prosperous in 1899 on account of the higher prcies.

Salt.—The domestic output of salt increased from 18,756,394 bbl. in 1898 to 19,861,948 in 1899, Kansas and New York each contributing largely to the increase.

Silica.—The production of vein and dike quartz in 1899 was 24,305 short tons ($56,481) against 20,593 ($47,333) in 1898. This material was used chiefly in pottery, for packing acid towers, and for grinding for various purposes. The production of grindstones, which are made out of quartzite, or a very hard sandstone, amounted to 43,959 short tons ($501,985) in 1899 against 38,859 ($438,675) in 1898. These were produced entirely in Ohio and Michigan. The production of oilstones, scythestones and whetstones in 1899 was valued at $115,220 against $107,990 in 1898. Utah produced 340 short tons of pumice stone, valued nominally at $1,700, against 144 ($720) in 1898. There was a production of 2,102 short tons of tripoli, valued at $4,729, against 1,865 ($3,367) in 1898, most of this product being obtained in Missouri. The production of diatomaceous earth in the United States in 1899 was 1,136 short tons ($8,216) against 1,392 ($11,002) in the previous year.

Slate.—The production of roofing slate in 1899 was 1,098,374 squares ($3,055,988) against 1,041,632 ($2,777,996) in 1898. The production of slate manufactures, chiefly blackboards and structural material, was valued at $540,434 in 1899, against $528,856 in 1898. The production of slate pigment, including Baraga graphite and various kinds of mineral black, was 4,911 short tons, against 7,886 in the previous year, the values of the products after grinding being respectively $50,338 and $70,671.

Soda.—The production of soda and soda products from salt, reduced to a common basis of 58% soda ash, was 387,020 metric tons in 1899, against 340,-622 in 1898. The average value of 58% ash at the works was $15·31 per metric ton in 1899 against $11·98 in 1898. California and Nevada produced 10,500

short tons ($152,250) of natural soda, basis 58% in 1899, against 7,150 short tons ($94,650) in 1898.

Strontium Sulphate.—There was no production of strontium sulphate in 1899, against 2 tons produced in Ohio in 1898 for experimental purposes.

Sulphur and Pyrites.—Louisiana, Nevada and Utah produced 1,565 long tons of sulphur in 1899 against 2,726 in 1898. The average price of Sicilian seconds at New York was $21·46 per ton, against $21·92 in the previous year. The domestic production of pyrites in 1899 was 178,408 long tons worth on an average of $3·27 per ton, against 191,160, averaging $3·08 per ton in 1898. Virginia was the largest producer each year, Massachusetts ranking second. Sulphuric acid was recovered as a by-product in roasting blende and pyrites, chiefly the former, to the extent of 59,572 short tons against 47,558 in 1898, the figures being reduced to a common basis of 66° B. Concentrated acid of the latter strength averaged $22·25 per 2,000 lb. at New York in 1899 against $21·40 in 1898.

Talc and Soapstone.—The production of soapstone for slabs and other manufactured articles in 1899 was 20,011 short tons ($189,504), against 18,862 ($158,635) in 1898. The increase in value was due to general conditions of prosperity. The production of common talc, mostly ground to powder, was 6,671 short tons ($51,763), against 9,112 ($78,645) in 1898. This was produced chiefly in North Carolina, Pennsylvania and Vermont. The production of fibrous talc, all of it from St. Lawrence County, N. Y., was 57,120 short tons ($272,595), against 54,807 ($285,759) in 1898.

Tungsten Ore.—There was a production in 1899 of 168 long tons of tungsten ore from South Dakota, Colorado, Arizona and New Mexico with a small amount from Connecticut, which was valued nominally at $33,600 at the mines. The statistics for 1898 were approximately 79 long tons, valued at $17,398.

Uranium Ore.—Colorado produced 43 short tons, valued at $551·28 per ton, in 1899, against 33 short tons, valued at $500 per ton, in 1898.

Venetian Red.—The production of Venetian and Indian reds in 1899 was 6,575 short tons ($121,519), against 3,270 ($64,570) in 1898. These figures include only the output at works where the original copperas was made, and do not include any that may have been made by second hands.

Wulfenite.—Arizona shipped 12 short tons of this mineral, valued at $12 per ton, in 1898. There was no output reported in 1899.

Zinc Ore.—The amount of zinc ore of domestic origin exported from the United States in 1899 was 28,220 short tons ($725,944), against 11,782 ($299,870) in the previous year. Most of the ore exported was mined in New Jersey.

Zinc Sulphate.—There was a production in 1899, by one concern, of 528 short tons, against 145 in 1898.

Zinc White.—The production in 1899 was 39,663 short tons, against 32,747 in 1898. The total value was $3,331,692, against $2,226,796 in 1898. Nearly all the American production of zinc white is made directly from ores, most of it from ores mined in New Jersey.

ALUMINUM AND ALUM.

UNDER this general caption are grouped aluminum, alum, bauxite, cryolite, corundum and emery, substances which, previous to Volume VII., appeared under individual captions. The present arrangement is a logical one, since both bauxite and cryolite are used as raw material in the manufacture of aluminum and alum, two industries so interwoven that a logical separation is impossible, and, since corundum, though employed mainly as an abrasive, is now used in part as a source of aluminum. In this view aluminum, alum and the aluminum minerals —bauxite and cryolite, corundum and emery—bear the same relation to one another as do copper, copper sulphate or bluestone and copper ores.

I. BAUXITE.

The only new producing district in America in 1899 was in Arkansas. The Georgia production increased somewhat, although not to a large extent. In Alabama the industry was in a satisfactory condition, although it was clearly demonstrated in 1899 that this region was not capable of supplying the demand. The entire production of the United States was 36,813 long tons ($101,235) against 26,791 tons ($66,978) in the preceding year.

Arkansas.—The bauxite occurs in a belt about 6 miles long and about 3 miles wide. It is exposed to view in many cases, but is mostly capped by white sandy loam averaging about 3 ft. in thickness. Underneath this the bauxite beds average 16 to 18 ft. thick. As the operation of stripping these is simple and labor costs only $1 per day, the conditions are unusually good for cheap mining. The ore varies from a creamy white to a brownish red in color, the latter discoloration being due to the presence of iron. Bands of limonite are occasionally encountered in the ore as well as ferruginous boulders. The occurrence is associated with igneous rocks, syenite and granite being encountered through the entire length of the deposit, and granite being found at the eastern extremity of the beds.

PRODUCTION, IMPORTS, EXPORTS, AND CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.					Imports.		Exports.		Consumption.	
	Alabama.	Georgia.	Total. (a)								
	Lg. Tons.	Lg. Tons.	Lg. Tons.	Value.	Per Ton.	Lg. Tons.	Value.	Lg. Tons.	Value.	Lg. Tons.	Value.
1895..	14,000	4,800	18,800	$56,400	$3·00	5,797	$34,782	.....	.....	$34,597	$91,182
1896..	9,796	7,300	17,096	42,740	2·50	2,119	10,477	.....	.....	19,215	53,217
1897..	13,083	7,507	20,590	51,475	2·50	2,645	10,515	$2,537	$5,074	20,708	56,916
1898..	13,848	12,943	26,791	66,978	2·50	1,201	4,238	1,000	2,000	26,992	69,216
1899..	14,144	19,619	36,813	101,235	2 75	6,666	23,768	2,030	4,567	41,449	12,436

(a) Arkansas produced 3,050 long tons in 1899, which is included in the total for the year.

The largest shipper in Arkansas is the Pittsburg Reduction Co., which shipped 1,720 tons, all of which it used in its own works. Mr. E. Metzenauer shipped about 900 tons; the disposition of which is not stated. The Illinois Chemical Co. consumed 400 tons, while the Southern Bauxite Mining Co. shipped some 30 tons. Prices have been unchanged; even the better methods of mining, drying and handling have not increased them. The lack of railway communication and freighting facilities has prevented several mines from being productive. The Little Rock & Hot Springs Western Railway was constructed in the early part of the year and passes through the bauxite region. It will doubtless assist in marketing much ore during the coming year. The Republic Mining and Manufacturing Co. has bought considerable bauxite land, and is preparing to extract ore, having abandoned its Alabama deposits, owing to the depth which they have reached. The Pittsburg Reduction Co. is preparing to work its mines on a much more extensive scale. The total production of Arkansas amounted to 3,050 tons.

The Arkansas Alumina Works were established during the year for experimental purposes. Outside of its experimentation the only work done was to dry ore for the market. The company reports that it will probably start the manufacture of aluminum hydrate during this year.

Georgia.—The industry in Georgia was of a very satisfactory condition during 1899, the production having increased to a considerable degree in the last two months. The prices also have slightly advanced. All the mines were worked to their full capacity. A combination of the five producers is spoken of, though as yet it has not been accomplished. Foreign shipments have decreased since the high rates prevalent at the time of the Spanish-American war. The Republic Mining and Milling Co. shipped both dried ore and ore as mined. All the product went to the American market. The Watters mine became a producer during the year; it is said to contain a large amount of high grade, low silica ore.

PRODUCTION OF BAUXITE IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
France	23,398	33,923	26,082	17,958	33,820	41,740	36,723
United Kingdom	7,489	8,880	8,097	10,574	7,365	13,449	12,600
United States	9,957	11,204	10,904	19,101	17,370	20,919	27,220
Total	40,694	54,007	45,083	47,633	58,555	76,108	76,543

II. Corundum and Emery.

The following notes of the corundum industry of North Carolina have been furnished by Leverett S. Ropes, of Murphy, N. C.: "Though the production for 1899 showed little improvement over the preceding year the industry is now in better condition than heretofore. A substantial gain may be expected for 1900, as new machinery and new properties will come into operation. Lack of water and the failure of the concentrating machinery to give satisfactory results, caused the shut down of the Isbel Corundum Co.'s plant. This plant shows a radical departure from the normal practice in washing corundum, though it was not built without knowledge of the results obtained by the old mill of the

Shooting Creek Corundum Co. This machinery consisted of a double two compartment Harz jig, a 5-ft. Frue vanner and two screens, which took the corundum, quartz, chalcedony and other hard material and fully separated them from the gangue (decomposed amphibolite and kaolinized albite) which was carried off in the overflow of water.

As this scheme was not rigidly followed in the construction of the new mill

PRODUCTION OF CORUNDUM AND EMERY IN THE UNITED STATES. (*a*)

Substance.	1896.			1897.			1898.			1899.		
	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.	Tons. (*b*)	Value.	Per Ton.
Corundum	250	$17,000	$68·00	293	$19,810	$67·61	786	$63,630	$80·96	970	$78,570	$81·00
Emery	1,850	*c* 89,000	48·11	1,900	*c* 92,000	48·42	2,956	*c* 143,800	48·65	3,000	150,000	50·00
Steel emery	326	50,624	155·29	324	51,824	159·95	330	46,200	140·00	337	47,250	140·21
Total	2,426	$156,624		2,517	$163,634		4,072	$253,630		4,237	$275,820	

(*a*) The statistics of production given in the above table have been changed considerably from previous volumes of THE MINERAL INDUSTRY owing to more precise returns, not only for 1898, but also for previous years. The change in the valuation of the products is due to basing this upon the price at the mine instead of at New York, as has been the case previously. However, in this industry the value figures are of slight significance owing to the great range between the different grades of these minerals. Consequently the difference in the annual average does not indicate change in market quotations so much as it does change in the proportion of different grades of mineral in the total. (*b*) 2,000 lb. (*c*) More or less nominal.

IMPORTS OF EMERY INTO THE UNITED STATES.

Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.	Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.
	Pounds	Value.	Long Tons.	Value.	Value.			Pounds	Value.	Long Tons.	Value.	Value.	
1892.	566,448	$22,586	5,280	$95,625	$2,412	$120,623	1896..	751,464	$26,520	6,289	$119,667	$1,971	$148,158
1893.	516,953	20,073	5,066	103,875	3,819	127,767	1897..	520,095	20,022	5,209	107,649	2,211	129,882
1894..	597,713	18,645	2,804	51,487	1,847	71,978	1898.	577,655	23,320	5,547	106,269	3,810	133,399
1895..	678,761	25,066	6,803	80,396	27,586	133,088	1899.	728,299	29,124	7,435	116,493	11,514	157,131

of the Isbel Co. it may be necessary to rearrange the jigs and vanners and to place the spinners as usual between the crushers and concentrates.

The mill structure is 72 ft. in height, and 120 ft. long, built substantially but on low ground, the rock being raised from the storage bins by a skip to the crushers at the top of the mill. The crushing machinery consists of a set of heavy geared Allis rolls, faced with teeth which reduce the material as it comes from the mines to about three-quarters of an inch. The amphibolite is not entirely decomposed and the unaltered rock is exceedingly tough, necessitating very heavy machinery for otherwise light work. The crushed ore passes to five trommels from 0·75 to 0·062-in. mesh, and the pulp from the last to four hydraulic classifiers. The rejections from the first screen are returned to a set of belt-driven smooth rolls and returned to the screens. Another set of smooth rolls is located on the same floor and receives the coarse middlings from the two-compartment jigs.

For concentrating 2 double two-compartment and 4 double three-compartment Harz jigs, 5 five-ft. and 4 six-ft. Frue vanners are used. The concentrates from the jigs are not rewashed further but are passed to the spinners, when they are scoured, this being accomplished by the particles of corundum acting upon each other. The overflow carries off the comminuted material, and

after a certain length of time the corundum is drawn off through gates at the bottom of the inverted iron cone in which the spinner acts. Drying, recrushing and grading follows. The vanners receive this pulp from the settling tanks receiving the overflow from the classifiers. It seems remarkable that there should be any corundum in this material, but quite a little shows in some of the concentrates from the vanners. The mill has a capacity of 60 cu. yd. per day of 24 hours. The test runs were not carried far enough, nor were the runs made under favorable enough conditions, to ascertain the percentage that the mill will save.

In December negotiations were closed for the purchase of a large area of land on the Blue Ridge corundum belt. Work on cabins and roads commenced at once.

These deposits might be called corundiferous mica-schist, and form a zone in the mica-gneiss formation of the Blue Ridge and Nantyhole Mountains in this section, now and then bounded with dense black hornblende, or again with the more feldspathic garnet bearing rocks of the same, and through all of which are thrown up the great bosses of peridotite and idumite rocks; with this is associated corundum and almost innumerable alterations.

The corundum mica-schist does not show the magnesian rocks, nor the other common associates of corundum when it occurs in peridotite rocks. Garnets are found in a portion of the corundum deposits, but are confined generally to the edge of the vein, where the writer noticed a 2 to 6 in. band carrying 20 to 30% garnet and an equal amount of corundum. These garnets, unlike the corundum, are fractured and brittle, so that by crushing and sizing the garnets are carried into the finer sizes, leaving the corundum very free and pure. In testing this material the writer has used vein rock, which yielded 42% corundum, 12% of which was in "nuggets" (crystals) too large to pass through a 0·5-in. mesh. Large lots tested have yielded 95%.

A peculiar feature of the occurrence is the freedom with which the crystals separate from the gangue; this is in part at least due to a film, at times occurring as thin but perceptible scales. The gangue when crushed is chiefly mica scales and a thin pulp of very fine feldspar, which seems to form the ground mass of the schist, which breaks up finely. Thus we have an ideal combination for crushing concentration and cleaning. The specific gravity of the corundum being much higher than that of the associated minerals and the peculiar susceptibility of the latter to the flotation make the crushers, rolls and jigs an ideal and economical method. I do not think the corundum will need scouring after concentration, as the crystals have no stain and seldom any wrapping, while all gangue material is thrown off in breaking down. The crystals are especially lustrous, extremely dense and probably possess cutting qualities surpassed by no commercial corundum, and by an occasional gem only. No gems are found; the corundum is uniform in character and quality. The formation is 3 miles long.

The deposit dips to the north at an angle varying from 30 to 50°. Its general strike is east and west, at times its elevation carrying it across the Georgian State line. It seems quite probable that a line of north and south faulting just west of the Foster mine has thrown that somewhat farther south, where it occurs

in the south side of the line at an elevation of about 2,500 ft., while west at the Bly Gap it is probably 3,500 ft., and on Scaley Mountain, where the outcrop is exposed on the east, south and west sides, it shows across the face of almost perpendicular cliffs at an elevation of 4,500 or 5,000 ft. above sea level, and hundreds of feet above the creek below, where huge blocks of vein stuff lie. The vein here reaches a thickness of 12 to 14 ft. at places, and a mill can be readily supplied with ore mining 10 or 15%. This assurance of an abundant and regular supply has been seldom afforded by the usually pockety formation of the peridotites.

Prospecting has been done along the Buck Creek-Truck rock belt with more or less promising results, but lack of capital, a fear of the past and present tangled titles has greatly retarded development. The mistake of placing the property in the hands of inefficient and inexperienced men has been frequently made. Corundum mining, like other industries, requires special knowledge and special training of high character. Other developments nearer railroads are taking place which in another year will probably influence the market, but at the end of the year the industry should be in such shape that consumers will know what to count upon, and prices should have settled to a definite proportion as dictated by cost of operations.

The National Abrasive Manufacturing Co., of Philadelphia, have purchased 100 acres containing one-half mile on the vein just west of the Foster property and will mine there; it is stated that the Isbel Corundum Co. is to resume operation at its mill on Shooting Creek.

Corundum in Ontario, and Abrasive Tests.

By Courtenay De Kalb.

The production of corundum from the deposits in Ontario will soon be an assured fact under the terms of a charter granted to the Canada Corundum Co., of Toronto. The directors of the company are J. N. Shenstone, B. A. C. Craig, and Lloyd Harris, all of Toronto. By agreement with the Provincial Government the corporation is obliged to expend $100,000 before July 1, 1902, in the acquisition of properties, development of mines, and erection of plant, an expenditure of not less than $25,000 being required for the latter purpose. They further undertake to expend $1,000 per year for three years in experiments upon the utilization of corundum for other than abrasive purposes, special attention to be given to the production of aluminum from this mineral. The company has purchased the Lobillard-Fitzgerald property in Raglan Township, Renfrew County, Ontario, and is privileged to locate 2,000 acres of corundum-bearing Government lands, and two water powers on the York branch of Madawaska River, in Dungannon Township. The initial operations will be on the Lobillard-Fitzgerald property, having slack-water navigation from this point in summer to Barry's Bay station on the Ottawa, Arnprior, and Perry Sound Railroad.

Corundum in Ontario was first found about 15 years ago by Herbert Wood on the Armstrong farm in Carlow Township, but the mineral was not identified until 1896, when specimens fell into the hands of W. F. Ferrier, of the Canadian

wears down, thus exposing fresh faces continually to the metal which is undergoing abrasion.

Comparative tests of Hastings corundum and North Carolina corundum as well as carborundum were made by the Hart Emery Wheel Co., of Hamilton, Ontario. Brass, cast iron, wrought iron and cast steel were subjected to the abrasive action of wheels composed of the several materials.

These tests illustrated forcibly that the adaptability of any particular abrasive varies with the kind of material to be abraded, and that the relative efficiency cannot be predicted from the absolute hardness or purity of the abrasive. The purer Hastings corundum gives better results on brass than the relatively impure commercial product from the North Carolina mines. Improved methods of preparation of the North Carolina corundum would undoubtedly increase its efficiency. Carborundum, however, is apparently superior with both brass and wrought iron, is equal with cast iron, but fails with cast steel. Tests of this character at emery wheel works everywhere show a like variation, so that no class of abrasive can claim to be universally superior to all others. It is a question of the kind of work to be done, but for general purposes it may be said that a wheel made of the purest corundum will be most satisfactory.

The cost of mining and milling corundum rock, carrying 15% of corundum, is estimated at less than 3c. per lb. for concentrates averaging higher than 80% pure. This would apply to either Canada or the Southern States, when operating on a large scale with improved modern machinery for crushing and concentrating.

III. Cryolite.

The importations of cryolite into the United States as in previous years, came from Greenland, and are given in the subjoined table. These shipments were made by the Pennsylvania Salt Manufacturing Co., of Natrona, Pa., which possesses the exclusive privilege to import this mineral into North and South America. The remainder of the output of the Ivigtut mines is shipped to Copenhagen.

IMPORTS OF CRYOLITE INTO THE UNITED STATES. (a)

Year.	Long Tons.	Value.	Year.	Long Tons.	Value.	Year.	Long Tons.	Value.	Year.	Long Tons.	Value.
1888..	7,898	$98,830	1891..	8,298	$76,350	1894..	10,684	$142,494	1897..	10,115	$135,114
1889..	8,603	115,158	1892..	7,341	96,932	1895..	9,425	125,368	1898..	6,201	88,501
1890..	7,129	95,405	1893..	9,574	126,688	1896..	3,009	40,056	1899..	5,879	78,676

(a) The values are those reported by the Custom House.

According to a report furnished by the Danish Government the total production in Greenland in 1896 was 6,058 metric tons, and in 1897, 13,150 tons.

The price of cryolite in the American market during the first quarter of 1899 was 8·50c. per lb.; later it dropped to 6·75c., and closed the year at 6·50c. per lb.

IV. Aluminum.

The increase in the production of aluminum in the United States—so noticeable in 1898—continued in 1899; the total production amounted to 6,500,000 lb. The greater part of the increase in consumption was due to the use of aluminum for electrical conductors and to the utilization of the metal to replace zinc and brass for many purposes.

PRODUCTION, IMPORTS, AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES.

Year.	Production.			Imports.	Exports.	Consumption.
	Pounds.	Value.	Per Lb.	Value.	Value.	Value.
1894	817,600	$490,560	$0.60	$4,120	(a)	$494,680
1895	900,000	495,000	0.55	10,301	(a)	505,301
1896	1,300,000	520,000	0.40	3,419	(a)	523,419
1897	4,000,000	1,400,000	0.35	4,731	(a)	1,404,731
1898	5,200,000	1,690,000	0.325	4,879	$239,997	1,454,882
1899	6,500,000	2,112,500	0·325	14,840	291,515	1,835,825

(a) The exports from the United States were not reported by the Bureau of Statistics in these years.

The statistics of aluminum production in Europe are not authoritative, several of the important companies being unwilling to make their figures public. The *Metallgesellschaft,* of Frankfurt-am-Main, gives the following statistics for Europe in its last annual report, to which we have added our own figures for the United States, and those of C. Le Neve Foster for England in 1898.

ALUMINUM: WORLD'S PRODUCTION AND COMMERCE. (IN KILOGRAMS.)

Year.	Germany.	Switzerland. (a)		England.	France.			United States.(b)		Total Production.
	Imports.	Production.	Exports.	Production.	Production.	Imports.	Exports.	Production.	Imports.	
1890	.....	40,538	137,200	70,000	37,000	109	187	97,797	930	165,335
1891	.....	168,669	201,900	52,500	36,000	593	130	76,238	1,779	233,407
1892	.....	237,395	277,900	41,000	75,000	2,007	42,287	133,811	20	487,206
1893	.....	437,476	404,100	.....	137,000	1,524	33,367	141,523	3,545	715,998
1894	.....	600,000	520,200	.....	270,000	2,958	30,787	370,860	2,405	1,240,860
1895	.....	650,000	490,900	.....	360,000	3,806	109,954	408,237	11,473	1,418,237
1896	591,500	700,000	651,100	130,000	370,000	7,012	192,763	589,676	317	1,789,676
1897	942,400	800,000	706,000	c310,000	470,000	6,300	224,000	1,814,388	825	3,394,388
1898	1,104,000	800,000	687,500	310,000	565,000	6,000	188,000	2,358,705	29	4,033,705

(a) The statistics of exports from Switzerland in 1890 and 1891 include aluminum alloys and manufactures of aluminum. (b) The United States has been an exporter of aluminum for several years, but these exportations were not enumerated by the Bureau of Statistics of the Treasury Department until 1898, in which year they amounted to $239,997. (c) C. Le Neve Foster, British Mineral Statistics for 1897.

PROGRESS IN THE ALUMINUM INDUSTRY IN 1899.

By JOHN B. C. KERSHAW.

Production.—One addition has been made to the number of works producing aluminum during 1899, namely, the Rheinfelden works of the Aluminum Industrie Actien Gesellschaft. The total number of producing works now stands at seven; details of these are given in Table I. The project to establish aluminum works in Norway is still unrealized; and there is no doubt that the capacity of the existing producing works in Europe is much in excess of the present demand for the metal. Two, if not three, of the existing works are devoting portions of their available supply of electrical energy to the production of calcium carbide, and on this account it is difficult to estimate the present output of aluminum in Europe.

The new plant at Rheinfelden is stated to have cost the Neuhausen Co. $558,000; and the charge for electrical energy is stated to be $28·80 per e.h.p. year for continuous use, but possibly the electro-chemical and electro-metallurgical factories pay rather less than this. The latter take the current from six continuous current dynamos running at 55 revolutions per minute, direct

coupled to the turbines, and each giving a current of 5,160 ampères at 120 volts pressure.

I.—DETAILS OF ALUMINUM WORKS IN AMERICA AND EUROPE.

Name of Company.	Locality of Works.	Horse Power.		Process.	Capital.
		Available.	In Use.		
The Pittsburg Reduction Co.	Niagara, No. 1.	.....	3,050	Hall	$1,600,000
The Pittsburg Reduction Co.	Niagara, No. 2.	.....	4,000	Hall	
The British Aluminium Co.	Foyers	14,060	3,000	Herault	2,880,000
Société Electrometallurgique Française	Le Praz	6,000	5,000	Herault	2,880,000
Société Industrielle d'Aluminum	St. Michel	6,000	2,000	Minet & Hall	364,800
Aluminum Industrie Aktien Gesellschaft	Neuhausen	4,000	4,000	Herault	3,077,000
Aluminum Industrie Aktien Gesellschaft	Rheinfelden	5,000	.....	Herault	

As regards the output of aluminum, the last complete statistics relate to the year 1898. The Pittsburg Reduction Co. again heads the list of producing companies with 2,355 metric tons, out of a total production of 3,958 metric tons. During 1899, this company increased their production to 2·948,8 metric tons. The figures for the production of aluminum each year since 1889, are given in the table of the world's production and commerce set forth in the preceding pages. The leading aluminum companies have considerably increased their capital during 1899, and important developments are now occurring in the industry.

The Pittsburg Reduction Co. has not only increased its capital, but has doubled the plant and output of its No. 2 Niagara works; and it is said to be financially interested in developments at Shawinigan Falls, in Canada. The rolling mills and wire drawing works at Niagara have also been considerably enlarged.

The British Aluminium Co. has raised £300,000 by bond issue during 1899. The greater portion of this was required to pay off existing mortgages and loans, but £80,000 will rank as new capital and will be available for new developments.

The Neuhausen Co. has increased its capital from 11,000,000 to 16,000,000 fr. during the year, in order to carry out the development of water power and erection of works at Rheinfelden, and at Lend-Gastein in Austria.

This company and the Foyers Co., have, however, taken up the manufacture of calcium carbide, and a portion of the new capital in each case may be directed to the production of this substance, rather than to that of aluminum; for at the present prices ruling in Europe, calcium carbide is the more profitable manufacture.

The Société Electrometallurgique Française has also increased its capital from 9,000,000 to 15,000,000 fr. during the year, and has completed very considerable additions to the works at Le Praz. Whether a portion of the new plant here is also to be diverted to the production of carbide, the writer, is unable to say.

As regards the dividends declared during 1899, the Neuhausen Co., in 1898, made a gross profit of 1,631,526 fr., as compared with 1,131,142 fr. in 1897, and has paid a 12% dividend on the ordinary shares. The British Aluminium

Co. made a gross profit of £28,581, and has paid 7% on the preference capital. The dividends of the Pittsburg Co., and of the two French companies, have not been ascertained.

The selling price of aluminum has undergone little alteration during 1899, and it is evident that by the present methods of production but slight further reduction is possible. The reported sales of large quantities of aluminum in bar, rod or wire form, for electrical conduction purposes, at the low price of 29c. per lb. *at the point of consumption* have most probably resulted in a loss to the manufacturers; but the policy of encouraging in this manner the trial and use of aluminum by electrical engineers, is, no doubt, a wise one, and at some future date it may be found to have repaid the present losses.

No new process for the industrial production of aluminum has been brought forward during the year; but the Peniakoff process is still receiving experimental trial at Selzaete in Belgium. This process depends upon the use of a bath containing a fused mixture of aluminum fluoride or chloride, and sodium sulphide. Into this bath aluminum sulphate and carbon are fed. The aluminum sulphide produced by the action of the carbon on the sulphate, dissolves in the molten bath, and is decomposed by the current. The advantage of the process is the low E.M.F. required to affect the decomposition of the sulphide; the disadvantage is the necessity for using aluminum sulphate, in place of the oxide, for feeding the baths. The writer doubts whether the saving in cost of energy will balance the greater cost of raw materials. The Blackmore process, which has recently been tried in New York,* seeks to attain the same ends by a different method; but this process also has not yet passed out of its experimental stage. The only two processes in actual use are, therefore, the Hall and the Herault processes. (See Table I.) But few facts are allowed to leak out concerning the actual details of the electrolytic decomposition of the aluminum oxide by either process; but Dr. Richards has recently stated† that the heat of the Herault process is so intense that the carbon acts as a reducing agent upon the aluminum oxide, and a portion of the yield of aluminum is thus obtained by chemical action. The heat is due to an envelope of carbonic acid gas which surrounds the anode, across which the current has to leap by means of an arc. Dr. Richards does not state in what works he witnessed this effect. If it be a usual feature of the working of the Herault bath, one would say that the loss in electrical energy—due to the high E.M.F. necessary to strike and maintain the arc—would more than balance the gain in aluminum due to the chemical reduction by carbon. In the same article Dr. Richards states that a bath 36×60 in. containing a depth of 6 in. of the fused aluminium and sodium salts, with electrodes 2 in. apart, was maintained at a temperature of 900° C. by the passage of the current.

Dr. Borchers, in a paper read at the sixth annual meeting of the German Electro-Chemical Society,‡ gives some interesting calculations relating to the aluminum manufacture. Taking the horse power in use at the different works, and the annual output of aluminum, he calculates that the Pittsburg Reduction

* THE MINERAL INDUSTRY, Vol. VII., p. 26. † *Aluminum World*, March, 1899.
‡ *Zeitschrift für Elektrochemie*, Vol. VI., pp. 61-76.

Co. work with an E.M.F. per bath of 4 volts; the Foyers Co. with an E.M.F. of between 4 and 5 volts; and the remaining companies with an E.M.F. of between 5 and 10 volts. As pointed out above, two of these companies, and possibly one other, are using an unknown portion of their available water power for carbide production; and calculations based on the assumption that it is all devoted to the manufacture of aluminum are therefore untrustworthy.

The litigation between the Cowles and the Hall companies in the United States, with respect to the Bradley patents, reached a further stage in the autumn of 1899, but at the date of writing this article, the judgment in this very protracted case had not been received.

During 1899 the Pittsburg Reduction Co. suffered a severe loss by the death of Capt. Alfred E. Hunt, its president. Capt. Hunt had contributed largely to the technological literature of aluminum, and his efforts in this direction had been of great service in bringing the new metal before the public.* The position occupied by the Pittsburg Reduction Co. to-day is largely due to his work. He is succeeded by Mr. R. B. Mellon. An appreciation of Capt. Hunt will be found in the issue of the *Aluminum World* for May, 1899.

UTILIZATION.—*Electrical Conductors.*—The chief feature of 1899, as regards the utilization of aluminum, has been the increased attention given to it as a substitute for copper in power transmission schemes. The extraordinary rise in the price of copper—from £57 10s., in December, 1898, to a maximum of £77, in April, 1899—and the maintenance of this high level with but slight variations during the remainder of the year, has led to a restricted use of the metal in electrical work, and to a readiness to try substitutes of lower cost. The offer of the Pittsburg Reduction Co. and of the Allgemeine Elektricitats Gesellshaft to supply aluminum in wire or rod form suitable for conducting purposes—at the low price of 29c. (the Berlin Co. asks 30c.) per lb., has therefore led to the trial of aluminum in a very large number of instances. A list of the schemes in which aluminum is already in use, or is about to be tried as an electrical conductor, is given in Table III.

In addition to the companies enumerated in Table III., the Brooklyn Avenue Street Railway Co., of Kansas City, the Waxahachie Electric Lighting Co., Texas, and the Northallerton Electric Lighting Co., England, are using aluminum for distributing their current; while at the Chicago stock yards it has been in use for a considerable period for telegraph and telephone purposes. In New York experimental telephone wires of aluminum have stood two winters' exposure without failure; and the General Post Office authorities in England are reported to have several aluminum lines on trial for long-distance telephone work.

The composition of the aluminum used for conducting purposes appears to vary considerably. The Snoqualmie Falls transmission line is made of an alloy containing 98·5% aluminum and 1·5% copper, while the Niagara Falls cables contain 99·5% aluminum. Northrup has made relative conductivity tests† with alloys made by the Pittsburg Reduction Co. specially for conducting

* THE MINERAL INDUSTRY, Vol. VI., contains a summary of one of Captain Hunt's articles on aluminum.
† *Electrical World*, Dec. 3, 1898.

purposes, which contain 0·5 and 0·75% copper. The addition of copper is made to increase the tensile strength of the aluminum, but its presence diminishes the conductivity; and a drop of 5% is shown by Northrup's tests to be manifested when the copper in the alloy has reached 0·75%.

III.—ELECTRIC POWER AND LIGHTING SCHEMES, IN WHICH ALUMINUM IS USED FOR CONDUCTING PURPOSES.

Name of Company.	Locality.	Weight of Al'min'm in Lbs.	Length of Cable in Miles.	Remarks.
Niagara Falls Hydraulic Power Co.	Niagara Falls.....	.....	.....	This company has two aluminum cables in use.
Snoqualmie Falls Power Co..	Seattle-Tacoma ..	22,000 72,000 67,000	⅞ 84	Transmission started Aug. 1, 1899, @ 29,000 volts. 10,000 h.p. is eventually to be transmitted.
Blue Lakes Power Co.....	Stockton.....	.....	43	1,000 k.w. is transmitted @ 25,000 volts.
Ragged Rapids Power Co.....	Severn River, Ont..	.....	18	2,000 h.p. is transmitted.
North Yuba Power Co.....	Sacramento.....	.....	65	1,000 h.p. is transmitted.
Big Cottonwood Power Co.....	Sandy, Utah.....	.....	.....	500 h.p. is transmitted @ 12,000 volts.
Metropolitan Street Railway Co.....	Kansas City, Mo...	.....	70	
Northwestern Elevated Ry. Co....	Chicago	150,000	.....	
Pennsylvania Railway Co.....	.....	....	16	Telegraph and telephone wires.
.....	Manila and Cavite, Philippine Isla'ds.	.....	25	Telegraph and telephone wires.
Hartford Electric Lighting Co.....	Hartford.....	.....	11	2,000 h.p. transmitted @ 10,000 volts.

A full discussion of the relative conductivities and costs of aluminum and of copper, will be found in the papers named below.* The tensile strength of 99% aluminum when drawn into wire is between 30,000 and 55,000 lb. per sq. in.; but the Pittsburg Co. have produced an alloy which possesses a breaking strain equal to 65,000 lb. per sq. in., or equal to that of cold drawn copper. The conductivity of this alloy is, however, only 50% that of copper.

The difficulties in jointing aluminum wires have been overcome by adopting mechanical joints. Several of these have yielded satisfactory results.

On the Snoqualmie Falls transmission line, the MacIntyre joint is in use. This is made by drawing the ends of the two wires in opposite directions through a small sleeve of aluminum, 9 in. long by 0·0625 in. thick, and by then flattening in the sides of the tube, upon the two wires. The projecting ends of wire are finally bent back, and twisted round the outside of the sleeve.

Military Outfits.—As regards the use of aluminum for army purposes there is little progress to report in 1899. MM. Ditte and Moissan have contributed some notes upon the aluminum vessels returned from the Madagascar campaign, to the French Academie des Sciences, and all interested in the use of aluminum for culinary or similar purposes, should read their contributions on this subject.†

The researches of Ditte would seem to show that aluminum cooking vessels require very careful attention in order to keep them in a good state of preservation, and that under certain conditions a progressive disintegration of the

* *Lightning*, London, July 13, 1899; *Electrical Review*, New York, Aug. 30. 1899.

† *Comptes Rendus*, March-April, 1899. Abstracted in *Electrical Review*, London, Aug. 11, 1899.

metal can occur. Moissan's experience of the new metal is more favorable. In England aluminum has not yet been adopted for army outfits, but in India some progress in this direction has been made, as a result of the persevering efforts of Prof. Chatterton of the Madras School of Engineering.

Boat-building.—The use of aluminum for boats and steam-launches has not progressed in 1899, owing to the unfortunate experiences with the earlier boats built with this metal.

The international racing yachts, the Columbia and the Shamrock, however, both contained considerable amounts of aluminum in the form of its alloys, in their structures. The outer plating of the Shamrock down to the water-line was of an alloy of this metal, with a tensile strength of 31,000 lb. per sq. in. The plates were 0·375 in. thick. The composition of the alloy was kept secret, but it is said to have been little heavier than pure aluminum. The deck-plating of the Shamrock was also of aluminum; but was covered with canvas put on by means of a specially prepared composition. An aluminum top-gaff was also used during the trials in American waters, but it unfortunately broke on the first day. Electrolysis is reported to have occurred between the aluminum plating and the manganese bronze used in the construction of the Shamrock's hull;* but possibly like many reports, this is untrue.

Ristori, the managing director of the British Aluminium Co., has recently published some notes upon the use of aluminum for boat-building.† He considers that in no case should aluminum, or its alloys, be exposed to the action of sea-water, without a protective coating of paint or varnish, and the early failures he attributes to the neglect of this precaution.

The German Admiralty authorities have published the results of a useful series of investigations, upon the resisting properties of various alloys to the combined action of air and sea-water, when in contact with other metals or alloys.‡ Aluminum bronze was found to be unattacked when it formed the positive element in an electric couple, but it was rapidly corroded when the relationship was reversed. As aluminum bronzes are being used largely in marine engineering, these investigations and results demand attention.

Cycles.—The utilization of aluminum alloys in bicycle construction makes but slow progress, and although the leading makers advertise aluminum bicycles, the sale of these is comparatively restricted.

Printing.—The application of specially prepared aluminum plates in printing has received extended notice in previous volumes of this Annual.§ It is, therefore, unnecessary to refer here in detail to the preparation of the plates, or to the advantages offered by this new material for lithographic printing work. Progress in this direction appears to be rapid, and a very large number of firms in Europe and the United States are already making a special feature of "aluminographic" printing. Trials of the new method have also been made with successful results in England.

A company, with a capital of $552,000, has been formed for exploiting the new process in Europe, and it is stated that 100 printing firms in Germany

* *The Electrician*, Oct. 10, 1899.
† *Cassier's Magazine*, Oct., 1899.
‡ *Engineer*, Notes, Aug. 4, 1899.
§ THE MINERAL INDUSTRY, Vols. VI. and VII.

have already adopted it. This figure is probably exaggerated. In the United States over 20 printing firms are now using the new method, and excellent results are said to have been obtained.

The only disadvantage of the new process is that sharp effects cannot be obtained; but possibly this defect may be overcome when greater experience has been gained in the preparation and use of the new material.

The chief advantage is, of course, in speed. These plates can be used in rotary presses, and some of the larger printing press engineering firms in the United States have already turned out machines specially designed for "aluminographic" printing.

Some useful practical notes by E. Wagner upon the use of aluminum in lithographic printing work, will be found in several issues of the paper named below.*

Metallurgical Reductions.—In previous issues of THE MINERAL INDUSTRY, the use of aluminum in metallurgical work has been dealt with at considerable length; and the latest application of this kind—the Vautin-Goldschmidt process for the production of chromium and its alloys—has also been fully described.† All of these uses of aluminum, whether in the form of metal or powder, depend upon its great affinity for oxygen, and its ability to rob all other metals of this element when brought into contact with them under the right conditions. The use of aluminum in steel and iron foundries, in galvanizing works, and in brass foundries, is, the writer believes, still progressing. The Vautin-Goldschmidt process has, during 1899, been acquired by a company—called the "Chemische-Thermo Industrie"; and the production of chromium and iron alloys is now carried out upon an industrial scale by this company at Essen, in Germany. The same process is about to be introduced on a manufacturing scale into France. These alloys are used in the production of chrome steel.

Miscellaneous Uses of Aluminum.—In a recent issue of the *Electrical Review*,‡ Scott has recommended aluminum to electrical engineers for certain parts of switches, brush holders, and testing instruments.

At Philadelphia, the Mint authorities are reported to have ordered 10,000 blanks the size of the 5c. nickel. The assumption that these are to be stamped for coinage, is, in the absence of specific information, somewhat premature. At Blackpool, Lancashire, an aluminum paint has been used for coating the trolley standards of the electric tramway system. At Puteaux, in France, MM. de Dion and Bouton are making trials of an alloy of aluminum called "Partinium," for the bodies of steam omnibuses. This alloy has a specific gravity of 2·89; a tensile strength of between 32 and 37 kg. per sq. mm.; and is said to be much cheaper than aluminum. The last statement must be received with scepticism, until the composition of the alloy is revealed. In India, aluminum is receiving trial for the bearings of engines and of shop-shafting, and also for the internal fittings of railway carriages. At Essen, in Germany, the Goldschmidt process has been turned to account for welding the track rails of the electric tramways.

* *The Aluminum World*, 1899. † THE MINERAL INDUSTRY, Vol. VII., p. 119.
‡ *Electrical Review*, London, May 26, 1899.

At the factory in France, where artificial silk is being manufactured, aluminum has been used for constructing some portions of the plant. This use is no doubt due to the high resistance offered by aluminum to chemical attack by the nitric acid, which is used in carrying out this new manufacture.

A good solder for aluminum, capable of producing strong and durable joints, has yet to be found; and no satisfactory method of electro-depositing aluminum has, up to the present, been discovered. The aluminum (?) coating of the dome of the Philadelphia City Hall, about which so much was heard two years ago, is stated to be losing its brightness, and Prof. Richards now expresses his doubts as to whether any aluminum was ever really deposited upon it. To pay for *aluminum*, and to get *tin*, was certainly not the intention of the authorities; but if tin continues to rise in value the substitution may prove profitable.

Alloys.—Several new alloys of aluminum have come under the writer's notice in 1899. "Aluminard" is the name given to an alloy much used in Birmingham for miscellaneous articles. It can be brazed without difficulty, and can be worked up into various forms. The composition of the alloy is for the present kept secret.

Dr. L. Mach has introduced an alloy containing from 9 to 23% magnesium, to which he has given the name of "magnalium." Attempts to form similar alloys of aluminum and magnesium in the past have failed, owing to the presence of impurities in the former metal. Only the purest grade of aluminum should be used. These alloys are harder than aluminum, and therefore work cleaner in the lathe. In color they resemble silver, and they take a high degree of polish. The writer is disposed to doubt the durability of this alloy when exposed to damp atmospheres, and its use will probably be restricted to small ornamental articles and jewelry.

The British Aluminium Co. has given special attention to the production of aluminum bronzes; and one of them marked in their lists, "R_4," is stated to have a tensile strength equal to 95,648 lb. per sq. in. This is, therefore, one of the strongest alloys known.

The new alloy, "Partinium," has been mentioned under miscellaneous uses of aluminum.

Properties.—Moissan, in the contribution to the French Academie des Sciences, already alluded to, has commented upon the great improvement in the purity of the best grades of aluminum. The average percentage of impurities now rarely exceeds 0·45%, while in 1893, the purest aluminum obtainable in large quantities, contained between 1·25 and 3·50% silicon, iron, and sodium. The presence of the latter element is now exceptional. Moissan gives the following figures in support of these views: Best grade aluminum in 1893 contained 0·93–1·64% Si, and 0·32–1·66% Fe, while in 1898, these impurities amounted to 0·02–0·13% Si, and 0·12–0·32% Fe.

Ditte's researches upon the resisting properties of aluminum to attack by acids and other chemical agents, have been continued* in 1899, and have caused considerable discussion.† Moissan has attributed some of Ditte's

* THE MINERAL INDUSTRY, Vol. VII.

† *Comptes Rendus*, March-April, 1899; *Electrical Review*, London, Aug. 11, 1899.

results to the impurities in the aluminum used in his experiments; while Ditte maintains that, under certain conditions, the purest grades of aluminum can become subject to a slow disintegration. Prof. Liversidge, of Sydney University, N. S. W., has also asserted that an aluminum sheet was quickly corroded when exposed to atmospheric influences in Australia.* The question is one of great importance, owing to the rapidly increasing use of the metal for electrical transmission lines, and the writer in the paper named below† has discussed the matter at length, and has urged the necessity for instituting independent observations, relating to the action of moist and impure air on the alloys of aluminum sold for conducting purposes. Two series of observations relating to the effects of atmospheric exposure upon the physical properties of aluminum wires of various compositions have been commenced by him in South Lancashire; and he takes this opportunity to urge others interested in the subject, to commence similar investigations under the different climatic conditions obtaining in Europe and America. There is no doubt that the deterioration of the bare aluminum lines will be less in the dry climate of the Eastern States, than in the moist and fog-laden climate of Western Europe; but still valuable information could be obtained by a large number of investigations of the kind indicated above.

Technology and Uses.

During the year there has been far more advance in the utilization of aluminum for already known objects than in its metallurgy. Its great expansion was in the direction of electrical conductors, which was stimulated by the high price of copper during the year. Small articles, where lightness is an advantage and which do not require great strength, have continued to be made of this metal. There has been much discussion about the durability of aluminum culinary implements, and some use of vessels in connection with military accoutrements, but it does not appear to gain any wide use in this direction. For instruments of precision it has been used for assay balances by Troemner, and also for mining transits where lightness is essential. There has been some extension for its use in furnishing the interior of war ships in Italy, Germany, and in this country. New alloys are constantly being devised with aluminum and other metal, which are claimed by their inventors to have desirable qualities, but in general it may be said, that the extension of the use of the metal has been rather unimportant, save in the direction of electrical conductors. Some of the uses not touched upon by Mr. Kershaw in the preceding article are as follows:

Aluminum in Ship Building.—Sir William H. White, of the British Admiralty, reviewed before the Institution of Mechanical Engineers (England), the use of aluminum in ship building. So far as expereince had gone, he considered there was almost positive proof that the inevitable corrosion of aluminum alloys, which took place in salt water, was associated with galvanic action. Certain alloys in which copper was not present withstood this corrosion better than steel, and that in his opinion an alloy which not only had resisting prop-

* *Chemical News*, May 6, 1898. † *Lightning*, Aug. 17, 1899.

erty, but the requisite strength for ship building, would eventually be found. Mr. Yarrow, in constructing his torpedo boat for the French Government, had been careful to replace aluminum by steel plates on that part of the deck near the uptakes from the furnaces, fearing that the strength might become seriously affected by an unusually high temperature. For the internal fittings of war ships the Germans have found that, owing to the high conductivity of aluminum, some non-conducting material must be placed upon its surface when there is any source of heat within a short distance.

Exhaustive tests covering a period of two years have been made by the German Admiralty on the corrosion of alloys in sea water. In the case of each alloy or metal a dozen strips were used, nine being placed in the water, and three being reserved for comparison as to quality and strength. At the end of every eight months three strips of each kind were removed, and placed in the testing machine in comparison with the strips which had not been immersed. The alloys represented in the tests were copper alloys rich in zinc, bronzes containing little zinc, pure tin bronze, pure aluminum bronze and iron-aluminum bronze. Experiments were also made on atmospheric deterioration, which showed that the iron-bronze alloys suffered practically no injury, while those which contained large quantities of zinc suffered somewhat. The conclusion deduced by the Admiralty from these tests is that the corrosive action between different metals depends upon their relative position in the electrical scale. The electrical relation of the metals in respect of corrosion is the same as in galvanic action; thus pure aluminum bronze, which is practically proof against the corrosive action of sea water when in contact with metals which are electro-negative toward it, is destroyed while in contact with electro-positive metals. These results are confirmatory of Sir William H. White's views, as expressed above.

Aluminum Wires.—Flamel* states that the advantage of aluminum wire over copper depends on circumstances. He gives the following constants for copper and aluminum respectively: Conductivity, 100 and 63; specific gravity, 8·93 and 2·60; weight corresponding to equal conductivity, 100 and 48; surface for equal conductivity, 63 and 100; diameter for equal conductivity, 10 and 12·6.

Moissan concludes† that aluminum alloy with copper cannot replace a silicon bronze wire for overhead telephone wires. Calculating the sag and the co-efficient of safety for the French climate he finds the co-efficient of expansion due to temperature to be 0·000023 for the aluminum alloy, and 0.000017 for the silicon bronze. Owing to the former being so much greater, the sag in warm weather would be increased, or for the same sag the factor of safety would be reduced in cold weather.

Wires, however, have been subjected to physical tests in Chicago, which so far show their fitness for use on fire alarm circuits. A No. 10 aluminum wire compared with No. 10 hard-drawn copper wire gave the following physical results: 27 twists in 6 in. before breaking as against 55; 275 lb. breaking weight against 515; 1·135 in. elongation in 5 ft. against 3·06 in. The

* *La Nature*, Dec. 16, 1899.

† *L'Industrie Electrique*, Dec. 10, 1899.

weights were respectively 51·5 lb. to the mile against 173 lb. While the weights were approximately as 1 to 3·5, the breaking weights and number of twists were as 1 to 2.

Aluminum Alloys.—Experiments in the Sibley Laboratory at Cornell University have shown that an alloy of two-thirds aluminum and one-third zinc possesses some useful qualities. It is white and takes a fine finish. It melts at 800 to 900° F., and can be liquefied in a ladle over an open fire. In the liquid form it fills a mould, running into the small parts better, it is claimed, than brass. It is, however, more brittle than brass. Small castings can be made from it without the use of a foundry furnace. The tensile strength, deduced from small bars, is 22,000 lb.; maximum fiber strands deduced from transverse tests, 44,000 lb.; modulus of elasticity, 8,000,000; specific gravity, 3·3.

Prof. R. C. Carpenter, of Cornell, has obtained a patent for an alloy of copper, zinc and aluminum, which is stated to have special properties for bearings, and in general for anti-friction purposes. It is said to have the properties of hardness, unusual strength, resistance to distortion from impact, pressure or extension; also to melt readily at a comparatively low temperature, and to flow freely and to shrink little on cooling. To manufacture the alloy, what is called a tempering alloy of copper and aluminum is made by melting the copper in a suitable furnace, and then gradually adding the aluminum while the copper is heated to a point but slightly above its melting point. The melting point of the alloy is considerably less than that of either of its constituents, and it is in a condition so that it can be readily combined with zinc. It is said that the operation of making this tempering alloy is attended with some little difficulty and special furnaces have to be provided, while its admixture with the zinc can be performed with the apparatus ordinarily found in brass foundries. The following proportions by weight are said to be the best for the final bearing metal: 2 to 10 parts Al, 5 to 20 parts Cu, and 100 parts Zn. It is necessary always that Cu should be in excess of the aluminum.

The presence of so small a quantity of 0·01% Al in brass, it is claimed, will impart such liquidity to the alloy, that from one-quarter to one-third more patterns may be added to the gate, over and above what could be used if pure brass castings were made. The alloy works a little harder, but the hardness can be reduced by annealing at a red heat and plunging the metal into water.

To make aluminized zinc, a method differing from Prof. Carpenter's is usually employed by brass founders manufacturing aluminum brass. The aluminum is first melted in a plumbago crucible and the proper quantity of zinc to make an alloy of the desired composition is added in small quantities to the molten metal, with frequent and thorough stirring. The alloy is then cast into ingots of approximately the same weight, so that the foundry men may introduce about the proper quantity into a melt without weighing the ingot. These are made to carry 5 to 10% aluminum. Coke or petroleum fire is recommended, as the alloy has a tendency to absorb impurities of coal liberated in firing.

In melting aluminum a portion is oxidized and is skimmed off the surface of

the molten metal. Contrary to what would be expected these skimmings are not the oxide of aluminum, as certain observations have led E. S. Sperry to believe this dross to be either a mixture of the metal with the oxide or the mixture of oxide and metal with the presence of nitrogen. Mr. Sperry's method of recovering a portion of the metal in the dross is to melt it with a known quantity of fused copper in a plumbago crucible. The dross is stirred in and skimmed off from time to time. The product is an alloy of copper and aluminum which can be used in the manufacture of aluminum bronze, or can be utilized in treating a further quantity of skimmings. The recovery is very imperfect, the skimmings being reduced only 28 to 34% in weight. These second skimmings are thrown away, and it was when some of these were accidentally wet that the smell of ammonia led Mr. Sperry to suspect the presence of nitrogen, which was confirmed by more exact experiments.

If the claims of the inventor, William A. McAdams,* can be substantiated, he has succeeded in making an alloy of much higher strength and stiffness than any aluminum castings yet made. Dr. Chas. F. McKenna has certified that the alloy (macadamite) has a tensile strength of 40,000 lb. to the sq. in., a transverse strength of 3,200 lb., and a compression strength of 100,000 lb. It is said by the inventor to cast and work well, to take a high polish and to roll even. Its specific gravity is 3·39. The patents cover the alloy itself, and the method of casting it. Briefly, the method consists in preventing, during cooling, the segregation of individual elements in an alloy of Al, 72%, Zn, 24%, and Cu, 4%, by hastening and regulating the cooling of the cast metal. On a sphere of 1·5 in. in diameter the heat should be removed as rapidly as the rate of one-fifth calorie per second, but never more rapidly than two calories per second; the best results are obtained by removing the heat at the rate of 1 to 1·1 calories per second, being much more rapid than the dissipation of heat in ordinary casting. For alloys of different weight and forms corresponding rates of cooling are observed.

E. S. Sperry describes a novel and peculiar behavior of an alloy of nickel and aluminum. The alloy was made by melting nickel under a cover of borax and stirring aluminum into it, when the alloy became incandescent and boiled. Some of the ingots which were cast were placed to one side and when in three months' time the box in which they were contained was opened, nothing was found except a dark gray powder. Experiments were made with the substitution of other fluxes for the borax, but the same results were encountered. An alloy, however, of 90% Ni and 10% Al does not seem to disintegrate in this manner.

An alloy of tungsten with aluminum is used in the bodies of motor-cycles, made by De Dion and Bouton in Paris. It is claimed the metal has a specific gravity of 2·89 cast and 3·09 rolled. Its tensile strength is given as 44,500 to 52,600 lb. per sq. in. The elongation varies from 6 to 8%. While nearly as light as aluminum it is said to be cheaper and to possess greater strength.

R. I. Roman† has patented an alloy of copper 86 parts by weight, nickel, 3;

* United States Letters Patent; reissue, No. 11,788, Nov. 14, 1899; original Nos. 634,904, 635,053, 635,054, Oct 17, 1899.

† English Patent No. 6,728, March 19, 1898.

tungsten, 1; and aluminum 10 parts, or slight variations of the same. The copper and nickel are first melted together and the tungsten, wrapped up in aluminum cartridges, is pushed into the alloy, which is granulated in cold water. The shots are dried and remelted under mixture of borax and fluorspar, when the required quantity of aluminum is added. The alloy is poured into chilled ingot moulds, melted again and cast into the required shape in sand or other moulds.

Aluminum Paints.—The consumption of finely divided aluminum for decorative painting has increased greatly. Letter boxes in many of the larger cities, receiving a coating as well as much structural iron work in exposed positions. The grinding of the so-called aluminum bronze in making paints has caused several explosions in Continental factories. M. Stockmeir* has investigated the cause. He found that the powder itself is insensible to shock and stable when being ground, but mixed with chlorate of potash it detonates either on being struck or simply rubbed. Shaken in a vessel containing air it can be detonated by an electric spark. The powder is capable of decomposing water with evolution of hydrogen, as it is hygroscopic, and will absorb up to 1·4% of atmospheric water, which becomes resolved into its elements to a certain extent. Thus in grinding 5 to 6 kg. from 43·4 to 52·08 liters of hydrogen may be evolved. This mixed with air may be easily exploded by a spark from a small stone or other foreign body passing through the rollers. Explosions may be avoided by observing the following precautions: (1) The powder must be perfectly dry before being placed in the grinding machine. (2) The shop should be kept at the same temperature as the machines in operation. (3) If the powder becomes brown the machine must be stopped, the charge removed and placed in a cool place, and the machine carefully cleaned and aired. (4) The shop should also be well aired if the powder begins to show signs of attack during drying, or while being mixed in the receivers. (5) The air should be kept as free as possible from aluminum dust.

Plating Aluminum.—MM. Lanseigne and Leblanc† give the following formulas for electrolytic baths. For plating with silver, 20 g. of silver nitrate, 40 g. of potassium cyanide, 40 g. sodium phosphate, 1,000 g. water. For plating with copper, 300 g. copper cyanide, 450 g. potassium cyanide, 450 g. sodium phosphate, 5,000 g. water. For plating with gold, 40 g. of gold chloride, 40 g. of potassium cyanide, 40 g. sodium phosphate and 2,000 g. water. For plating with nickel, 70 g. nickel chloride, 70 g. sodium phosphate, 1,000 g. water. The baths are to be heated up to and maintained at 140 to 158° F. The anodes must consist of the same metal which is dissolved in the bath.

Action of Acids on Aluminum.—In a communication to the Académie des Sciences, M. A. Witte referred to the popular belief that aluminum was in general unacted on by acids as erroneous. The apparent inalterability of this metal is really due to the fact that in the presence of such reagents, the metal clothes itself with a layer of aluminum, thin, but continuous and impervious. When, however, steps are taken to remove this coating or to prevent its

* *Zeits. für Untersuch. Nahr. u. Genuss-mittel*, 1899, p. 49, through *Chemical News*.
† *Journal de Pharm et de Chimie*.

formation, aluminum is very readily dissolved by acids, as was to be expected from its great heat of oxidation. A solution of common salt and acetic acid acts on the metal pretty readily, the reaction being greatest near the surface, where atmospheric oxygen gets most ready access to the metal. The solution in question has the power of dissolving alumina, thus robbing the metal of its protecting coat and exposing its surface to fresh oxidation, which is dissolved in turn. A solution of sea salt alone will act on the metal if oxygen and carbonic acid gas are supplied at the same time. Further, if a plate of aluminum, after immersion in sea water, is exposed to the air without being cleaned, the corrosion will continue for an indefinite period.

Metallurgical Processes.—The process of F. A. Gooch* heats either the oxide or the sulphide, the latter being much more readily reduced. The process differs from those previously devised inasmuch as the aluminum sulphide is formed in the bath itself and not externally produced. Three parts of sodium fluoride to two parts of aluminum chloride are fused in an iron tank with sleeves for the insertion of the anodes. The bottom and the sides halfway up are lined with carbon, the rest of the tank and the sleeves are lined with alumina, which prevents the anodes from forming electrical contact with the tank and casing. The anodes, formed of carbon, and perforated longitudinally, are connected by copper rods to a metal bar which rests on wooden supports. The carbon lining acts as a cathode. Carbon bisulphide is passed through the perforated anodes into the bath and the gaseous products arising from the reaction escape through a valve, lined with compact carbon, suspended over the bath; aluminum sulphide and carbon monoxide are formed at the same time. Then an electrical current of about 0·9 volt, varied according to the constitution of the bath and size of the electrodes, is passed through the bath, reducing metallic aluminum at the cathode while sulphur passes from the anode through the outlet in the overhanging bell. Alumina is added to the bath through the space between the rim of the bell and the interior surface of the tank and may be fed with regulation of the supply of carbon bisulphide so that the production of aluminum sulphide and the consequent metallic aluminum keeps pace with the electrolysis.

E. S. Sperry employs an apparatus similar to that of Gooch, and makes use of similar reactions. He forms a bath by fusing together a suitable compound of aluminum and a suitable halogen compound of a metal more electro-positive than sulphur, adding to the bath a suitable compound of sulphur together with alumina, then passing an electric current of low voltage through the fused mass.

V. Alum and Aluminum Sulphate.

Alum, Artificial.—The consumption of alum in the United States in 1899 increased, while production is now under the control of a few concerns. The two largest producers are the Pennsylvania Salt Manufacturing Co., and the General Chemical Co. The latter firm has absorbed the plants of the Nichols Chemical Co., of New York and of the Martin Kalbfleish Chemical Co., of New

* United States Patent No. 631,253.

UNITED STATES PRODUCTION AND IMPORTS OF ALUM.

Year.	Production.								Imports. (a)		
	Alum.			Aluminum Sulphate.			Total Reckoned as Alum. Sh. Tons.	Total Value.	Short Tons.	Value.	Per Ton.
	Short Tons.	Value.	Per Ton.	Short Tons.	Value.	Per Ton.					
1896..	14,090	$442,700	$30·00	42,340	$1,056,000	$25·00	77,700	$1,478,700	2,763	$86,372	$31·26
1897..	15,456	463,680	30·00	46,355	1,158,875	25·00	79,600	1,622,555	2,373	78,683	33·24
1898..	18,791	563,730	30·00	56,663	1,416,675	25·00	97,302	1,980,405	(b) 893	16,187	18·13
1899..	27,276	845,556	31·00	81,805	2,106,479	25·75	127,430	2,952,035	(b) 858	14,953	17·49

(a) Includes alumina, alum, alum cake, aluminum sulphate, aluminous cake, and alum in crystals or ground (b) There was also imported in 1898, 1,205 short tons ($76,884) of alumina hydrate, or refined bauxite, and in 1899, 1,926 short tons ($119,202).

York. The General Chemical Co. manufactures alum from bauxite, while the Pennsylvania Salt Manufacturing Co. uses Greenland cryolite.

The statistics of the production of alum and aluminum sulphate given in the above table are computed from the consumption of bauxite and cryolite in the United States, and the production of aluminum, it being assumed that what is not used for aluminum is used for making the sulphates. The yield of both American and imported bauxite is well known, and consequently this is an accurate method of determining the production in so far as it is expressed in terms of crystallized alum. The division into crystallized alum and aluminum sulphate is estimated, and consequently is only approximate. However, since it is apt to be misleading to report the entire production as crystallized alum, of which really only a comparatively small amount is made, we began in 1897 to report our statistics of this industry in the present form. Any apparent discrepancy is thus accounted for.

In 1896 there were no producers of alum shale in the United Kingdom, but in 1898 the output was 13,617 tons, valued at £1,702.

Market Conditions in 1899.—The market prices of alum showed little change, lump selling in New York in 1899 at $1·65 and $1·75 per 100 lb., ground at $1·85, and powdered at $2·50. At the same time aluminum sulphate was valued steadily at $1·25 and $1·50 per 100 lb. for the commercial quality, and $2 for the purest.

ALUM, NATURAL.—But little work has been done on the deposits on the Gila River, in New Mexico, although they are said to be large, and could be worked profitably on a large scale. China exports some natural alum, in 1898 shipping from Shanghai 4,294 tons with the value of £11,429, as against 4,758 tons, valued at £18,071 in the previous year. About 3,000 tons of alunite were shipped from Bullahdelah, New South Wales, to the works of the United Alkali Co., Runcorn, England. The deposit here is said to be extensive.

AMMONIA AND AMMONIUM SULPHATE.

THE production of ammonia (reported as its equivalent sulphate salt) and ammonium sulphate by by-product coke oven plants in the United States in 1898, was 3,589 short tons. Statistics for 1899 are not available at the time of publication. Ammonium sulphate is recovered at only a few of the gas works in the United States. Those at Peoria, Ill., Dayton and Cleveland, O., Indianapolis, Ind., and Boston, Mass., are the principal ones. At Rochester, Washington, Buffalo and Richmond, Va., the gas works product is turned into ammoniacal liquor and sold to chemical works.

The imports of ammonium sulphate into the United States for the past five years is given in the tables of exports and imports at the end of this volume.

The report of the Chief Inspector of Alkali Works for the United Kingdom for 1898 shows that the production of ammonium sulphate during that year amounted to 196,422 tons, a slight decrease from the previous year. From gas works, 129,590 tons were obtained; from shale oil works, 37,264 tons, a figure almost identical to that of the previous year; from blast furnaces, 18,000 tons, a trifling increase; and from recovery coke ovens, 11,568 tons, an increase of nearly 1,000 tons from this source. No very remarkable increase is expected from either the shale oil works or blast furnaces, though if the bee hive coke ovens in use were all replaced by recovery ovens, it is calculated that the additional production would amount to 110,000 tons annually. It is anticipated that in future years a large increase will be shown from this source of production. As to the method of recovery in gas works, reference should be made to the article by Hans A. Frasch, MINERAL INDUSTRY, Vol. VII. The production of ammonium sulphate for 1895, 1896, 1897, 1898 is shown in the following table:

PRODUCTION OF AMMONIUM SULPHATE IN THE UNITED KINGDOM. (In tons of 2,240 lb.)

	1895.	1896.	1897.	1898.		1895.	1896.	1897.	1898.
Gas works...	119,645	127,498	132,724	129,590	Shale works.	38,335	37,822	37,153	37,264
Iron works...	14,588	16,511	17,779	18,000	Coke works.	7,083	9,078	10,624	11,568

American Market.—The year just closed has realized better profits to the manufacturer than its predecessor, notwithstanding the increased cost of raw material. The trade has been kept well in hand by the two largest combina-

tions—the American Agricultural Chemical Co. in the North and the Virginia-Carolina Chemical Co. in the South. A good export business has been built up in the leading ammoniates, while our home consumption has also improved. The outlook for the year 1900 is for higher prices, as the cost of manufacturing fertilizers will likely be from $1·50 to $2·50 per ton over 1899, owing to the upward movement of raw material. A number of new plants have been erected during the year, while the producing capacity of other works has been enlarged. The average price per 100 lb. of gas liquor ammonium sulphate, basis 25%, in New York in 1898 and 1899, was as follows:

Year.	Jan.	Feb.	Mch.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average.
1898.....	$2·365	$2·613	$2·457	$2·415	$2·394	$2·481	$2·445	$2·488	$2·575	$2·535	$2·606	$2·630	$2·500
1899.....	2·716	2·662	2·667	2·810	2·975	3·337	3·190	3·111	3·065	2·975	2·850	2·895	2·938

English Market.—In England there was an active though irregular market for ammonium sulphate throughout the year, prices being reached which had not been attained for five years previously. Quotations per long ton in January were £10 2s. 6d., rising to £10 10s. Toward the end of February prices fell to £10, but rose to £11 at the end of March without altering the demand. May was an extremely active month and prices rose to £13 before the end, although a decline set in after that time. In September the market opened at £12 6s. 3d., but declined rapidly, £10 12s. 6d. being touched in the middle of November. After this, a reaction set in and prices reached £11 8s. 9d. in the middle of December, but fell off toward the holidays to an even £11, or 17s. 6d. higher than the opening prices of the year.

Technology and Uses.

W. Carrick Anderson and James Roberts* state that although distillation in the blast furnace can be carried on for a fairly lengthy period at a lower temperature than that of the gas retort, the yield from blast furnaces on the average is only 20·2 lb. of ammonium sulphate per short ton of coal as against 22·4 lb. from the gas works. This is ascribed to the presence of iron ore, which brings about the oxidation of ammonia by the oxide of iron at a low temperature. A sample of splint coal which gave, when distilled, about 31·65 lb. of ammonium sulphate per ton of coal, yielded when treated as nearly as possible under the same conditions of temperature, but in the presence of calcined clay band ore in the proportions of an ordinary blast furnace charge, only 23 lb. ammonium sulphate per ton of coal. Another coal, which gave 30·73 lb. ammonium sulphate per ton when distilled with ores yielded only slightly above 17 lb. In this paper on the recovery of nitrogen in coal distillation, the writers come to the following general conclusion:

(1) When coal is heated, part of the nitrogen contained in the nitrogenous constituent comes off as ammonia in the gases, part goes to form nitrogenous compounds in the tar, and the rest remains fixed in the solid residue or coke.

* *Journal of the Society of Chemical Industry*, Dec. 30, 1899.

(2) The percentage which is thus evolved as ammonia is dependent on the structure of the nitrogenous constituent of the coal and on two other factors: The amount of hydrogen available to combine with and carry off the nitrogen, and the period during which the nitrogenous body is exposed to the influence of the hydrogen above a certain limit of temperature.

(3) Heating the nitrogen-containing body has no effect alone in dislodging the nitrogen; there must be a simultaneous action of hydrogen or certain hydrogenized compounds.

(4) The presence of free nitrogen in the gases obtained from the distillation of coal, amounting frequently to 25% of the total nitrogen of the coal, is due to dissociation of the ammonia at temperatures above 500°C.

(5) The amount of this dissociation is increased by the rise in temperature and by the increase in the time during which the gas is exposed to the heat. For this reason, although in a coke oven the charge is subjected to a more gradual heating than in a gas retort, the quantity of ammonia recovered may be actually much less than in the latter case, because in the coke oven the ammonia liberated in the lower part of the charge is subjected to dissociation conditions for a longer time in passing upward through a thick crust of strongly heated coke.

(6) In blast furnaces the iron oxide exercises a destructive effect on the ammonia at comparatively low temperature, and while, owing to the very gradual descent of the raw coal toward the hotter region of the furnace, a considerable part of the ammonia will be formed and will escape before the limit of dissociation is attained, yet the influence of the iron oxide in the ore, which is active at very much lower temperatures, may be such as to more than counterbalance the benefit derived from these favorable conditions, and bring out the average yield of sulphate at a lower figure.

According to Wagner* the conditions which have affected the actions of the manures are of various kinds. Ammonium salts have the advantage that until nitrified, the nitrogen is fixed in the soil. Sodium nitrate is often of use in light soils. It increases the power of obtaining water; it supplies nitrogen and phosphoric acid to the plants and to some extent it replaces potash. On the other hand sodium nitrate induces the production of hard crusts on clay soil and renders the application of lime necessary. The utilization of atmospheric nitrogen is still attracting the attention of chemists and inventors. Sir William Crookes suggests that the nitrogen supply for agricultural purposes be obtained from the atmosphere electrically, basing his estimates on experiments of Lord Rayleigh in oxidizing nitrogen to obtain a residue of argon; he estimates that the cost of a ton of niter reduced electrolytically by steam power would be $130, but with using water power at a cost of $0·015 per kw. hr. it would become $25 only. It is somewhat dubitable, however, that there is sufficient water power to produce the 12,000,000 tons annually required. Dr. Ludwig Mond has found that, using his improved gas producer, nearly 100 lb. of ammonia can be recovered per ton of fuel by introducing 2·5 tons of steam for each ton of fuel consumed into the producer. Eight of these producers have been introduced at the plant of the Solvay Process Co., at Syracuse, replacing the use of coal in their steam

* *Deutsche Landw. Presse*, 1898, Nos. 30 to 31.

plant, and elsewhere. The production of fuel gas and the recovery of ammonia is said to yield 25% profit on the capital invested.

John M. McCandless, State Chemist of Georgia, points out that fertilizers are frequently adulterated with what is known as a "filler," which often consists of pulverized locomotive cinder, graphitic schist, iron cinder from the pyrites furnaces, powdered coal slack, crematory ashes and other substances. These materials often contain a very small amount of nitrogen, phosphoric acid and potash, usually unavailable or becoming available only in the course of time. The manufacture of these adulterants has grown rapidly, and companies have been organized and factories built for this special purpose. He suggests as a remedy that the sale of goods known as ammoniated acid phosphates and acid phosphates with ammonium potash, containing less than 2% of ammonia or potash, should be absolutely prohibited, as they are usually loaded with a filler.

The Transport of Ammonia.—The British Board of Trade has classified liquefied ammoniacal gas as dangerous and has instructed that special precautions should be observed when this is carried as cargo. The vessels containing it should have been tested to 675 lb. to the sq. in., and should be stored where they cannot be affected by the heat from the boilers and remote from living quarters. The drums should be 12 gal. capacity when an aqueous solution of ammonia of sp. gr. 0·880 at 60°F. is carried, and an empty space equal to 5·33% of its total volume should be left in each drum. These drums should be capable of withstanding a pressure of 66 lb. to the sq. in., and should be packed in cases with not more than four drums in each, the interstices being filled with sawdust. Care should be taken not to expose drums containing liquid ammonia to the rays of the sun or the influence of heat. Ammonia solution of a sp. gr. of 0·959 and upward, is exempted from these instructions.

A. Lunge and W. Hefter describe the Urban method of determining ammonia. Briefly outlined it is as follows: The ammonia is passed directly from the reservoir into a glass pipette with a stop cock at each end, where it is weighed. This pipette is placed in a vertical position and connected with three absorption tubes, filled with solid potash, the weight of each tube having been accurately determined. The upper stop cock is then opened to allow the gaseous ammonia to traverse the potash tubes slowly. After the evaporation of the liquefied ammonia, the pipette is placed horizontally in a hot air oven, and gradually heated to 70 to 80°C., at the same time, passing a current of dry air, freed from carbonic acid through the apparatus. The weak point of this method is that the impurities which should be found in the absorption tubes are volatile at from 70 to 80°C., and pass off with the current of gas. Urban improved this method, first devised by Bunte and Hitner, by weighing the residue from the evaporation after heating to 30°C. only. The small amount of ammonia remaining therein does not affect the accuracy of the operation. The Lange and Hertz method* is preferred by the authors on account of its simplicity.

* *Zeitschrift für Angewandte Chemie*, 1897, p. 224.

ANTIMONY.

THE Mathison Smelting Co., of New York, with works at Chelsea, Staten Island, has been, with the exception of the Chapman Smelting Co., of San Francisco, the only producer of metallic antimony in the United States. It is understood that the latter company is controlled by the former, and both decline to give information.

PRODUCTION AND IMPORTS OF ANTIMONY IN THE UNITED STATES.

Year.	Production.				Imports.				
	Metal.		Ores.		Metal or Regulus.		Ores.		Total Value.
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	
1895.....	932,000	$70,332	2,166,000	$37,905	3,449,901	$251,850	668,610	$31,893	$83,743
1896.....	1,226,000	84,717	300,000	4,750	3,583,300	170,092	1,180,828	21,392	191,484
1897.....	1,500,000	107,250	1,000,000	16,000	1,146,696	46,955	5,502,132	167,373	214,328
1898.....	2,000,000	165,000	(a)	(a)	2,025,133	143,909	3,725,222	50,256	194,165
1899.....	b2,500,000	241,250	1,200,000	20,000	3,160,697	240,988	3,982,133	47,841	288,829

(a) There was an increase in the production, but complete statistics were not collected. (b) Estimated.

California.—During 1898 the Chapman Smelting Co., San Francisco, is said to have produced about 300 tons of antimony from about 600 tons of ore. The greater portion of the ore came from Nevada, Utah and Idaho, some 40 tons only coming from California. The Chapman works refuse to buy ore which contains less than 40%, and the Hallett process of smelting and refining is used. C. Solomon, Jr., the manager in San Francisco, states that all the ore carries gold and silver in appreciable quantities, though this is not paid for nor is it extracted. The mines near Mojave were operated during the year and a reduction plant erected. The vein is said to be from 4·5 to 20 ft. wide of ore averaging 15%. The occurrence of stibnite is said to be regular and not in bunches. Assorted ore, it is estimated, will go 50%. Timber is in the vicinity of the property and water is within a half mile.

Nevada.—The antimony deposits near Winnamucca will be, it is stated, worked actively. They were purchased by Reno capitalists in July.

THE NEW YORK ANTIMONY MARKET DURING 1899.

Throughout the year the market has been active, and prices have ruled higher than during 1898, but have not fluctuated much, the ratio of supply and demand remaining about the same. It is true the consumption in this country has increased and the production has not increased to the same extent, but imports

have become larger, and the European values have thus continued to determine the prices here. The principal supplies continue to be drawn from England, while those from the Continent are constantly increasing. Japan has during the past year exported but little.

The year opened with Cookson's selling at 9·25c., Hallett's, United States Star, "C," and Hungarian at 8·75 to 8·875c. At the end of January, in consequence of a general depletion of stock, the market slowly advanced about 0·25c. In February the demand became very persistent, and in spite of large arrivals, an advance to 10c. for Cookson's, and 9·75 to 10c. for other brands, was established. These prices ruled for a number of months, but during the summer Cookson's advanced to 10·50c. and the other brands to 10c. At the end of the year values eased off somewhat and closed, Cookson's 10c., Hallett's, United States Star, "C," and Hungarian, 9·50 to 9·625c.

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK. (IN CENTS PER POUND.)

Year.	Brand.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
1893....	Cookson's	11·00	10·75	10·75	10·75	10·13	10·50	10·50	10·25	10·30	10·25	10·00	10·25	10·50
	Hallett's	10·25	10·38	10·00	10·00	10·00	9·90	9·88	9·75	9·60	9·75	9·50	9·25	9·88
	L. X.	10·50	10·63	10·38	10·25	10·38	10·00	10·13	10·00	10·00	10·00	10·00	10·00	10·20
1894....	Cookson's	10·25	10·00	10·13	10·13	10·13	9·75	10·00	10·00	9·50	9·63	8·50	8·38	9·63
	Hallett's	9·50	9·38	9·50	9·38	9·25	9·13	8·75	8·88	8·88	8·25	8·25	8·13	8·88
1895....	Cookson's	8·00	8·25	8·15	8·00	8·00	8·00	8·00	7·85	7·85	7·75	7·75	7·75	7·95
	Hallett's	7·25	7·25	7·10	7·00	7·00	7·05	7·10	7·20	7·25	7·13	7·00	7·00	7·11
	U. S. Star	8·25	7·50	7·38	7·50	7·35	7·50	7·50	7·75	7·50	7·50	7·50	7·50	7·56
	Japanese	7·00	7·00	6·88	6·88	6·88	7·00	7·00	7·20	7·00	6·88	6·88	6·88	6·97
1896....	Cookson's	7·55	7·48	7·37	7·44	7·50	7·37	7·00	7·00	7·00	7·00	7·00	7·18	7·23
	Hallett's	7·10	6·90	6·87	6·81	6·75	6·75	6·75	6·50	6·50	6·50	6·46	6·68	6·71
	U. S. Star	7·25	7·09	7·00	7·00	7·00	6·91	6·75	6·75	6·75	6·75	6·75	6·93	6·91
	Japanese	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75	6·75
1897....	Cookson's	7·12	7·31	7·37	7·44	7·25	7·25	7·25	8·00	8·00	8·00	7·81	7·81	7·55
	Hallett's	6·80	6·90	7·06	7·00	7·00	7·00	7·00	7·28	7·37	7·50	7·40	7·40	7·14
	U. S. Star	6·80	7·00	7·06	7·22	7·05	6·78	6·87	7·28	7·37	7·50	7·40	7·50	7·15
	Japanese	6·75	6·80	6·75	7·00	6·75	6·78	6·87	7·15	7·25	7·25	7·15	7·18	6·91
1898....	Cookson's	8·00	8·00	8·25	8·37	9·20	9·65	9·75	9·75	9·75	9·70	9·25	9·25	9·08
	Hallett's	7·37	7·56	7·81	7·96	8·64	8·97	9·06	9·06	9·06	9·08	8·81	8·81	8·51
	U. S. Star	7·50	7·59	7·81	7·92	8·64	8·97	9·06	9·06	9·06	9·08	8·81	8·81	8·52
	Japanese	7·44	7·58	7·81	7·90	8·59	8·97	9·06	9·06	9·06	9·08	8·81	8·81	8·51
1899....	Cookson's	9·31	10·25	10·37	10·10	10·50	10·50	10·50	10·50	10·50	10·50	10·50	10·50	10·37
	Hallett's	8·81	9·62	9·75	9·80	10·00	10·00	10·00	9·81	9·65	9·72	9·75	9·75	9·67
	U. S. Star	8·81	9·62	9·75	9·80	10·00	10·00	10·00	9·81	9·56	9·50	9·50	9·50	9·65
	Chapman's	.....	9·62	9·75	9·80	10·00	10·00	10·00	.....	.....	.....	.....	.....	.....
	Hungarian	.....	.....	9·75	9·80	10·00	10·00	.....	9·75	9·56	9·50	9·50	9·50	.....

WORLD'S PRODUCTION OF ANTIMONY ORE. (*a*) (IN METRIC TONS.)

Year.	Austria.		Borneo. (*b*)		France and Algeria.		Hungary.		Italy.		Japan.	Mexico. (*b*)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Tons.	Value.
1894..	686	$32,943	599	$57,320	6,319	$86,231	1,265	$27,851	1,504	$45,059	1,170	80	$2,896
1895..	695	26,688	657	64,100	5,703	75,367	1,240	26,877	2,241	40,254	1,045	600	14,281
1896..	905	38,702	(*c*)	(*c*)	6,333	84,509	1,361	22,572	5,066	60,590	827	3,231	32,093
1897..	864	38,255	(*c*)	(*c*)	5,466	83,532	1,800	34,563	2,150	27,657	(*c*)	5,873	71,885
1898..	679	57,167	.....	.....	4,571	69,462	2,201	20,219	1,981	43,822	(*c*)	.....	.....

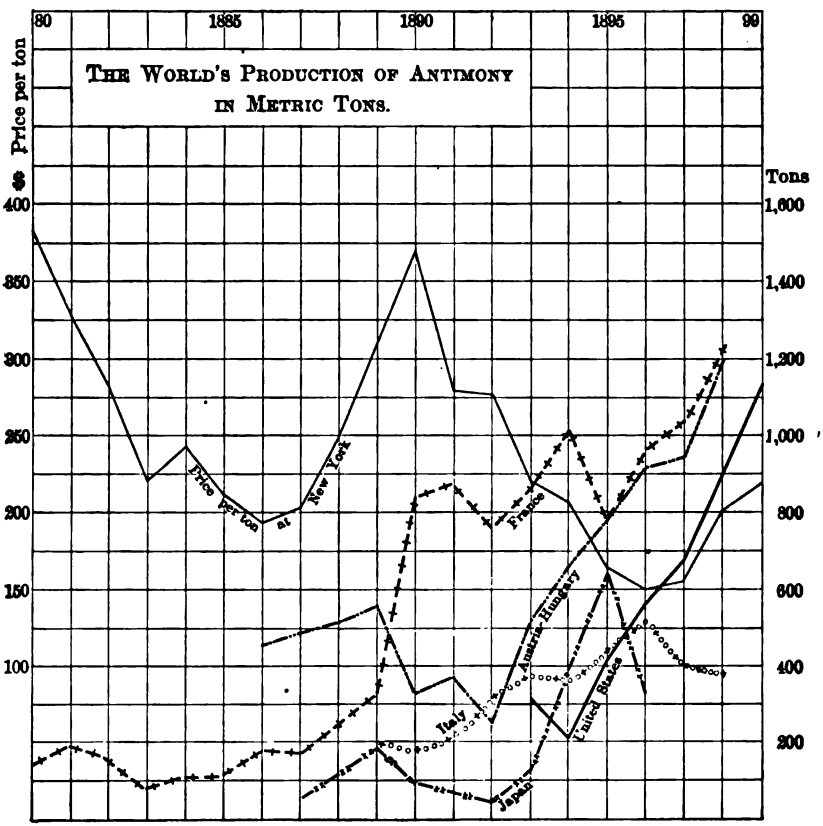
Year.	New South Wales.		New Zealand.		Portugal.		Queensland.		Spain.		Turkey. (*b*) (*d*)	United States.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Tons.	Value.
1894....	1,270	$98,720	45	$3,805	808	$47,698	28	$1,400	15	$900	1,545	150	$9,075
1895....	486	26,255	55	7,430	753	33,889	(*c*)	(*c*)	44	700	*Nil.*	982	37,905
1896....	135	9,170	21	2,250	595	26,775	(*c*)	(*c*)	54	1,385	100	136	4,750
1897....	172	18,060	10	763	417	28,063	(*c*)	(*c*)	354	6,483	400	454	16,000
1898....	84	4,580	.....	.....	245	6,796	.....	.....	180	2,149	.....	.....	.....

(*a*) The foreign statistics are derived from the official reports of the several governments; those for the United States were collected specially for THE MINERAL INDUSTRY. (*b*) Export figures. (*c*) Not yet reported. (*d*) Fiscal years. The Turkish statistics are of doubtful accuracy.

WORLD'S PRODUCTION OF ANTIMONY METAL. (*a*) (IN METRIC TONS.)

Year.	Austria.		France and Algeria.		Germany. (*d*)		Hungary. (*b*)		Italy.		Japan.	New South Wales.		United States.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons	Value.	Tons.	Value.	Tons.	Tons.	Value.	Tons.	Value.
1894..	279	$51,523	1,012	$136,024	424	$65,244	885	$50,505	351	$57,626	408	30	$4,535	205	$29,200
1895..	296	44,175	779	98,734	969	107,960	465	57,507	423	56,986	640	44	6,460	422	70,332
1896..	422	52,655	969	130,217	1,326	156,111	500	59,313	538	72,440	311	23	3,200	556	84,717
1897..	424	51,350	1,033	141,857	1,665	210,744	523	63,360	404	57,072	(*c*)	(*c*)	(*c*)	680	107,250
1898..	343	42,236	1,226	163,200	2,711	302,892	855	109,681	380	62,550	(*e*)	(*c*)	(*c*)	907	165,000

(*a*) From the official reports of the respective countries. (*b*) Crude antimony and regulus. (*c*) Not reported. (*d*) Includes manganese. (*e*) Statistics not yet available.



Although the United Kingdom does not appear in the diagram, it is perhaps, the most important producer of antimony. The ores, however, are derived almost entirely from foreign sources, and there are no statistics as to the output of metal.

Asia Minor.—At Djinli Kaya, southeast of Smyrna, two antimony lodes occur which have yielded a considerable quantity of ore. The first ore extracted was high grade stibnite, but the ore at present requires concentration and as it contains a considerable quantity of pyrites it is doubtful if a successful separation can be made.

Australia.—At Neardie, in the Wide Bay district of Queensland, antimony was worked for a number of years, but the mines, like those of Northcote, near Herberton, are closed down. In conjunction with silver and copper, antimony ore has been found at Ravenswood.

Burma.—Antimony ore containing gold has been found near the Siamese frontier. A concession for working it has been given to a Dutch merchant who has erected machinery.

Corsica.—Ores of antimony are found in many localities in the western half of the island; the important ones are those of Luri-Castello, Meria and Ersa in the extreme western extension, called Cape Corsica. The output of these mines for the past 10 years has averaged about 1,500 tons per year. Stibnite occurs in veins which cut the oldest part of the schist formation and are probably older than the Pre-Silurian. Serpentines quartz, calcite, zinc blende and an unknown green mineral accompany the sulphide, while pyrite bournonite and cinnabar are occasionally met with. The veins vary greatly in extent, the average thickness being only 8 to 12 cm. and ore shoots, up to 50 to 80 m. long can be seen in some of the mines.*

France.—In 1898 the product of regulus and oxide of antimony rose to 12·26 tons oxide from 10,033 tons in the preceding year. All the ores employed were of French origin. Three-quarters of the production came from the Haute-Loire.

Mexico.—Deposits of antimony are known to exist at Point Lobos and near Santa Anna in Sonora. Small quantities of ore were shipped during the year from these mines, which are owned by Americans.

Nova Scotia.—Considerable work was done at the Rawdon antimony mines. The Northup has been unwatered and a shaft sunk to 240 ft. The ore occurs in shoots of various width, the vein averaging about 8 in. wide. There is another vein on the property about 4 in. wide, which was worked during the past season. The Messervey and MacDugall mine, a short distance north of the Northup, has a vein averaging about 5 in. wide. On this a shaft has been sunk 55 ft. deep. A sample of antimony ore from this district assayed by the Mines Department gave 45·75% antimony and 2·48 oz. gold per short ton. A second sample gave 18·21% antimony, and 0·23 oz. gold.

Progress in the Metallurgy of Antimony.

Haremann has devised a process of eliminating antimony from antimonial lead ores by roasting the crushed ore with alkaline carbonate and carbon. A double sulphide of the alkali and the antimony is formed which can be extracted by lixiviation, leaving the precious metal contents and the lead behind. The antimony is precipitated as a sulphide by addition of an acid, preferably, sulphuric acid, as then an alkaline sulphate is formed which is recovered for future use.

Antimony in Antifriction Alloys.—Prof. S. Behrens has investigated the qualities of babbitt metal. When babbitt metal in a pasty condition is pressed between hot slabs of iron, tin containing 3% of antimony and copper will flow out and leave a hard brittle paste, which is chiefly formed of a tin, antimonial alloy. An alloy composed of 90% Sn and 10% Cu gave a hard residue, which on being decomposed and some metallic copper washed away, was shown to corre-

* M. Neutien, *Etude sur les gites Minéraux de la Corse*, Vol. XII., 1897, pp. 281-296. Abstracted from *Zeitschrift für Praktische Geologie*, February, 1899.

spond nearly to the formula of CuSn. In alloys of antimony the tin was found to be more strongly combined. From an alloy containing 10% Sb after it was thrice heated and pressed a residue was obtained which when purified by extraction with hydrochloric acid and washed in a stream of water corresponded closely to the formula $SbSn_2$. In alloys containing 80% Sb this element can be made to crystallize out in nearly a pure state. Another alloy obtained by the treatment seemed to correspond to the formula SbSn. On the microscopical examination of bearings which had done duty on railroad cars, in the cushions which were marked as having been unduly heated rectangular crystals of the $SbSn_2$ were found poorly developed or entirely absent. Tests were made with model cushion blocks which had been cast under varied conditions. The blocks were fitted on a mandril of polished steel, running in a wooden casing at a speed of 1,600 revolutions per minute. The apparatus was so arranged that the pressure on the mandril could be varied at pleasure, and the temperature could be observed on a thermometer fitted on the blocks.

The following table gives the pressure per pound to the square inch in the longitudinal section of the mandril, and the mean increase of temperature after running for a minute. Block A was cast in a mould and cooled by running water; block B in a mould heated to 100°F. and block C in a mould heated in molten zinc.

Amount of Pressure.	4 lb. per sq. in.	8½ lb. per sq. in.	17 lb. per sq. in.	43 lb. per sq. in.
	°	°	°	°
Increase of temperature in block A	0·50	1·12	1·50	3·89
Increase of temperature in block B	0·64	0·74	0·75	1·64
Increase of temperature in block C	0·55	0·72	2·62	4·64

After the series of experiments with the charge of 43 lb. per sq. in. block A presented a running surface scarred by irregular grooves and scratches; on block B rectangular crystals stood out in relief and the surface looked as if it had been eaten by hydrochloric acid. The crystals were smooth and bright but interstices were covered with small scars. On block C the surface was scarred by grooves and scratches as well as many formed rectangular crystals. On examining the metallic sediment from the oil which had been used as a lubricant, shavings, threads and angular fragments were found in that from block A; spherical and egg-like bodies looking like minute drops of mercury in the sediment from block B, while in the sediment from block C spheroids and angular fragments both were found. The spheroids in block B were found to consist of tin with a considerable mixture of antimony.

Analytical Methods.—A discussion of methods of antimony analyses is to be found in the *Journal of the American Chemical Society.** The author, Thos. Brown, Jr., divides these ores into three classes: (1) The soluble ores, such as stibnite; (2) the practically insoluble ores, such as cervantite; (3) the partially soluble ores, as oxysulphides (stibnite mixed with cervanite). Ores of the first class are treated as follows: 1 g. of the finely ground ore is treated in a No. 3 beaker with 25 c.c. of concentrated HCl covered and boiled until apparently

* September, 1899.

decomposed and the solution does not occupy more than 15 c.c. in bulk. About 2 g. of tartaric acid is added and as soon as that is dissolved from four to six drops of concentrated HNO_3, and the whole boiled for a short period. On cooling and dilution H_2S is passed through for about an hour and the antimony precipitated as sulphide. After warming slightly it is allowed to stand in the cold for an hour or more and then filtered. These precautions having been attended to, no more antimony will be precipitated either on standing or passing additional H_2S. If lead or other metals of the fifth group has been found it may be necessary to dissolve the precipitate in excess of ammonium sulphide. The antimony is again precipitated as before. The precipitate in any case, is well washed and dried in an air bath at 110°C., to constant weight. As the precipitate contains a certain amount of free sulphur it is necessary to determine the amount of black anhydrous antimonious sulphide it contains. This is done by weighing a certain amount of the precipitate into a platinum boat which is placed in a combustion tube and the free sulphur driven off in a stream of CO_2.

The insoluble ores are best treated by fusion with a flux composed of equal parts of flower of sulphur and sodium carbonate. One gram of ore is used to eight or more parts of the mixed flux. The mixture is put in a crucible of 40 c.c. capacity covered with a portion of flux and placed in the muffle and heated cautiously to fusion which takes place in about 10 minutes. Many precautions have to be observed, as it is a delicate matter to fuse the readily volatile antimony even at a low heat without loss. The fusion once accomplished, the mass is dissolved in hot water and the solution which contains the antimony filtered off from the sulphides of iron and those of the fourth group. The antimony is precipitated as sulphide by adding a slight excess of HCl. The antimony in it is estimated as before, as antimonious sulphide.

The partially soluble ores require a double treatment following the steps of the two previous methods. The majority of ores in Mexico require this double treatment, which has given very good results in practice, being in most cases much higher than the results obtained by fire assay.

ARSENIC.

GERMANY and England still are the main sources of the supply of arsenic and arsenical compounds. Both countries show a falling off in 1898, from the production of the previous year, that in England being specially marked. As there is no production in this country, and as the consumption is increasing, the imports of arsenic, including white arsenic, metallic arsenic and arsenic sulphides, rose from 8,686,681 lb. ($370,347) in 1898, to 9,040,871 lb. ($386,791) during 1899. The imports during previous years were: 1893, 6,092,377 lb. ($180,333); 1894, 7,063,442 lb. ($218,636); 1895, 6,984,273 lb. ($237,745); 1896, 5,813,387 lb. ($215,281); 1897, 7,242,004 lb. ($352,284).

ARSENIC PRODUCTION IN ENGLAND AND GERMANY. (*a*)

Year.	Prussia.		Saxony. (*b*)		Total Germany.		England. (*c*)	
	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.
1894.....	1,147	$57,877	1,229	$109,113	2,389	$166,798	4,878	$243,070
1895.....	1,788	98,363	1,217	108,826	3,014	207,416	4,875	260,990
1896.....	1,750	130,539	882	90,626	2,637	221,397	3,674	227,415
1897.....	1,924	143,775	1,063	152,122	2,989	270,949	4,239	373,975
1898.....	1,624	121,818	1,053	131,710	2,679	253,623	4,241	268,985

(*a*) From the official reports of the respective countries. (*b*) Includes arsenious acid, red, yellow and white glass, and metallic arsenic, except the figure for 1897 which represents metallic arsenic alone. (*c*) Arsenious acid.

The Devon Great Consols mine is one of the largest producers of arsenic in the world. Operations commenced as early as 1844, when the mine was worked for copper. Until April, 1899, the value of the production of copper had amounted to £3,468,122, while the arsenic was valued at £590,178. Dividends have been paid aggregating £1,223,878, while the owner of the land received royalties amounting to £279,679. The deposits of copper extended for two miles in length, but the arsenical pyrites, or mundic, as it is locally termed, was considered of so little value that it was thrown on the waste dump for many years; now the chief source of profit is from this mineral. The metallurgy practiced here was thoroughly described in THE MINERAL INDUSTRY, Vol. III.

Determination of Arsenic.—J. F. Bennett, Jr.,* describes a modification of Pierce's method for the determination of arsenic in ore. Instead of neutralizing by ammonia, and expelling carbonic acid by boiling with excess of nitric acid, or by using zinc emulsion, as in Candy's modification of Pierce's method, the author adopts the following practice: 0·5 g. of ore is fused with 6 to 10 times its weight of a mixture of sodium carbonate and potassium nitrate in equal parts. The filtrate, after the mass has been treated with boiling water and

* *Journal of the American Chemical Society*, 1899, pp. 431-435.

filtered, contains the arsenic as an alkaline arsenate. It is acidified with excess of acetic acid, and boiled to expel carbonic acid. On cooling it is neutralized with sodium hydroxide, phenol-phthalein being used as an indicator, and a neutral solution of silver nitrate is added with continual agitation. The precipitate readily subsides, is washed once or twice by decantation, is then collected on a filter and washed thoroughly with cold water. It is then dissolved in dilute nitric acid, the solution made up to definite volume, and the silver determined by titration with standard potassium thiocyanate. The silver, however, may be directly determined by cupellation, if chlorides are absent. By a number of carefully concluded experiments it was found by adding 1 mg. of silver to the results obtained by cupellation the results were in close agreement with those obtained by titration.

J. and H. S. Pattison* describe a method of separation of arsenic from other metals of the same group, which depends on the insolubility of As_2S_3 in strong HCl, the sulphides of lead, cadmium, tin, antimony, and bismuth being entirely soluble. Care must be taken that the arsenic should be in triad form. This is accomplished by the addition of stannous chloride. Zinc sulphide in some cases must be added to insure complete saturation of the acid by H_2S. The As_2S_3 when separated by filtering, is dissolved by heating with concentrated H_2SO_4 until all the sulphurous acid is expelled, when the solution may be titrated with standard iodine solution after dilution.

Manufacture of Plumbic Arsenate.—Arsenate of lead is used largely in destroying the gypsy moth. According to A. H. Kirkland,† it is most economical to use high-grade sodium arsenate, free from impurities, in making this compound; foreign substances, as sodium chloride, neutralize a portion of the soluble lead nitrate before the reaction of the sodium arsenate commences. He also recommends the use of lead nitrate instead of lead acetate as commonly used. In manufacturing a ton of lead arsenate 888 lb. of sodium arsenate are required, costing $46·18, and 2,398 lb. of lead nitrate costing $161·87, the total cost being $208·05. Using sodium arsenate but employing lead acetate, the total cost is $246·92 per ton of lead arsenate, leaving a balance of $38·77 in favor of using lead nitrate instead of lead acetate.

Market Conditions.—In 1899 the glass trade and Paris green markets were large consumers of arsenic, while prices showed little fluctuation, owing to a combination of the producers. White arsenic sold at 4 to 5c. per lb. in New York, according to time of delivery. Some Canadian arsenic from Deloro, Ontario, was sold in this market at quotations fractionally less. The mines there are able to produce arsenious acid cheaply as a by-product in treating auriferous arsenical pyrites. Red arsenic from Germany held steadily at 7·75 to 8·50c. per lb.

Paris Green.—The demand in 1899 was good, though competition among makers caused some fluctuation in prices. Pure Paris green sold in the first half of the year at 12 and 14c. per lb. in New York, and in October, at 11 and 12c., while in December the quotations were 14 to 15c. per lb.

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* *Journal of the Society of Chemical Industry*, Vol. XVII., p. 211.

† A paper read Aug. 19, 1899, before the Association of Economic Entomologists.

ASBESTOS.

THE production of asbestos in the United States in 1899 was 912 short tons ($13,860) against 885 short tons ($13,425) in 1898. Although asbestos has frequently been found in California, no mines of prominence have as yet been developed. The Drouillard and Helm mine, in Tulare County, however, has produced some asbestos during the year, which is said to be of fair quality, as much as $75 per ton being paid for it. In Georgia, the Sall Mountain mine was sold early in the year to London parties for a sum stated to be in the neighborhood of $2,000,000. A discovery was made in Idaho on the south fork of the Clearwater River, 5 miles above Harpster. The asbestos here is stated to be of unusually fine fiber, and to occur in wide seams.

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.				Imports.		
	Short Tons.	Metric Tons.	Value.	Value per Metric Ton.	Manufactured.	Unmanufactured.	Total.
1894.....	265	240	$4,300	$18·91	$ 15,989	$240,029	$256,018
1895.....	664	602	11,837	19·66	19,731	225,147	244,878
1896.....	716	650	12,670	19·49	15,654	229,084	244,738
1897.....	840	762	12,950	16·99	10,570	264,220	274,790
1898.....	885	803	13,425	16·72	12,899	287,636	300,535
1899.....	912	827	13,860	16·76	8,949	303,119	312,068

Canada (By J. Obalski).—Asbestos has been worked as usual in the districts of Thetford and Danville with a reopening of two mines at Black Lake, during the summer months. The demand is good and the prices are about the same, the quantity produced having been of 23,266 tons of a value of $598,736, including 7,695 tons of asbestic. The quantity shipped is about 2,500 tons larger than the production, probably on account of some stock left in hand last summer.

The total value of asbestos shipped to the United States and Europe, as per custom's returns during 1899 was 14,520 short tons, valued at $453,176.

Tasmania.—The Australian Asbestos Co. was formed during the year to work the mineral formation near Beaconsfield. Development work only was carried on.

A. Turchenetzky* has patented a method of manufacturing fireproof decorations, by treating asbestos scrap with a solution of alkaline silicate of soda, the colloidal silicate uniting the threads of asbestos. A dilute solution of an alkaline silicate is used, and the mass is then immersed in a saturated solution of alkaline bicarbonate. This deposits the silica between the asbestos particles.

* United States Patent No. 631,719; English Patents Nos. 4,628 and 5,954, 1898.

ASPHALTUM.

ALL mineral bituminous products are treated under this heading, as in previous volumes of THE MINERAL INDUSTRY.

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(In tons of 2,000 lb.)

States.	1897.			1898.			1899.		
	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.	Tons.	Value.	Per Ton.
Bituminous sandstone:									
California	a45,470	a$128,178	$2·82	a 46,836	a$137,575	$2·93	a 40,321	a $116,097	$2·70
Kentucky	2,000	8,000	4·00	2,700	8,700	3·22	1,783	7,132	4·00
Total	47,470	$136,178	$2·87	49,536	$146,275	$2·95	42,104	$123,229	$2·75
Asphaltic limestone:									
Utah	2,140	10,700	5·00	(c) 150	750	5·00	250	1,250	5·00
Indian Territory	250	750	3·00	13,949	69,745	5·00	10,128	53,172	5·25
Total	2,390	$11,450	$4·79	14,099	$70,495	$5·00	10,378	$54,422	$5·24
Asphaltum:									
California	a27,397	a486,620	17·76	a25,690	a 482,175	18·77	a 15,060	a 308,130	20·46
Grahamite:									
Utah	1,756	52,680	30·00	b 2,675	80,250	30·00	3,150	97,650	31·00

(a) Statistics of the California State Mineralogist. (b) Including 115 tons produced in Colorado. (c) Estimated.

The Asphalt Co. of America was incorporated during the year to take over the property belonging to the United Asphalt Co., the Barber Asphalt Paving Co., the New Trinidad Asphalt Co., Ltd., of London, and the Alcatraz Asphalt Co., operating in California. The Venezuelan production is also controlled by this corporation, which is virtually a trust, controlling, it is said, 95% of the companies producing crude asphalt. The head office of this company is in Philadelphia and the capitalization is $30,000,000.

Alaska.—It is stated that an important discovery of asphalt or bituminous rock has been made on Mary Island, off the Alaskan Coast. Interested parties claim that owing to its accessibility and the cheap transportation rates that are obtained, it will become a serious competitor to the Californian asphalt companies.

California.—The extensive properties of the Alcatraz Asphalt Co. are situated in Santa Barbara County, and occupy 3,500 acres. The largest property, known as the Mesa Deposit, situated 1,900 ft. above sea level and 25 miles from the coast, is said to extend a mile in length, a quarter of a mile in width, and to have a minimum depth of 125 ft. It is capped by a small amount of earth only, and is estimated to contain between 20,000,000 and 30,000,000 tons of bituminous rock, which will yield some 4,000,000 to 5,000,000 tons of pure bitumen. This company also controls deposits of bituminous sand on the seashore of Carpenteria, and mass asphalt deposits 12 miles west of Santa

Barbara. The plants of this company have a capacity of 50,000 tons of bitumen annually.

(By A. E. Heighway.)—The largest producer of asphalt for paving purposes still continues to be the Alcatraz Asphalt Co., with mines in Santa Barbara County and a refinery at Alcatraz landing on the coast. Its mining and refining methods are novel. This company owns and controls all the deposits of asphaltum on the Sisquoc ranch in Northern Santa Barbara County, 29 miles from the ocean at 2,000 ft. elevation. The asphalt deposits are large and in the form of bituminous sands. The asphalt is mined, lifted by an electric crane and deposited in steel cars and is then conveyed by steam and electric power 1,000 ft. to the extracting works on the side of the mountain. The works are so located that all material moves by gravity. The crude rock delivered at the extracting works is dumped into a large hopper, from which it passes through a series of large steel crushing rolls, thence into large revolving steel jacketed heating drums; the mineral as it becomes soft by heat is met by an inflow of gasoline and when thoroughly saturated passes into other revolving drums which act as agitators. The asphalt in solution is now passed into a 5-in. steel pipe and flows by gravity to the refining plant. It is then delivered into the stills and carefully subjected to even heat, which evaporates the gasoline. The asphaltum is then drawn into cooling chambers preparatory to barreling. The gasoline on condensing is pumped back to the extracting works. The asphaltum, 99% pure, is stored for shipment.

The American Oil and Asphalt Co., Francklin Oil Co., and the Western Oil and Asphalt Co. are new companies formed to refine crude petroleum and extract the asphalt, which is used mostly for varnishes, pipe covering, dips, wire insulations, roofing paper and in place of coal tar for roofs. There is a limited demand for the distillate which is used for a tree wash, fuel, or gas engine purposes and stock for lubricating oils. Owing to the increased production of oils in Southern California, the American Oil and Asphalt Co. is at present enlarging its works to three times their present capacity; when completed will have a capacity of 1,000 bbl. per day. The total production of California for the year is valued at $375,000.

There is a large area of bituminous rock of low grade in Santa Cruz County, occurring in horizontal masses, though vein-like deposits are also found. At the City Street Improvement Co.'s mines 12 miles west of Santa Cruz, there is 10 ft. of bituminous rock lying upon sandstone and covered by shales. The Gilson Asphaltum Co., of St. Louis, has secured control of the Buena Vista asphalt field near Bakersfield, Cal. A railway branch is to be built from the Southern Pacific Railroad to the mine and a refinery has been erected at Madison, Ill.

Indian Territory.—The production of asphaltic sandstone and limestone fell off during 1899. The total shipments were 10,128 tons. The Elk Asphalt Co., of Kansas City, Mo., is developing a property near Davis, and promises to make large shipments during the present year. At Impson Valley, in the reservation belonging to the Choctaw nation, a mineral resembling albertite (although insoluble in turpentine) has been found in thin stringers in decomposed shale. As yet the extent of the deposit is not known.

Kentucky.—Although several new companies with headquarters in Louisville and St. Louis were formed during the year, no change has been made in the situation, the mines remaining closed down, owing to the opposition of the Barber Asphalt Co., and other corporations interested in the production of asphalt.

Utah.—The gilsonite claims, some 60 miles from Vernal, Utah, were purchased by Omaha parties during the year. It is stated that the properties will be worked actively. Of the 3,015 tons of grahamite produced in 1899, the greater portion came from this State. The production was a slight increase above that of former years.

West Virginia.—I. C. White* described the grahamite of Ritchie County, W. Va. This fills a vertical fissure about a third of a mile long, varying in width from a few inches at the terminals, to 4 and 5 ft. in the middle. It is at right angles to the Burning Springs-Eureka anticlinal which crosses the formation about 7 miles west of the deposit, and is doubtless genetically connected with it. Lesley, who described the deposit in 1863, asserted "this gash was once an open fissure, communicating with some reservoir of coal oil (petroleum) which still, it may be, lies beneath it undisturbed." This has been verified by striking at a depth of 1,500 to 1,600 ft. in close proximity to the grahamite fissure, a continuation of the Cairo oil field. Mr. White points out that the fissure made by tension from the Burning Springs-Eureka uplift, was filled with petroleum from this pool and subsequently the oil was gradually converted into grahamite by oxidation from the infiltrating water at the normal temperature. He forms a conclusion moreover, that such minerals as albertite, gilsonite, uintaite and practically all forms of asphalt, including the pitch coal of Coos Bay, Oregon, and the Trinidad asphalt deposits, are oxidized petroleum. The latter deposit he considers an oil pool uncovered by erosion and hardened by volatilization and oxidation.

ASPHALTUM IN FOREIGN COUNTRIES.

Trinidad has maintained its supremacy in the production of asphalt as there has been no deposit discovered elsewhere of equal quality to compete with it.

Cuba.—The asphalt deposits, described in THE MINERAL INDUSTRY, Vol. VII., have not been exploited and by those who have seen them recently are stated to be of very poor quality and inaccessible. It is not anticipated that much development will take place in this direction.

France.—In 1898 22 asphalt mines produced 182,000 tons of bituminous schist, 36,000 tons of asphaltic limestone, and 11,000 tons of bog-head, making a total of 229,000 tons with a value of 1,669,000 fr.

Italy.—Italy still produces a fair amount of bituminous rock, some of which is consumed locally; a considerable amount is shipped to Germany and the rest to foreign countries, some coming to the United States.

New Zealand.—A mineral which has been identified as ambrite is found in the coal seams of the North Island in New Zealand. It has no commercial

* *Bulletin of Geological Society of America*, No. 10, pp 277, 284.

value and is left in the coal, which, however, it assists in its calorific effect. It resembles a mineral described by Stanek as reussinite. Its origin is doubtless the same as the kauri gum, and its qualities are the same, due allowance being

THE WORLD'S PRODUCTION OF ASPHALTUM AND ASPHALTIC ROCK. (*a*) (IN METRIC TONS.)

	Austria.	France.	Germany	Hungary.	Italy.	Russia.	Spain.	Trinidad. (*e*)	United States.		Venezu'la (Bermudes). (*e*)
	Asph. Rock.	Asph. Limestone. (*b*)	Asphaltum.	Mineral Pitch.	Asph. Rock.	Asphalt. (*c*)	Asph. Rock.		Asphaltum. (*f*)	Asphaltic Rock. (*g*)	
1894..	116	33,374	55,961	2,370	60,493	16,054	985	109,933	10,612	31,018	7,151
1895..	404	39,891	59,563	2,284	46,713	15,783	790	92,862	23,156	42,976	3,073
1896..	390	34,394	61,559	2,740	45,456	22,192	1,117	100,400	18,519	39,531	6,197
1897..	300	30,946	61,645	3,057	55,389	22,222	1,656	133,310	24,854	45,283	11,598
1898..	643	36,000	67,649	3,125	98,750	(*d*)	2,354	108,792	23,306	57,728	*Nil.*

(*a*) From the official reports of the respective countries, except where noted to the contrary. The production of grahamite in the United States and Manjak in Barbadoes is not included (*b*) France produces a large amount of bituminous shales, used for distilling oil, which is not included in these statistics. (*c*) The statistics for 1895 and 1896 are official returns furnished by the Department of Mines, St. Petersburg, to C. Le Neve Foster for his Blue Book on Colonial and Foreign Statistics for 1897. (*d*) Not yet reported. (*e*) Exports (crude equivalent) reported by The New Trinidad Lake Asphalt Co. (*f*) Statistics reported by the California State Mineralogist, the entire American product being derived from California. (*g*) Statistics based on direct reports from the producers, including asphaltic limestone and sandstone.

NOTE.—There is a considerable production of asphaltic stone in Switzerland of which no account is taken in the above table, the Swiss government not publishing any mineral statistics.

made for the pressure it has undergone. An analysis gave C, 80·95%; H, 9·87%; O, 9·18%; its formula would probably be $C_{16}H_{26}O_2$.

EXPORTS OF LAND ASPHALT FROM TRINIDAD, IN TONS OF 2,240 LB.*

Year.	To United States.			To Europe.			To Other Countries.			Grand Total of Exports in Crude Equivalent.
	Crude.	Épuré.	Total Equivalent in Crude.	Crude.	Épuré.	Total Equivalent in Crude.	Crude.	Épuré.	Total Equivalent in Crude.	
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1895....	4,445	199	4,744	1,770	2,368	5,322	.....	169	254	10,320
1896....	11,943	71	12,049	842	1,988	3,924	.....	.....	.....	15,973
1897....	19,243	*Nil.*	19,243	238	700	1,343	415	178	682	21,268
1898....	18,160	*Nil.*	18,160	700	258	1,067	404	312	872	20,119
1899....	25,618	345	26,130	275	280	695	.....	100	150	26,975

EXPORTS OF PITCH-LAKE ASPHALT FROM TRINIDAD, IN TONS OF 2,240 LB.*

Year.	To United States.			To Europe.			To Other Countries.			Grand Total of Exports in Crude Equivalent.
	Crude.	Dried.	Total Equivalent in Crude.	Crude.	Épuré and Dried.	Total Equivalent in Crude.	Crude.	Épuré and Dried.	Total Equivalent in Crude.	
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1895....	61,702	2,256	64,976	5,058	7,365	16,104	.....	.....	.....	81,080
1896....	60,637	.....	60,637	8,320	8,052	20,391	.....	1,300	1,918	82,946
1897....	71,969	1,769	74,407	14,029	13,510	34,856	.....	500	690	109,943
1898....	46,069	1,692	48,424	15,708	13,228	35,537	698	1,646	2,999	86,960
1899....	70,111	480	70,777	21,837	13,749	41,956	.....	1,699	2,359	115,092

* The exports prior to 1895 will be found in THE MINERAL INDUSTRY, Vol. VII.

Switzerland.—The asphalt deposits of the Val de Travers in the Canton of Neuchatel were discovered over 200 years ago, though productive work has been confined to this century. Bituminous rock is here quarried from 13 to 26 ft. thick. This is ground fine and marketed.

Trinidad.—(O. E. Thurber.)—As to the progress of the industry during the last year, I cannot say that there is very much to report: the figures as you will notice show a considerable increase over those of 1898 both in Trinidad

Lake Asphalt, Trinadad Land Asphalt, and Bermudez Asphalt. The increased shipments for 1899 no doubt are largely due to the general revival of business and to the growing demand in various cities for asphalt pavements. The amount of work in prospect for 1900 gives every reason to anticipate that the consumption of asphalt will fully keep pace with 1899.

Venezuela.—Mr. O. E. Thurber, of the New Trinidad Lake Asphalt Co., Ltd., reports to us the shipments of asphalt from the State of Bermudez, as follows: Previous to 1891, 41 tons of 2,240 lb.; 1891, 250; 1892, 1,131; 1893, 1,743; 1894, 7,038; 1895, 3,025; 1896, 6,099; 1897, 11,346; 1898, *nil;* 1899, 11,825. All of this material has come into the United States with the exception of 950 tons in 1893, which was lost at sea.

TECHNOLOGY AND USES.

The only new processes in the refining of asphaltum* which appear to have any merit are those of A. S. Cooper,† the present State Mineralogist of California, and A. E. L. Bell. Cooper's process consists in utilizing the solvent powers of crude petroleum which partially dissolves bitumen from the sand. This treatment is then followed by agitation with benzine, which dissolves the entire bituminous contents. The solvent is then evaporated, or distilled from the asphalt, and returned to treat fresh quantities of material.

Bell has patented a device for revolving and mixing asphalt or bituminous rock with the solvent. The apparatus is provided with a helical vane to move the material longitudinally, and longitudinal vanes between the convolutions of the helical vanes to further agitate and raise the asphalt rock. It is steam jacketed and provided with a special feed way for the introduction of the rock and solvent.‡

Asphalt Paving.—Experience in Berlin has now shown that coal tar or petroleum residue may be successfully utilized in making pavements. Artificial paving material is made by mixing and heating tar with sulphur, followed by an addition of chloride of lime to the semi-liquid mass, which, after cooling, is crushed and mixed with crushed blast furnace slag. The powder is then compressed at a high pressure to the form required. The pavement is said to be durable, noiseless and to give a sound foothold for horses. Lohlendorfer's process is one for solidifying petroleum. In this 10% of soda lye and 10% of fatty matter is treated in a boiler with enough petroleum refuse to make a total of 100 parts. The mixture is heated for one hour at a temperature below the boiling point of petroleum. The partially saponified mass has the property of taking up large quantities of crude petroleum. The mixture is then run into moulds and allowed to cool. It may be afterward cut into pieces of any desired shape.

Although brick pavements are being introduced in many places in competition with asphalt, a certain amount of the latter material is used in filling the interstices between the bricks. In Indianapolis, a street paved with wooden blocks was treated with asphalt. The interstices were filled with a tar and asphalt mixture and a small amount of asphalt covering was laid. When

* Refer to the thorough article by Hans A. Frasch in THE MINERAL INDUSTRY, Vol. VII.

† United States Patent No. 617,296.

‡ United States Patent No. 617,712.

hardened a second coating was put on and rolled and smoothed in the ordinary manner.

A. W. Dow* points out that asphalt cement is properly a liquid, or physically answers to the condition of liquidity. In experiments conducted by him a smooth block from a Washington pavement, placed directly over the small hole in a board, when subjected to temperatures of 26°, 75° and 140°F. flowed perceptibly. Mr. Dow's experiments showed, moreover, that the quality of the sand used had much to do with a paving mixture. While it was impossible to make a hard mixture with rounded sand particles, the general deduction was made that the sharper the sand grains, the harder the pavement.

Analysis of Asphalt.—As the sulphur contents of asphalt have a material bearing on its economical use, considerable attention has been paid to determining this element in bituminous products.

E. H. Hodgson has made a comparison of Carius' method, that by deflagration, the sodium peroxide method, and Eschska's scheme using calcined magnesia, with the result that the first named process of employing a sealed tube gave uniformly the best results on asphalt of varying compositions. The details of Carius' method were as follows: 0·5 g. of the finely ground asphaltum is placed in a glass tube with 15 c.c. of fuming nitric of sp. gr. 1·60; the open end of the tube is drawn out but not sealed, and the tube is heated in a water bath from 80 to 90°C. for 6 to 8 hours, then an additional 5 c.c. of acid is added and the tube sealed and heated in a furnace for 4 to 5 hours at about 150°C. The tube is then allowed to cool and opened, resealed and reheated to 180 to 200°C. for 2 to 4 hours; then finally cooled, opened, and the liquid diluted and filtered. If the residue contains anything but sand, it is treated by fusion with 6 g. of mixed carbonates and 1 g. of potassium nitrate. The solution from this fusion, after treatment with boiling water and filtration is added to the first solution. A piece of caustic soda is added to the filtrate, and it is evaporated to dryness with hydrochloric acid to drive off the nitric acid. After the silica is removed, the liquid is treated with barium chloride as usual. The percentages of sulphur recovered from the undecomposed residue rose at times as high as 0·54%. The method, however, is troublesome, and these investigations were taken up later by S. F. and H. E. Peckham.† The method finally adopted, which gave thoroughly concordant results was as follows: 0·5 g. of bitumen is mixed with 16 g. each, of dry sodium carbonate and potassium nitrate. The salts are thoroughly pulverized and mixed in an unglazed porcelain mortar; two-thirds of the mixture is removed to a sheet of glazed paper, the remaining salts in the mortar are then thoroughly mixed with the sample taken for assay, and removed to a second sheet of glazed paper. The mortar is then cleansed with the remaining flux and the whole mixed thoroughly together. It is charged little by little, into a 2-oz. platinum crucible and heated to dull redness. A blast lamp is not required, as fusion takes place easily. The contents of the crucible are dissolved in water by placing it in a beaker over night. The crucible is then washed out and the contents of the beaker acidified with hydrochloric acid. The solution is then evaporated to dryness, the silica is dehydrated, moistened with hydrochloric acid, treated with water, and the

* *American Gas Light Journal*, No. 69, p. 862. † *Journal of the American Chemical Society*, September, 1899.

silica filtered off and weighed after ignition. The solution is rendered alkaline with ammonia, the excess of that reagent being removed by boiling, and the iron and alumina precipitate dried and weighed in the usual manner. The iron can be determined in another portion by potassium permanganate and the alumina by difference. Lime may be determined as oxalate. The solution now freed from silica and bases, is boiled, acidulated with hydrochloric acid and the sulphuric acid precipitated by barium chloride added in small quantities from a pipette. When care is exercised, this method for sulphur is very accurate.

The Occurrence and Genesis of Californian Asphalt.*

By A. S. Cooper.

Veins of asphaltum consisting of rents, seams and fissures, filled with asphaltum, are usually nearly vertical, and occur in the vicinity of shales which have been burned, contracted and reddened by chemical heat. These fissures are filled with bitumen, owing to the petroleum being forced upward, principally by rock pressure. The subsidences near which asphalt is usually found are caused by the contraction of the shales by heat.

The oil is partly evaporated, the evaporating portion forming gas reaching the surface through passages which are closed to the viscous oil. On account of its divided state the oil is readily oxidized and in the presence of hydrogen sulphide or sulphurous anhydride is resinified, forming asphaltum.

Adjoining the selvage of the veins, when near the surface of the earth, the substance is brown and friable, fully two-thirds of which is asphaltum. In the interior of the vein it is black and shining, and, when cold, breaks with a conchoidal fracture, and is fully two-thirds petrolene. The proportional amount is very variable. When at a sufficient depth, so as to be removed to a considerable degree from atmospheric influences, the brown margin of asphaltum does not exist. As a general rule, the farther from atmospheric influences, the more liquid the asphaltum will be. Bivalvular shells, filled with petroleum, are frequently found in these asphalt veins, the bitumen on the shells being much more liquid than the surrounding asphalt, and of a lighter color. The asphalt veins are parallel to the periphery of the red shale deposits, and exist along the margin of the subsidences, the seams for the reception of the asphalt having been opened by the movement of the formation toward the contracting shales, permitting the ascent of the asphalt, forced up by rock pressure. The parallelism of the veins to the periphery of the burnt shales is frequently changed by the varying hardness of different strata, or by orogenic contortions.

When the hanging walls consist of hard and close-grained strata, the bitumen will accumulate below the same, which on account of its being conformable with the plane of bedding of the adjoining strata, gives it the appearance of having been deposited at the same time as the other strata. When these shales are burning, large amounts of sulphur vapors are disengaged. It is reasonable to believe that these vapors sulphurized the bitumen in their neighborhood, forming asphaltum.

* Bulletin No. 16, California State Mining Bureau, December, 1899.

The demand for *blanc fixe* (barium sulphate artificially produced) was good, and prices in New York in 1899 varied from 1· to 2·75c. per lb., depending upon quantity and seller.

Manufacture of Baryta.—Caustic baryta is usually obtained by means of decomposing barium nitrate at about 1,200°C. This process is not only the most rational, but also one that yields the purest product, a very important fact when it is intended to produce barium peroxide.

The decomposition of barium nitrate is due to the following reaction taking place:

$$Ba(NO_3)_2 = BaO + N_2O_5.$$

But this reaction is purely theoretical. In practice the nitrous fumes are decomposed and lost, which makes this method expensive. For this reason patents have been obtained for several new processes.

One of these is dependent upon the decomposition of barium carbonate by carbon at a moderate temperature; and another on the decomposition of barium hydrate by carbon at a temperature of about 1,200°C.

This last process is preferable, as barium hydrate is less costly. Barium hydrate may be heated to 100°C. or 150°C. only, kept there for some time to drive off the water of crystallization; but to eliminate the last water a very high temperature is required. For this purpose a mixture of barium hydrate and of powdered coke, after being heated in cast-iron enameled vessels, is placed into a clay or graphite crucible, heated in a furnace to a temperature of 1,200°C., and kept at this temperature for 8 or 10 hours. Caustic baryta is thus obtained in the form of a grayish porous mass, which is easily transformed into barium peroxide.*

Determination of Barium Sulphate.—Geo. O. Marrs has devised a process for the determination of barium sulphate in ores. One gram of finely ground ore is treated with hydrochloric, nitric and sulphuric acid and evaporated until fumes of sulphurous anhydride are evolved. The residue consists of silica, barium sulphate, and of lead sulphate, if lead is present in the ore. The lead sulphate is removed by washing with a hot ammonium acetate solution. The mass is filtered and incinerated with the filter paper without being previously dried. After ignition the matter is brushed into a platinum crucible and fused with mixed sodium and potassium carbonates; the mass is then treated with hot water, then by acetic acid when on a filter, diluted up to 200 c.c. and titrated with a standard solution of potassium bichromate (4·4 g. per liter) using silver nitrate as an indicator. The indicator is prepared and used as follows: A little sodium bicarbonate is placed in each hole of the spot plate and a moderately strong and neutral solution of silver nitrate is dropped on to the soda, when bichromate solution is run into the solution about to be tested. When a drop of the solution is placed upon the sodium bicarbonate, a perceptible red color appears when an excess has been applied. If a great excess is present a dense purplish red color appears. The great advantage in this method is that much time is saved, only a few minutes being required after the fusion.

* *Le Mois Scientifique et Industriel*, June, 1899.

BISMUTH.

THERE was no recorded production of bismuth in the United States during 1899, but the Ballard mine at Leadville, Colo., made several shipments of ore to England. As stated in previous volumes of THE MINERAL INDUSTRY, the production of this metal is controlled by a ring composed of Johnson, Matthey & Co., Ltd., and the Saxon Government, which allots to each country its production and fixes the prices for the ore. When bismuth ruled at $1·25 a pound the following prices were paid on delivery at the works, all deductions for treatment having been made: 10% ore, $150 a ton; 15% ore, $250; 20% ore, $350; 30% ore, $550; 40% ore, $750; 50% ore, $1,000 a ton.

The Jingera Mineral Proprietary mine, near Pambula, New South Wales, produces bismuth ore associated with silver, and furnishes nearly the whole production of New South Wales.

The chief European mines producing bismuth ores are those of Schneeberg, Altenberg, Annaberg and Johann-Georgenstadt in Saxony. The bismuth occurs most frequently in the native state although sulphides and carbonates are also found. In Bohemia ores of bismuth are found associated with those of silver near Joachimstahl. The ore from here is exported to Saxony for treatment. The deposits of Tasna and Chorolque in Bolivia are somewhat important. The bismuth occurs in the native state and as ocher and carbonate. The ore exported runs from 20 to 30%. It is reported that bismuth is found in commercial quantities on the Samosin peninsula, near Lake Toba in Sumatra. Bismuth has been found in several localities in Tasmania. The silver copper ore of the Curtin-Davis mine is said to contain up to 5% of that element, for which returns are not made by the smelters.

J. Ranald* has patented a process for recovering bismuth from its sulphides. The ores after pulverization are treated with a boiling solution of ferric chloride by which the soluble chloride of bismuth is formed. From the solution the metal is precipitated by iron and ferrous chloride is formed. The solution can be regenerated by electrolysis and re-used. It is claimed that the sulphur which is precipitated, can be recovered from the residue.

C. Reichard† has devised a volumetric method of estimating bismuth in ores or its compounds. After solution an excess of alkali is added and the liquid treated with chlorine water or gas or the oxidation may be effected by the direct addition of the bismuth solution to a mixture of sodium hypochlorite and caustic soda. The liquid is boiled until the precipitate changes from an orange to a dark red color. The excess of soda and chlorine is then removed by washing

* English Patent No. 10,022, July 30, 1899.

† *Zeitschrift für Analytische Chemie*, Vol. XXXVIII. No. 2, p. 100.

by decantation. The precipitate is boiled in a solution of arsenious acid in caustic soda which contains 0·01 g. of As_2O_3 per cm. until it is completely converted into the white hydrate $Bi_2(OH)_6$. After complete reduction the liquor is acidified with excess of sulphuric acid and filtered hot and the residual arsenious acid in the filtrate is titrated with permanganate solution. As the reduction takes place slowly it is advisable to work with small quantities.

Notes on the Metallurgy of Bismuth.

BY W. BORCHERS.

Bismuth occurs native, also in the form of oxide, as bismuth ocher; sulphide, as bismuth glance, and in other minerals where it plays a subordinate rôle. Certain metallurgical products are also important for the extraction of bismuth, among them being crude lead, litharge, hearths, nickel and cobalt speiss.

Extraction of Crude Bismuth.—The liquation process, formerly used, has been superseded by direct smelting owing to the losses in metal in the former process.

Reduction in Furnace.—The reduction process is suited particularly for oxidized ores and the above-mentioned metallurgical products. Sulphides must be subjected to a preliminary roasting to drive off the sulphur and oxygen. The oxychloride is prepared by adding a quantity of slaked lime to take up the chlorine, and after drying it may be then treated as an oxide. Reduction is carried out in small crucible furnaces, or reverberatory furnaces.

(*a*) *Crucible Furnace.*—The first thing necessary in this process is good crucibles. These can be prepared best at the works. Clay containing 60 to 70% silica and 20 to 26% alumina is of the right quality. To this a small amount (not over 20%) of artificial graphite or charcoal should be added. The crucibles have a height of 25 to 30 cm. and are about 38 cm. in diameter at the top. They should be formed by hand and then dried, after which they can be burned in the same furnace that is used for treating the bismuth.

In comparison with the high price of bismuth the cost of fluxes is inconsiderable; it is of prime importance to obtain an easily meltable slag, so as to avoid the loss by volatilization of the bismuth and its compounds, which is usual at high temperatures. The usual fluxes are slags from former operations, soda, limestone and rarely fluorspar. The silica ratio of the slag lies between a singulo- and a bisilicate.

A very suitable form of furnace, even when the operations are small, is shown in Figs. 1, 2, 3, 4 and 5. Fig. 1 is a vertical section through the generator and the three heating chambers; Fig. 2, a horizontal section on the line of the gas channels; Fig. 3, a horizontal section along the plane of the air channels and connecting flues; Fig. 4, a horizontal section below the combustion chambers; Fig. 5, a vertical section through the generators. The heating chambers for the crucibles, of which there are six in this case, are grouped about a channel which conveys the gas from the generator immediately adjacent. Connection is made between the main gas channel and each of the chambers by funnels and a U tube. When a chamber is in operation, the air necessary for combustion is

admitted through one or two of the neighboring chambers, preferably those in process of cooling, where it is warmed. After combustion the gases do not pass directly out, but are made to circulate through such chambers as are in process

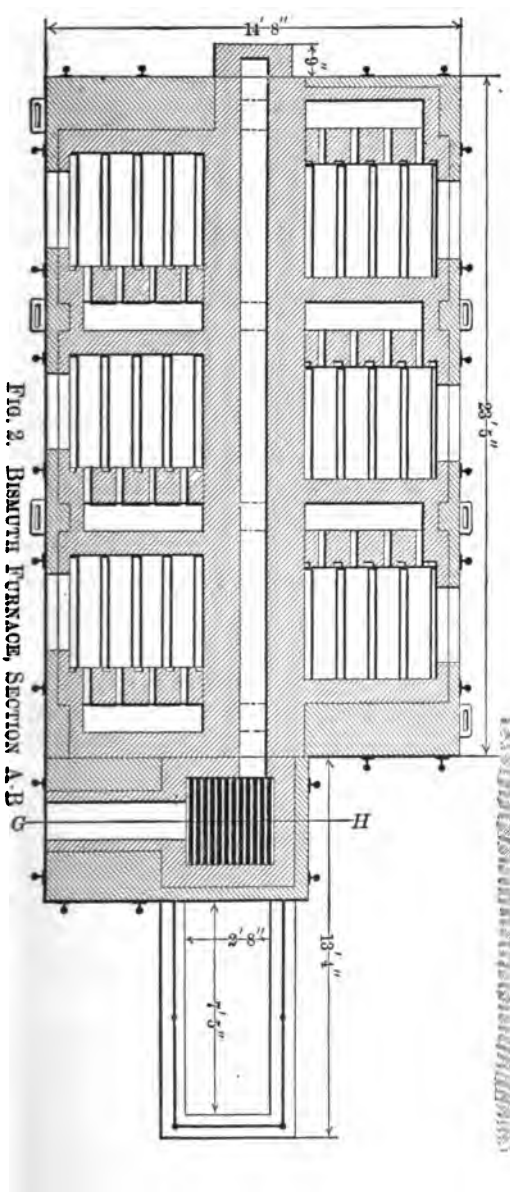

FIG. 2. BISMUTH FURNACE, SECTION A B

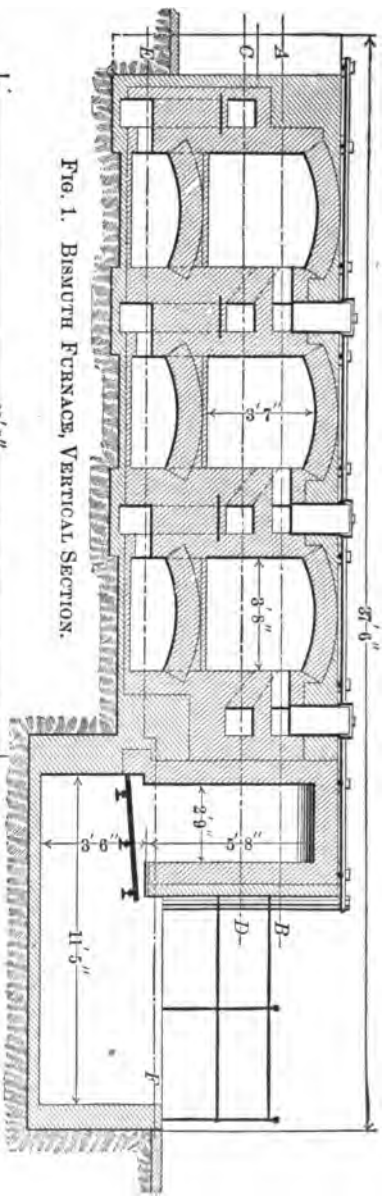

FIG. 1. BISMUTH FURNACE, VERTICAL SECTION.

of warming. By these devices a great saving of fuel is effected, and sudden changes of temperature, which often seriously damage the crucibles, are avoided. The flames which, as can be seen from the position of the gas and air channels, are formed in the upper part of the chambers, play around the crucibles and

then pass out through the grating in the bottom into a receiver. From here they are carried by means of a connecting flue, which is placed as near as possible to the front of the furnace, and by small flues into the second chamber. After passing through this in the same manner they are allowed to enter an outlet pipe underneath which connects the furnace with the chimney.

It is evident that in this system the process is uninterrupted; each chamber in turn is charged, warmed, burned and cooled, while every other chamber is at one step or another of these operations. The advantages of the system are small consumption of fuel, control of heat so that the temperature can be raised and lowered gradually, and the adaptability of the furnace for burning the necessary crucibles.

Before introducing the fluxes into the crucibles they should be thoroughly mixed. The mixing may be done on a hard floor, preferably one of cement. A part of the slag is first spread out and then over this layers of soda, charcoal, ore and the other fluxes, and again a layer of slag and soda, the whole forming a truncated pyramid. Beginning at one end the mass is next shoveled in vertical sections into a heap in such a way that the material is spread evenly over the surface. After repeating the operation once more the ore and fluxes will be thoroughly mixed. Before charging the crucibles should be sprinkled with a little pulverized slag. The charge is covered with a thin layer of slag, soda and charcoal. The reduction is accomplished in a crucible furnace in the manner previously described.

(*b*) *Reverberatory Furnace.*—Success in operating a reverberatory furnace depends in a great measure upon the proper construction of the hearth-floor. The strong solvent power of slags rich in alkali silicates renders it necessary to give special attention to this point. In the first place the hearth should be isolated from the flue and fire-box walls by means of air or water chambers. If not so isolated these places, being heated from both sides, offer most favorable conditions for the leakage of the bismuth, and the loss of the latter will be very great. The fire-bridge should be made hollow so as to allow the circulation of air or water. If the waste gases are to pass beneath the furnace before reaching the chimney, the flue should be independent of the furnace walls. It is well to elevate the hearth slightly, say 500 to 600 mm. above the ground, by means of iron rails laid on piers.

After a furnace has been in operation for some time the mortar in the walls is eaten away, its place being taken by masses of bismuth which, after a while, will begin to trickle down through the bottom and escape. If operations are continued much longer there will be no metal to tap off, the leakage being sufficient to allow it to escape as fast as formed. It is then necessary to stop working and to tear up the hearth. Formerly the entire furnace had to be torn down, as the walls and arch rested upon the hearth—a very expensive operation in both time and money.

The writer has introduced some modifications in the construction of a reverberatory furnace which are worthy of notice. The arch and side walls of the furnace are supported independently of the hearth by iron beams, while the hearth is composed of a single course of stone blocks laid on heavy iron rods.

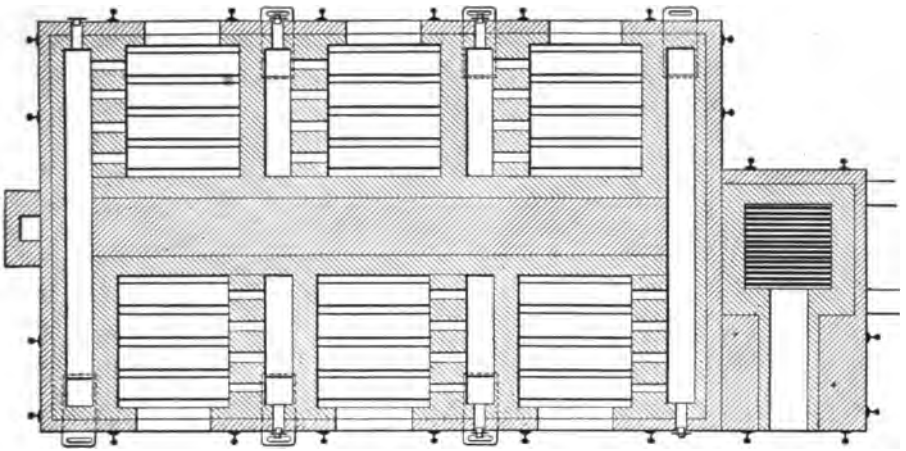

FIG. 3. BISMUTH FURNACE, SECTION C-D

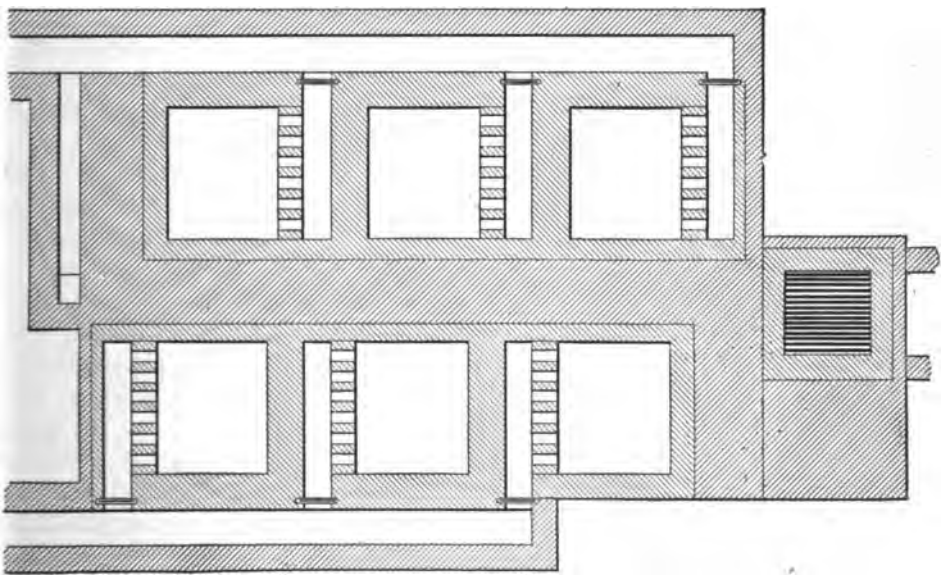

FIG. 4. BISMUTH FURNACE, SECTION E-F

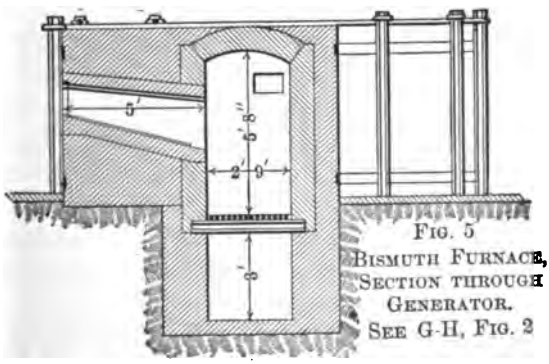

FIG. 5
BISMUTH FURNACE,
SECTION THROUGH
GENERATOR.
SEE G-H, FIG. 2

The hearth is carried on wheels resting upon an inclined track. By this construction, which is shown in Figs. 6, 7, 8 and 9, the heavy mass of stone work in the hearth is avoided, and there is no possibility of the molten metal escaping through the crevices before hardening. At the end of a campaign the hearth may be removed without disturbing the other parts of the furnace, and the stonework renewed. Its simple construction also lightens the labor of cleaning the stone of the bismuth that has collected during operations.

Before charging the furnace, a layer of slag should be spread over the bottom of the hearth and allowed to melt as a protection against volatilization. The charge should be well mixed in the manner previously described. The metal is run off at the tap-hole shown in Fig. 9, which is closed with a clay plug. The slag is allowed to run out from time to time through a tap-hole in the back of the furnace. This is closed with a stone plug, whose surface is coated with a thin layer of clay. The metal flows into iron moulds, where it quickly hardens. If any impurities, as matte or speiss, are carried out with it they can be broken out by a hammer. After crushing into coarse pieces the bismuth is ready for refining.

(*c*) *Precipitation Method.*—This method, which is based upon the same principle that governs the treatment of lead ores by precipitation, is applicable especially to ores carrying sulphides of antimony and arsenic. In addition to the usual fluxes some iron scrap is added to the charge in order to break up the bismuth compound. The silica ratio of the slag lies between a singulo- and a subsilicate. Usually pieces of matte and speiss will be found in the bismuth, as compounds of lead, antimony, arsenic and other metals are found in the ores of this class.

(*d*) *Wet Method.*—The wet method is employed chiefly for extracting bismuth from litharge, a by-product of many lead smelters. The bismuth is first concentrated into as small bulk as possible by repeated oxidation and reduction in a furnace. It is then treated with hydrochloric acid, which changes the oxide of lead and bismuth into chlorides, the former of which is nearly insoluble while the latter is quite soluble. Any lead that may have been dissolved is precipitated by pouring the solution into hot water and may then be separated. The bismuth is precipitated for the most part as oxide by a partial neutralization, collected on a filter and washed; the remainder is treated by the crucible method and the product added to the first to be roasted and reduced.

Refining of Crude Bismuth.—The methods of refining, so far as they have been described in literature, are to be used with caution. They often give good results for certain well-defined conditions, but in other cases may be extremely injurious. The most common impurities which have to be dealt with are lead, antimony and arsenic. The first of these must be gotten rid of at the beginning, as its presence interferes with the successful carrying out of further operations. The bismuth is melted in small iron kettles under a cover of salt, potassium chloride, caustic soda and enough oxychloride of bismuth to take up the lead. The molten mass should be constantly stirred. In from one to three hours the lead will be changed completely into chloride, while a corresponding amount of bismuth will separate from the oxychloride. Antimony is removed in the

same manner, with a flux of soda, potash, and sulphur, which changes it to sodium sulphantimonate. For arsenic the flux should consist of caustic soda

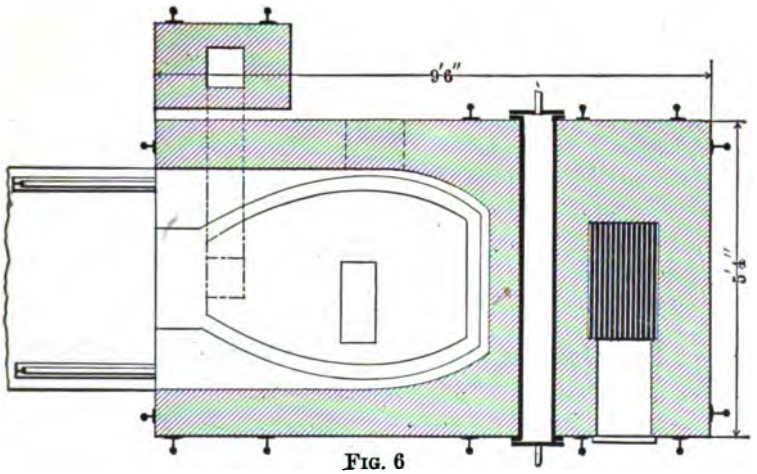

FIG. 6

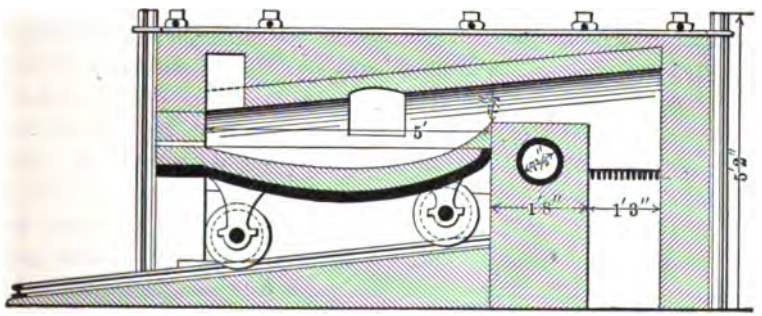

FIG. 7

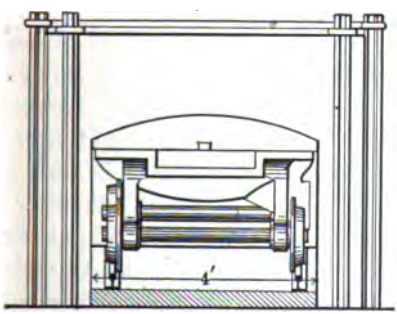

FIG. 8

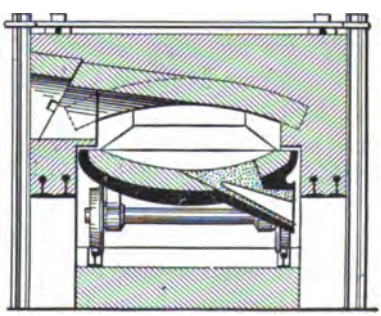

FIG. 9

REVERBERATORY FURNACE WITH REMOVABLE HEARTH FOR BISMUTH.

and saltpeter. While the metal is still molten an iron bar is placed in each kettle. After cooling the slag is dissolved by water and the metal can then be easily removed by means of the bar.

BORAX.

THERE was a large increase in the production of crude borax in the United States in 1899. The greater part of the output was derived from the colemanite deposits of California, smaller amounts being produced by the marsh deposits of California and Nevada. The production of crude borates and commercial borax and boric acid for the American market has, for several years past, been confined almost exclusively to American sources of supply. While this has resulted in a large increase of business it has also caused the reopening of operations on nearly all of the Nevada and California marshes and has led to the discovery of a second deposit of colemanite. This progressive condition has stimulated the invention of processes for extracting boric acid from low grade ores. During 1899 the Borax Consolidated, Ltd., completed its extensive borax and boric acid works at Bayonne, N. J., detailed reference to which is made further on in this review. This company is now operating both this plant and the one at Alameda, Cal., to their full capacity. The colemanite mines near Daggett, Cal., supply raw material to these factories and also to the factory of Charles Pfizer & Co., of New York, and the boric acid works of the Stauffer Chemical Co., San Francisco, Cal. The output of raw material is nearly 100 tons per day, the borate of lime averaging about 30% anhydrous boric acid. On the same deposit, a few miles west, the Western Mineral Co. and the Columbian Borax & Chemical Co. have large deposits of clay containing boric acid—low grade material with calcium borate. The former company has been extracting the acid for the past three years and now has a considerable output which is being increased. Sulphuric acid is used to free the boric acid from the lime, and a plant for the manufacture of the latter is now being erected at the works four miles north of Daggett. The Columbian Borax & Chemical Co. has recently obtained control of 12 borax claims six miles N. W. of Daggett and is erecting a plant for the manufacture of boric acid in that town. The process of manufacture has not yet been made public. About 65 miles west of Lancaster, in Ventura Co., Cal., a ledge of high grade colemanite and pandermite has been opened during the past year. Its present production is about 150 tons per month. This ore is manufactured into borax by Thomas Thorhuldsen & Co., of Chicago (who also manufacture the output of the Western Mineral Co.) and into boric acid by the Stauffer Chemical Co.

of San Francisco. The Pacific Coast Borax Co. has recently built a railroad from Daggett to its colemanite mines at Borate, a distance of 12 miles. At Marion, midway between the two points, a mechanical reverberatory roasting plant of a capacity of 75 tons of raw ore per day has been constructed to handle the low grade ores which hitherto have been left on the dump at the mines. Colemanite possesses the property of flying into minute particles when heated and, by a simple bolting process, the valuable contents of the ore in the form of a fine powder, is removed from the waste lumps. This plant has been idle for a year, but improvements have been made that will probably insure its continuous operation for some time to come.

There is a small amount of borax produced in Oregon. On Lake De Garmo in Lake County, there is a layer of borax varying from a few inches to 3 ft. thick underneath the salt, which evaporates out when the lake dries up during the summer months. It is purposed to work this and other adjacent lakes on a larger scale during the present year.

Borax Consolidated, Ltd., the formation of which was described in THE MINERAL INDUSTRY, Vol. VII., had a most profitable year's business; £243,000 was netted, or £76,000 more than was estimated in the prospectus; 12·5% was paid on the ordinary shares, while £50,000 was placed to reserve account. The chairman, at the annual meeting of the company, stated that the company had entered into a contract to supply nearly all of the refiners of borax and boric acid. The demand for boric acid and borax is on the increase, as new industrial uses are being found for them, and as yet no substitutes have been discovered.

Statistics.—The production of crude borax in 1899, chiefly colemanite or calcium borate, was 24,068 short tons, an increase of 7,733 tons over the production in 1898. The Borax Consolidated, Ltd., furnished almost the entire output and the value is estimated at a nominal price of $21 per ton of 2,000 lb.

THE WORLD'S PRODUCTION OF BORATES, ETC. (*a*) (IN METRIC TONS.)

Year.	United States. Calcium Borate.	Chile. Calcium Borate. (*b*)	India. Borax. (*b*)	Germany. Boracite.	Italy. Boric Acid, Crude.	Peru. Calcium Borate. (*b*)	Turkey. Pandermite (*b*) (*c*)
1894.....	5,950 (*d*)	6,700	367	176	2,746	800	9,100
1895.....	6,126 (*d*)	4,532	400	150	2,633	4,000	9,061
1896.....	12,310	7,486	340	184	2,616	1,179	12,626
1897.....	17,600	3,168	280	196	2,704	11,850	11,875
1898.....	13,911	7,034	184	280	2,650	7,178	(*e*)

(*a*) From official reports of the respective countries except the United States. (*b*) Exports. (*c*) Fiscal years. (*d*) Product of refined borax. The manufacture of boric acid was begun in the United States in 1896, in which year there was a production of 621,000 lb. There are no statistics for subsequent years. (*e*) Statistics not yet available.

Argentine Republic.—A colemanite property in the province of Salta with an area of 2,224 acres was acquired by the Argentine Borax Co., Ltd., with a capital of £150,000. The purchase price to an intermediate vendor selling at a profit was £100,000, leaving £50,000 for working capital. It is estimated that there is 1,500,000 tons in sight.

PROGRESS IN THE TECHNOLOGY OF BORAX.

Borax Refining at Bayonne, N. J.—The works of the Borax Consolidated, Ltd., at Bayonne, N. J., embody some of the latest features of mill design,

including the driving of the machinery by independent electric motors. The crude colemanite which reaches the works in sacks is crushed on the ground floor and then conveyed to a Griffin mill which reduces it to the fineness of flour. From the mill the pulp is carried by a screw conveyor to the boot of an elevator by which it is raised to the first floor, where it is dropped into a tank capable of holding 100 tons of mineral and water, in which it is dissolved by boiling water. From this tank the solution and residue are drawn off to settling tanks on the second floor, from which the clear solution is run to the crystallizing vats on the first floor, while the sediment is raised by a centrifugal pump to a tank on the third floor. From this tank it is pumped into a filter press under a pressure of 500 lb. per sq. in. and then further compressed to 1,000 lb. per sq. in. The liquor flowing from the filter press is led back to the settling tanks while the refuse cakes go to the dump.

The crystallization vats are made of sheet iron, about 20 ft. long, 6 ft. wide and 6·5 ft. deep. Across the top of these vats are iron bars of 2-in. pipe from which wires about 5 ft. long and 0·25 in. diameter hang into the vat. Upon these wires and the sides and bottom of the tank the borax crystallizes as the solution cools. After the crystallization has gone on as far as possible, the mother liquor is pumped off and run through a storage tank for further use as a solvent. The borax is then broken off the wires and sides of the vat and loaded directly into a tram car capable of holding 1,500 lb. which is lowered into the vat for this purpose. When filled, it is hoisted out and conveyed by overhead trolley to the crushing rolls where the borax is dumped. From these crushers the borax is raised by elevators to the fourth floor, where it is sorted into three sizes, namely: (1) refined crystal; (2) refined screenings, and (3) granulated borax. Numbers 1 and 2 drop directly into bins from which they are sacked and shipped, while No. 3 is conveyed to a dryer and after that to a Cyclone pulverizer. The dryer is an inclined, rotary cylinder through which a current of hot air is passed. From the Cyclone pulverizer the fine material is sucked up by a Sturtevant blower and deposited in dust chambers and apparatus similar to that employed in flour milling. This finely pulverized borax is then barrelled for shipment.

The borax from the bottoms of the crystallizing vats contain more or less impurities, wherefore this product is kept separate from the rest and is reground, redissolved and refined as previously described.

Manufacture of Boric Acid.—Bigot* describes a method of making boric acid by treating 100 parts of calcium borate with 150 parts of an ammonium salt, preferably sulphate, in a closed vessel with prolonged heating. Ammonium borate is first formed which splits up into boric acid and ammonia. The latter is condensed and collected, while the former is dissolved out of the residue by water and crystallized by evaporation. To prepare borax, two molecules of ammonia and one molecule of sodium nitrate are added to two molecules of boric acid. When the mixture has been heated it is dissolved in a minimum quantity of water. Ammonium nitrate is obtained as a by-product.

Estimation of Boric Acid.—R. Schwartz† describes a method of estimating

* Rev. Prod. Chim., 2 (11), p. 163.

† *Chemiker Zeitung*, XXIII., 47, p. 497.

boric acid in calcium borate, devised by Hoenig and Spitz, which is said to be both rapid and accurate. It may be carried out two ways: (1) From 1 to 2 g. of powdered calcium borate are mixed with 5 to 10 c.c. of hydrochloric acid (sp. gr., 1·124) and about 100 c.c. of water. It may be digested in the cold for two hours or heated in a reflux apparatus in a water bath for a quarter to one-half hour. The liquid is filtered and after washing the residue the filtrate is made neutral to methyl orange by addition of quinque normal caustic soda free from carbon. (2) Carbonic acid is passed into the cold neutralized liquid with frequent shaking for half an hour, after which it is closed and allowed to stand two hours with occasional shaking. The liquid is then filtered, evaporated to about 20 c.c. on the water bath, neutralized with hydrochloric acid in the presence of methyl orange and heated to boiling. The solution obtained from the first or from the further treatment as described is made up to 100 or 200 c.c., and 50 c.c. of this solution is mixed with 50 to 60 c.c. of glycerin and made up to 150 to 200 c.c. by water. It is then titrated with quinque normal caustic soda, using phenol-phthalein as an indicator. It is necessary to neutralize the glycerin beforehand by means of caustic soda and hydrochloric acid.

An Iodimetric Method for the Determination of Boric Acid.—It has been found that an acid formed by the action of mannite upon boric acid separates, under certain circumstances, from a mixture of potassium iodide and iodate, a quantity of iodine which is equivalent to the quantity of metaboric acid, BO,OH.

Glycerin usually acts in the same way as mannite in the process of forming boric acid combinations. After experimenting with the method the following scheme is suggested: The borate is dissolved in as little hydrochloric acid as possible, shaken up and diluted to a point when, by adding the mixture of potassium iodide and iodate, every 25 or 30 c.c. of the solution will contain 1 decigramme of boric acid, B_2O_3.

The largest part of the hydrochloric acid present in the solution is then neutralized with some sodium hydroxide testing from time to time the solution with litmus paper, and seeing that the liquid retains sufficient acid to give an acid reaction, with litmus paper. Then from 3 to 5 c.c. of a 40% solution of potassium iodide are added and also from 5 to 10 c.c. of a 5% solution of potassium iodate, an excess of these is needed so that, at the time of the elimination of the iodine, the hydrochloric acid and the boric acid may react one upon another. The iodide freed by the hydrochloric acid is made colorless by the addition of a strong solution of sodium thiosulphate, after this a sufficient quantity of iodine to give to the liquid a slight yellow coloring is added to the solution with shaking. The liquid is then saturated with mannite, about 10 to 15 g. for each 50 c.c. of the liquid, and to this is added a measured excess of a solution of thiosulphate of a known strength (usually 8 or 10 c.c. more than needed theoretically). The solution is then saturated once more, if needed, with mannite and left in a cold place for 40 to 60 minutes. After this the excess of thiosulphate is titrated with iodine.*

* *Chemiker Zeitung*, Nov. 27, 1899.

BROMINE.

THE production of bromine in the United States, including the proportionate amount of bromine contained in potassium bromide decreased during 1899, falling from 486,978 lb. to 433,003 lb. The price, however, increased from 28 to 29c. The production of bromine in the world is still controlled by the associated American producers and by the Leopoldshall-Stassfurt convention, which has several years longer to run.

PRODUCTION OF BROMINE IN THE UNITED STATES.

Year.	Michigan.	Ohio.	Pennsylvania.	West Virginia.	Total.	Metric Tons.	Value.	
							Total.	Per Pound.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.			
1895.....	90,280	152,360	104,647	107,567	394,854	179	$102,662	26c.
1896.....	42,000	212,850	154,600	149,835	559,285	249	143,074	26c.
1897.....	(a) 147,256	124,972	116,967	97,954	487,149	221	136,402	28c.
1898.....	(a) 141,232	106,860	119,998	118,888	486,978	221	136,354	28c.
1899.....	(a) 138,272	82,368	111,150	101,213	433,003	196	125,571	29c.

(a) Including the bromine equivalent of the product recovered as potassium bromide.

Production of Bromine.—(By B. C. Hinman.)—The mother liquors at Stassfurt are essentially solutions of magnesium chloride carrying 0·5% bromine as magnesium bromide. This solution is allowed to descend through a tower packed with cylindrical pieces of clay and meets an ascending stream of commingled chlorine gas and steam, the volume of the steam being sufficient to raise the solution to the boiling point. The amount of chlorine is regulated so as to be present in the quantity just sufficient to liberate all the bromine. Bromine in a gaseous form passes with the steam from the top of the tower and is led through a worm immersed in hot water which causes condensation of the bromine and steam.

The product is caught in receptacles where the liquid bromine sinks to the bottom owing to its high specific gravity, while the water which is a saturated solution of bromine and water is returned from time to time to the still or is sold as a disinfectant under the name of "red water." The present consumption is supplied by running the works one month in the year. When not in operation the mother liquors are allowed to run to waste.

An electrolytic method for the recovery of bromine from the waste liquors of potassium chloride works is described under the caption "Progress in Electro-Chemistry during 1899," later on in this volume.

CALCIUM CARBIDE.

By John B. C. Kershaw.

Production.—*Number of Factories.*—The year 1899 has been marked by a most rapid increase in the number of factories engaged in the production of calcium carbide. Many of the larger factories planned by the great German electrical engineering firms have now entered upon the producing stage of their development, and have commenced to manufacture carbide upon a scale which will compel them to seek markets in all corners of the Continent of Europe. In the last issue of The Mineral Industry a list of 44 places was given, where carbide factories were already in operation or were in course of erection. That list was incomplete, and as regards France especially, a much larger number of carbide works were operating at that date than was supposed. By the aid of the very full details of the present position of the carbide industry in France, given by Guilbert in a paper read before the Buda-Pesth Acetylene Congress in May,* these omissions in last year's list have been remedied, and the revised list shows that the total number of factories—operating or building—is now 93. When one recalls the fact that five years ago the carbide industry did not exist, one is struck by the feverishly rapid rate of development in this new industry. Table I contains a list of these 93 places, grouped under the different countries; a distinction being made between factories in operation, and factories in course of construction.

The more important figures relative to the factories in France have been abstracted from Guilbert's paper, and are presented in tabular form in Table II. These factories are situated in the two mountainous districts of the Haute Savoy and the Haute Pyrenees, and all use water power. The figures show that France alone is in a position to more than double her present output of carbide, without erecting one additional carbide factory.

Outside France, the more important extensions in the industry are occurring at Niagara Falls and Sault Ste. Marie, in the United States; at Chaudière, in Canada; at Rheinfelden, in Germany; at Sarpsborg, in Norway; and at Iajce, in Hungary. Further developments are also promised on the Hardanger Fiord, in Norway, where Messrs. Siemens & Halske, of Berlin, are reported to

* *Zeitschrift für Elektrochemie*, No. 50, 1899.

be fiancially concerned in the utilization of two waterfalls (aggregating 19,000 h.p.) for carbide manufacture.

I.—LIST OF PLACES WHERE CARBIDE FACTORIES ARE NOW IN OPERATION, OR IN COURSE OF ERECTION.

Country.	Factories in Operation.	Totals.	Factories in Course of Erection.	Totals.
France	St. Michel; Sechilienne; Chedde; Lancey, Chapareillon; Epierre; N. D. de Briançon; La Bathie; Bellegarde; Froges; Le Praz; Serres; Giffre; Albas; Arudy; Crampagna; Castelet; Marsac; Salles du Salat;* S. Etienne; La Bastide	21	Lourdes; S. Felix; Villelongue; Bellegarde; Arudy	5
Switzerland	Neuhausen; Vernier; Valorbes; Wynau; Vernayaz; Lauterbach; Thusis; Klosters; Berne; Langenthal; Gampel	11	Hagnek; Gurtnellan; Amsteg	3
Austria	Meran; Iajce; Matrei; Lobbowitz; Sebenico, and three unnamed	8	Petrosseni; Paternion; Lend-Gastein	3
Germany	Rheinfelden (2); Bitterfeld; Frankfort; Augsburg; Lechbruck; Lauffen; Guttstadt	8		
Italy	Terni; Narni; Foligno; St. Martino nr. Ivrea; St. Marcel; Salisano; Lugano	7	R. Velino	1
United Kingdom	Foyers; Bradford; Ingleton; Wood-green	4		
United States*	Niagara Falls (2); Sault Ste. Marie; Holcombe Rock; Hinton; Appleton	6		
Norw'y & Sweden	Sarpsborg (2); Trollhatten; Meraker	4	Ljungan; Brattfors; Kinservik	3
Russia	Zombkowitz; Wiborg	2	Sinnakoski Falls	1
Spain	Barcelona; one on the River Ebro	2		
Canada	St. Catherines; Three Rivers, Quebec	2	Chaudière	1
Belgium	Brussels	1		
		76		17

* Three of these carbide factories are reported to have ceased producing since the Union Carbide Co. has taken control.

II.—HORSE POWER UTILIZED IN FRANCE FOR CARBIDE PRODUCTION.

Works.	Utilized.	Available.	Works.	Utilized.	Available.
Bellegarde	1. 1,200 2. 500	8,000	Giffre	880	
			Froges	275	
Chapareillon	800		Le Praz		15,000
Chedde	2,000	12,000	Albas	1,450	
La Bathie	1. 1,250 2. 1,500	2,500	Arudy		2,000
			Crampagna	600	
N. D. de Briançon	3,000		Castelet		2,500
Epierre	2,600		Marsac	350	
St. Beron	2,000	4,000	Salles du Salat	300	800
Sechilienne	1,200	3,500			
Serres	600		Total	20,505	50,300

At Niagara Falls, the Union Carbide Co., according to Prof. G. Forbes, are utilizing 10,000 h.p. in the new factory—or two-fifths of the power for which it is ultimately designed. Dunlap* gives some details of the electrical machinery at this factory. When complete it is to contain 100 furnaces, utilizing 25,000 h.p., and producing 100 tons carbide per day. Some particulars of the development at Sault Ste. Marie, and of the Horry furnace were given in the last issue of THE MINERAL INDUSTRY. At Chaudière the carbide factory of Messrs. Bronson & Weston,† is to be in operation by January 1, 1900. Two 3,000 h.p. generating sets are being installed, the dynamos being direct coupled to the horizontal turbines. The current is to be generated at 2,300 volts, and is to be transformed down to 75 volts at the furnaces. Two 400-h.p. sets will supply continuous current at 225 volts for operating the crushing and mixing machinery. The factory is to cost $225,000. A very full account of the Rheinfelden Generating Station was recently published.‡ Each of the two

* *Electrical Review*, May 19, 1899. † *Electricity*, New York, July 26, 1899. ‡ *Electrician*, August 4, 1899.

works established there is being supplied with current from six special generating sets, at 120 volts and 155 volts respectively. A portion of this supply of 12,000 h.p. is being used in each works for carbide production. At Sarpsfos, Messrs. Schuckert & Cie, of Nuremberg, are financially interested in the development of the Falls, and are supplying the generating machinery for one, if not both, of the factories. The first factory is to utilize 3,000 h.p.; and the writer believes that production of carbide has already commenced. An output of 5,000 tons carbide per annum is expected. A second factory has been erected on the opposite side of the Falls, and here 1,500 h.p. is to be utilized, and a production of 3,000 tons carbide per annum is assumed. A third carbide factory of 1,500 h.p., at Meraker, is also stated to be nearing completion; and a fourth, at Kinservik, is expected to be ready for operation in 1900. The last will have 3,000 h.p. at its command.

Messrs. Schuckert & Cie are also financially interested in, and have supplied the generating machinery for the carbide works at Iajce, in Bosnia, with the exception of the turbines, which are by Ganz & Co., of Vienna. The power available here averages 8,000 h. p. throughout the year. The first 2,000 h.p. unit of plant, is reported to have started on April 1, 1899, and the remaining three units were expected to be operating before the end of the year. These fine works are owned by the Bosnische Elektricitäts Aktien Gesellschaft, of Vienna.*

As regards the present output of carbide in Europe, it is wholly impossible to offer any figures that shall even approximate to the true total. If all the 76 factories enumerated in Table I were producing their maximum output, the total would certainly be extremely large. But many of the smaller factories cease production entirely in summer owing to the failure of the water supply; and even the larger factories, whose water power is diminished, but not seriously, during the dry season, reduce their output of carbide on account of the slackened demand. The annual production of carbide is therefore greatly below the apparent aggregate capacity of the factories engaged in the industry. With regard to the output of individual factories, the Foyers works are now stated to be producing at the rate of 3,000 tons per annum; and the new Niagara works at the rate of 12,000 tons per annum.

Financial Details.—With regard to the financial position of the industry, carbide production does not in every case lead to large profits, and the warnings given by the writer in earlier articles on this manufacture, are in some cases being justified by events. Thus, the carbide works at St. Béron, in France, have been advertised for sale by auction—a proceeding hardly characteristic of a flourishing manufacture. A similar fate has also overtaken the English works at Cradley Heath, near Birmingham, where a 300 kw. generating plant and eight carbide furnaces, had been erected in 1897. The Brussels Co. is also in difficulties and is being voluntarily liquidated, but the writer is not certain whether this company manufactured carbide.

These facts show that the manufacture of calcium carbide is not one that may be lightly rushed into by capitalists ignorant of the conditions of supply

* *Elektrotech. Zeitschrift*, April 13, 1899.

and demand in this new industry, or of the position as regards furnace patent rights. Owing to non-acquaintance of promoters, and others, with the practical necessities of electric furnace work, as applied to carbide production, large sums have in many cases been paid for patents which, on further trial, have proved valueless; and in this manner many of the existing carbide companies have been over-capitalized at the commencement of their career. As the price of carbide falls, such companies must inevitably go into liquidation, or write large sums off their capital account; and those companies which are using simpler forms of furnace, and have paid nothing for patent rights, will unquestionably be in the stronger position.

A new arrangement has been entered into at Foyers, under which the British Aluminium Co. undertakes the manufacture of the carbide, and the Acetylene Illuminating Co., which formerly leased the power from the Aluminium Co., now restricts its activities to the sale of the product. The consolidation of the carbide companies in America, under the Union Carbide Co., was referred to in the last issue of the THE MINERAL INDUSTRY. One of the Italian carbide companies has made a profit of $40,468 during 1898-1899, and a portion of the 12% dividend paid by the Neuhausen Aluminum Co. has no doubt been earned on carbide.

Furnaces.—The furnaces in use in the 76 works now in operation vary greatly in design and construction. According to one authority, a return is being made to simpler forms, many of the highly complicated structures which formed the basis of patents having, on trial, proved of little practical value.* At Langenthal, in Switzerland, the furnaces are merely chambers of refractory firebricks, into which weigh-boxes of sheet-iron, containing the charges of lime and coke, are pushed.† Borchers has patented a new furnace, which is designed to utilize the heat lost by radiation and conduction from the furnace walls. The furnace body is of sheet-iron, and is provided with a water jacket. The steam generated during the passage of the current and the cooling of the fused carbide, is used for power generation.‡ At Bellegarde the Bertolus form of furnace is in use§ while at St. Marcel, in Italy, the furnace patented by Memmo has been adopted.‖ It is interesting to note that at Langenthal, at St. Marcel, and at Bellegarde-sur-Valserine, triphase current is being used with success for carbide production. In the last issue of THE MINERAL INDUSTRY it was pointed out that the continuous system of carbide production was electrically less efficient than the intermittent system, though the former yields a more homogeneous block of carbide. Since competition between the 76 producing works is likely to grow very keen, it is probable that furnaces, working on the intermittent system will gradually displace the other type. An illustrated description of the Memmo furnace will be found in the paper named below.¶ This furnace is protected in Germany by Patent No. 105,386. The Herault carbide furnace is probably at present the form most widely adopted in Europe. This will be found described in Dr. Borchers' report on carbide—published in the 1896 volume of THE MINERAL INDUSTRY.

* *L'Industrie Electrochemique*, December, 1898.
† *Excursion Electrotech. en Suisse*, Chapter X.
‡ *Electrician*, May 19, 1899.
§ *L'Industrie Electrochemique*, December, 1898.
‖ *L'Electricien*, July 8, 1899.
¶ *Zeitschrift für Angewandte Chemie*, Oct. 24, 1899.

These two furnaces are continuous in operation. At Neuhausen and at Bitterfeld, according to Borns,* continuous furnaces designed by Rathenau are in use.

Working Details.—Keller has recently given a very full account of the plant and methods of work adopted at the Meran works in the Austrian Tyrol.† This factory is owned by the Acetylen Gas Gesellschaft, of Vienna, which has reduced its original capital to 200,000 fl., and has made a new issue of 600,000 fl. in order to complete the plant at Meran. This company intends now to confine itself to carbide production. The water used has a head of about 90 m.; two pipe-lines, 2 m. in diameter, carry it to the turbine house. This contains 5 horizontal turbines by Ganz & Co., direct coupled to five 1,200 h.p. dynamos, running at 320 r.p.m., and generating current on the triphase system at 10,000 volts pressure. Two of these dynamos supply the towns of Meran and Boyen with current; two supply the carbide works; and one is held in reserve. At the carbide works the current is transformed down to 33 volts, and is used in the Gin & Leleux intermittent form of furnace. Each furnace takes 8,000 ampères at the voltage named, or 260 kw. electrical energy. The lime used is obtained in the locality, and after crushing is mixed with the crushed coke in specially designed mixing machines. The carbon electrodes are made at the factory, and their cost represents about 16·5 fr. per ton of carbide produced. The carbide, as it comes from the furnaces, is in blocks 950 kg. in weight, and is of about 94% purity.

Pierard‡ has given a fairly full description of the plant and methods of work at the St. Marcel carbide factory in the Val d'Aosta, in Italy. Triphase current is used at this factory in Memmo furnaces, three separate carbon electrodes being used to convey the current into the furnace. These may either be used alone, or a fourth neutral electrode may be fixed in the hearth of the furnace. The three arcs which are produced, lead to a more regular heating effect. The furnace is of the continuous type, the molten carbide being tapped at intervals. This factory utilizes about 800 h.p.

Yield of Carbide.—With regard to the yield of carbide per e.h.p. day, there is little progress to report, and the actual yields obtained in some of the works now in operation are believed to be much below the maximum possible in well-conducted works. At Meran, in Austria, an output of 5·2 kg. per kw. day is said to be obtained with the Gin & Leleux type of furnace, and according to M. Gin, this equals a thermal efficiency of 92%.§

At St. Marcel, in Italy, with the Memmo furnace, an output of 4·88 kg. per kw. day has been obtained.‖ Liebetanz, in his paper before the Buda-Pesth Congress, based his estimates on a yield of 5 kg. per kw. day of 24 hours,¶ and this may be taken as the average for the best managed carbide factories. Dr. Borchers, in a recent review** of a book by Liebetanz, on calcium carbide, has stated that most factories consume between 5·4 and 5·76 e.h.p. hours per kg. of carbide produced. This equals a yield of 5·7 to 6·0 kg. carbide per kw.

* THE MINERAL INDUSTRY, 1897.
† *L'Eclairage Electrique*, May 6, 1899.
‡ *L'Electricien*, July 8, 1899.
§ *L'Eclairage Electrique*, May 6, 1899.
‖ *L'Electricien*, July 8, 1899.
¶ *Zeitschrift für Angewandte Chemie*, July 11, 1899.
** *Zeitschrift für Elektrochemie*, Sept. 7, 1899.

day, or one considerably higher than that taken by Liebetanz as the average of present practice. These figures may with advantage be compared with those given by Cowles, in the 1897 volume of THE MINERAL INDUSTRY. The theoretical yield is there given as 16·22 lb. carbide per e.h.p. day, equals 22·05 lb. per kw. day, or 10 kg. The electrical efficiency of the Gin & Leleux furnace, on his basis, is therefore 52%.

Cost and Selling Price.—The selling price of carbide still remains much higher than is justified by its cost of production. This is due chiefly to freight charges, which often add 50% to the price f. o. b. at the factory. Thus at Lourdes, in the Haute Pyrenees, where a carbide factory is now in course of erection, carbide from Switzerland and America is sold for $115 per ton, while it could be produced in the district for less than $57. In Switzerland carbide is selling for $67 per ton, while the same carbide in Paris costs $91 per ton. At Meran, in the Austrian Tyrol, it is estimated that the carbide costs only $34·80 per ton f. o. b., after allowing for interest on capital and depreciation of plant. At Niagara it is estimated that the carbide now produced there costs between $36 and $38·40 per ton, and it is hoped to reduce this total to $28·80 by improvements in the furnaces, and in the details of manufacture.

In a consular report it has been stated that the wholesale price of carbide in France is $67 (packages extra), while the retail price is at the rate of $105 per ton. Stern, in an article on acetylene lighting in Germany, states that the sale price of carbide in that country in small lots is at the rate of $84 per ton. He considers this high price to be due to the action of speculators, who have little regard for the advancement of the industry. Liebetanz, in the paper alluded to above, estimates the cost of carbide, with steam power, at $56 per ton, and at $42·7 when water power is used. He points out that when the freight charges are excessive, it may be more economical to produce carbide in the district where it is consumed by the aid of steam power, than to import it from a distant factory operating by water power. This is a truth which needs emphasizing in these days, when almost every water power in Europe is being exploited for the manufacture of calcium carbide.

In Germany the average price of carbide, as sold to the consumers in large quantities, has been $84 per ton. This grade yields 4·5 to 4·8c. ft. of acetylene per lb. carbide. The British Aluminium Co. have arranged to sell the carbide produced at Foyers at $86 per ton, packages being charged extra. Keller, in the article already named,* upon the Meran factory, has given very full details of the cost of production. This factory is one of the newest, and is under scientific management, and as the figures are apparently based on actual results, the items of cost are reproduced below:

	Per Ton Carbide.			Per Ton Carbide.	
Electrical energy (6,400 h.p. hrs. @ 50 fr.)	45·80	Fr.	Interest and depreciation	24·85	Fr.
Lime	18·80	Fr.	Gen'l office and managem't chgs.	20·00	Fr.
Coke	26·00	Fr.	Repairs	7·50	Fr.
Electrodes	16·50	Fr.	Transport to railway siding	8·00	Fr.
Mechanical power, for mixers, etc	5·00	Fr.			
Labor (Laborers at 3·50-4·00 fr. per day)	18·50	Fr.	Total	190·95	Fr.
Packing	5·00	Fr.			

* *L'Eclairage Electrique*, May 6, 1899.

Since the weight of the package is included as carbide, 10 fr. is deducted for this gain, and the net cost works out to 181 fr., or $34·75 per ton.

Pierard, in his article on the work of the Memmo three-phase furnaces at St. Marcel, states that the electrodes are consumed at the rate of 5 to 6 kg. per 180 kg. carbide, and that the cost works out to 30 fr. per ton of carbide produced. This, it will be noticed, is considerably higher than that given by Keller.

Patent Validity.—There has been a lull in the litigation over carbide patents since the judgment *re.* the Bullier Patents in Germany in 1898. In that country the manufacture of carbide is now legally open to all who desire to engage in it, without payment of any royalty. In the United Kingdom the litigation which was threatened by the holders of the Willson patent rights against other companies early in 1899, has not been carried to a further stage. The companies who are manufacturing carbide at places other than Foyers, consider that the legal proceedings, if they occur, will be fruitless, and that the Willson claims cannot be upheld in the United Kingdom.

Analysis of Carbide.—A modified form of apparatus for testing commercial carbide is described by Hanekop.* This is based on that suggested by Lunge & Cedercreutz; the acetylene after liberation is not measured, however, but its volume is calculated by the loss in weight of the apparatus. The solid impurities of the carbide can also be determined.

UTILIZATION.—*The Acetylene Gas Industry.*—The numerous uses which have from time to time been suggested for calcium carbide have none of them developed to any important extent, with the exception of that for the generation of acetylene gas, and under the heading of *Utilization* this industry will alone be dealt with.

Generation of Acetylene Gas from Carbide.—The Patent offices of all countries continue to be bombarded with patents relating to acetylene generators. The writer has recently estimated that in the United Kingdom alone 600 patents for acetylene generators or lamps, have been granted; and it is his opinion that barely 1% of these patents are worth the time and money that have been expended upon them.

It is satisfactory to find that the truth of this opinion is gradually becoming recognized, and that others besides the writer are giving expression to it, in print.† The principles upon which the majority of these patent generators are based, have been acted upon for years by chemists in the laboratory generation of gases,‡ and the only novel features of the apparatus are generally automatic cut-offs of the water supply, which in practice fail to act at critical moments in the generation of the gas. It is worthy of note that at the Acetylene Gas Exhibition, held at Cannstadt in Germany in 1899, all the highest awards, with one exception, went to generators in which the carbide was projected into a large volume of water. This form of generator is that which is now commonly adopted in Germany, and the opinions expressed by Prof. Lewes in his paper

* *Zeitschrift für Angewandte Chemie*, June 30, 1899; abstracted in *Journal of the Society of Chemical Industry*, August, 1899.

† *Journal of Acetylene Gas Lighting*, May, 1899.

‡ Letter to *The Engineer*, Aug. 25, 1899.

before the members of the Society of Chemical Industry in London in June, 1898, have thus received practical endorsement.* The Allgemeine Carbide und Acetylene Gas Gesellschaft, of Berlin, have erected this type of generator at a very large number of places in Prussia, and the feed of carbide in these is not automatic, but is carried out by hand labor. Any gas engineer or practical chemist could be trusted to design an acetylene gas generator of this type, which would work satisfactorily when fed by hand; and the idea that there can be any master-patents for such a piece of apparatus is absurd.

With regard to the financial position in the acetylene gas industry, the above remarks will show that this is not very stable. Too many companies have been floated—especially in the United Kingdom—whose chief asset is a patent of dubious value. The only profits heard of in connection with these companies, are generally those pocketed by the inventors (?) and promoters. The editorial remarks of the English paper† devoted to the interests of the acetylene industry may be cited in support of this statement. In these it is asserted that more than one English company is already in the hands of debenture holders. During the twelve months ending August, 1899, 15 acetylene gas companies, with an aggregate capital of over $2,000,000, were floated in the United Kingdom; while in the same period three companies of a similar character went into liquidation through inability to pay their debts. In Germany there has apparently been less speculation in the new industry; and according to Liebetanz there are only nine companies engaged in the manufacture and sale of generators. The aggregate capital of these nine companies is $1,080,000, and they give employment to 1,020 persons. Under fair conditions of capitalization this industry is of course a remunerative one, as is proved by the last year's balance sheet of one of the more reasonably capitalized English companies, which is paying a 6% dividend.

With regard to the scientific aspects of acetylene generation, the year 1899 has not produced much additional information. Dr. Erdmann, in a paper read at the Cannstadt Acetylene Gas Exhibition already alluded to, discussed the question of generator efficiency, and stated that his own experiments showed that a diminution of between 20 and 25% in the lighting power of a given weight of carbide occurred, when heating was allowed to take place in the generator.‡ This confirms the opinions to which Prof. Lewes gave expression, in the paper mentioned above.

Purification of the Acetylene Gas.—The purification of acetylene gas is now receiving considerable attention from scientific investigators, and some valuable contributions to the literature of this subject have recently been published. It was formerly supposed that the acetylene gas might be safely burnt as it came from the generator without purification, but it is now recognized that for domestic lighting and for public halls it is absolutely essential that some process of purification should be adopted. The chemicals that have been recommended and tried for this purpose are:§

(*a*) Lime water, by Lundstroem; (*b*) bleaching powder, by Lunge &

* *Journal of the Society of Chemical Industry*, June, 1898.
† *Journal of Acetylene Gas Lighting*, April-May, 1899.
‡ *Die Chemische Industrie*, July 15, 1899.
§ *Journal für Gasbeleuchtung*, 1899, p. 198.

Cedercreutz; (*c*) bleaching powder and lead chromate, by Wolff; (*d*) acid cuprous chloride solution, by Franks; (*e*) chromic acid, by Ullmann.

The first removes all impurities with the exception of the phosphoretted hydrogen, the most dangerous to health; it is therefore useless alone. The use of bleaching powder is condemned by Caro,* on account of the possible formation of nitrogen trichloride, and therefore only the methods of Franks and of Ullmann can be considered of practical value. The absorbents adopted in these two methods may be used in conjunction with powdered quartz, and these mixtures have been patented under the names of "Frankolin" and "Heratol" respectively. It is stated that 5·5 g. of chromic acid suffice to purify 1 cu.m. acetylene gas; but it is evident that the amount required must depend upon the percentage of impurities present. Caro recommends that the purified gas should be subjected to a final scrubbing action with paraffin oil, in order to remove any impurities picked up from the chemical reagents.

In relation to this question of purification, Liebetanz, in his report upon the exhibition of acetylene generating and purifying apparatus at Buda-Pesth in the early part of 1899, makes the following remark: "It is to be urgently recommended that makers of acetylene generators should entrust the design of the purifying apparatus to those who have a special knowledge of the subject; for the purifying apparatus exhibited showed in many cases a complete ignorance of chemical knowledge on the part of the designer, and often formed a dangerous addition to the complete apparatus."†

Rose, in his report upon the exhibits at the still more recent Cannstadt Exhibition in Germany, states that "Frankolin" (see above) costs 36c. per kg., and that this amount will purify between 1,000 and 1,700 cu. ft. of the crude gas. The "Frankolin" can be regenerated after use.‡ Erdmann, in the paper read at this exhibition (see above), discussed the use of bleaching powder as a purifying agent, and disagreed with Caro's condemnation of it. He stated that danger only arises when the generator is of faulty design; *i.e.*, when the gas passing to the purifier is considerably heated. If the temperature of the generator and of the gas issuing from it, be kept low, no dangerous results, in his opinion, can follow. Ahrens has also recently investigated the cause of the dangers, which, under certain conditions, follow from the use of bleaching powder for acetylene purification.§ Owing to these dangers and to the smell of chlorine imparted to the purified gas, bleaching powder was discarded after trial at a central generating station at Veszprim, in Hungary. Its use also caused a fire at the Buda-Pesth Exhibition, and an explosion at a mill in Wurtemberg; while in a recent letter,‖ Dr. Dent gives details of three other fires attributable to the use of bleaching powder for purifying acetylene gas. Ahrens' investigations have led him to the opinion that the heating of the bleaching powder which had occurred in all these cases, was due to the sawdust with which it had been customary to mix it before use, in order to

* *Zeitschrift für Angewandte Chemie*, July 11, 1899.
† *Dingler's Polyt. Journal*, June 10, 1899.
‡ *Journal of Acetylene Gas Lighting*, July-October, 1899.
§ *Zeitschrift für Angewandte Chemie*, Aug. 15, 1899.
‖ *Engineer*, Nov. 10, 1899.

obtain a more porous material. Ahrens therefore recommends that kieselguhr, brick-dust, or coke-powder, should be used in place of sawdust for this purpose. An anonymous writer has also discussed the question of acetylene purification at considerable length.* This authority recommends bleaching powder as a purifying agent, on account of its lower cost, and would remove the smell of chlorine which is imparted to the acetylene, by passing the gas through a second purifying chamber containing slaked lime. The danger of fire or explosion, he would avoid by using some inert inorganic substance for mixing with the chloride. Coke is condemned as practically as dangerous as sawdust; but quartz is named as a most suitable material. It is somewhat difficult, however, in the absence of a chemist, to always guarantee that the crude gas shall be free from ammonia; and all these authorities admit that in presence of ammonia the use of bleaching powder is dangerous. Cooling the gas and washing with water, will, however, generally remove this impurity.

One may sum up the present position therefore, by stating that "Frankolin" and "Heratol" are safe and efficient purifying agents, but are costly; while bleaching powder is cheap and efficient, but is dangerous in ignorant hands, and should never be used except under constant scientific supervision. That it is still being employed, is proved by Pfeiffer's description of the purifying apparatus used by the Hungarian State Railways.†

Testing Acetylene Gas.—The modified form of the Lunge & Cedercreutz apparatus for testing commercial carbide, has already been referred to. (See "Analysis of Carbide.") The impurities of the acetylene gas may also be determined with this apparatus. Phosphoretted hydrogen, the most dangerous and most common impurity of crude acetylene gas, can be detected by the color of the acetylene flame, and by the smell and haziness which it imparts to the atmosphere of the closed room, in which an acetylene flame is kept burning. According to Ahrens, its presence may be detected chemically by aid of Berge's reagent—(10 pts. of mercuric chloride and 80 pts. of a 30% solution of hydrochloric acid) which yields first a cloudiness, and then an amorphous precipitate with phosphoretted hydrogen. As pure acetylene also yields a precipitate with this reagent (in this case crystalline), considerable skill and practice are necessary to obtain reliable results with the test.‡

Hempel & Kahl describe a method for determining quantitatively the amount of phosphoretted hydrogen present in crude acetylene gas.§ An acid solution of cupric sulphate is used for absorbing this impurity.

Properties of Acetylene Gas.—The year 1899 has not produced great additions to the knowledge of acetylene. Hanekop, in the article already named, states that 1 liter of pure acetylene at 0° C. and 760 mm. weighs 1·161 g. At the Buda-Pesth Congress, Grittner contributed a paper upon the action of acetylene gas on metals, in which he stated that the dry and *pure* gas has no action upon copper; whereas, even in the absence of ammonia, this metal was attacked by the impure gas.‖ According to Erdmann, the temperature of the luminous portion of an acetylene flame is lower than that of the ordinary

* *Engineer*, Nov. 3, 1899.
† *Zeitschrift für Angewandte Chemie*, Aug. 28, 1899.
‡ *Zeitschrift für Angewandte Chemie*, July 11, 1899.
§ *Ibid*, Jan. 2, 1898.
‖ *Ibid*, July 11, 1899.

coal gas flame; a statement in direct conflict with the theory that luminosity depends upon the temperature to which the carbon particles are raised during combustion. In the Rossler gas furnace, a temperature of 1,500° C. has been attained with acetylene; but when used for heating purposes it is necessary to insert a wire gauge in the mixing tube, to prevent explosions of the air and acetylene gas mixture.*

Utilization of Acetylene for Lighting Purposes.—The attention which has been paid to the scientific aspects of the generation and purification of acetylene, has improved the practical value of the gas as an illuminant, and its use for this purpose has undergone considerable extension during the past year.

The high price of carbide in certain districts remote from a producing factory is, however, a considerable hindrance to the further extension of acetylene lighting, and there is no doubt that in many places where it has already been adopted, at the present price of carbide, it will be found a very costly illuminant. It is customary for those engaged in pushing the sale of acetylene apparatus, to urge the use of acetylene because it is cheaper than coal gas or oil, in the particular locality where they may chance to be operating. In numerous instances this is absolutely false, and such tactics will not conduce to the permanent gain of the carbide or acetylene industries. *The Progressive Age*, a paper favorable to the acetylene industry, has recently been commenting on this tendency to underrate the cost of the gas when dealing with prospective buyers of generating apparatus, and has referred to the distribution of pamphlets and leaflets, in which the ratio of cost between illumination by coal gas and by acetylene, is stated to be 4·2: 1·0.† The lowest ratio found in any of these circulars was stated to be 1·0: 1·0, while the actual ratio calculated on the prices then current in New York was 0·5: 1·0; acetylene being twice as costly as coal gas for equal illuminating power.

The dissemination of false information of this character has led to the discarding of acetylene in some cases, after actual trial. Thus, the Canadian Electrical Association has recently made an inquiry concerning the position and prospects of acetylene lighting in Canada.‡ It found that there had been 90 installations of acetylene apparatus, involving the operation of 217 generators; but that 62 generators, or over one-fourth of the total number, after a few months' trial, had been put out of use.

An anonymous writer§ recently discussed this question of relative costs of acetylene and coal gas, in a singularly fair and temperate article, and summed up by urging the general adoption of acetylene as an illuminant, on grounds—not of economy, but of health. Though in favor of the new illuminant, this writer was obliged to admit that, as compared with coal gas or oil, it was, in most places at present much more costly, and was likely to remain so.

That there are localities where acetylene is cheaper than coal gas is, of course, true, and the correct policy of the carbide and acetylene companies is to seek out these, rather than to direct their attention to districts where coal gas or electricity are in the stronger position. In such localities oil is the

* *Die Chemische Industrie*, July 15, 1899.
† *Progressive Age*, April 1, 1899.
‡ *Lightning*, Aug. 31, 1899.
§ *The Engineer*, March 31, 1899.

only real competitor, and though acetylene can never hope to compete with oil on the ground of cost, yet it may do so with some success on the grounds of convenience and health.

As regards actual progress in Great Britain a large number of country houses in England, Scotland, and Ireland have been lighted by acetylene; the light has been adopted in a few small factories in Lancashire and Yorkshire; several portable flare lights are in use on the Manchester Ship Canal at Salford; and at St. Michaels in Kent, and at Hawes in Yorkshire central supply stations have been erected.

At Scarborough the Electroid Gas Co. has been formed to supply a mixture of acetylene and some unnamed diluent, to the villages in the vicinity. The mixed gas is said to have an illuminating value ten times that of coal gas, and to be one-third less costly than acetylene. It is supplied through ordinary gas mains and fittings.*

It is impossible to give any accurate estimate of the total number of lights now installed in the United Kingdom, or of the actual annual consumption of carbide, but it is evident from the above details that acetylene lighting is making progress in certain districts.

In France the most noteworthy sign of progress is the decision of the General Omnibus Co., of Paris, to use acetylene for lighting the trams on the Louvre-St. Cloud section. The carbide is to be specially treated by the Serpollet process in order to obtain a coating of glucose upon it. On bringing into contact with water, a sucrate of lime is formed, which, being a soluble salt, facilitates the use of comparatively small generators. Details of the apparatus to be used will be found in the paper named below.†

In Germany several public installations of acetylene for lighting purposes have been carried out; *e.g.*, Olivia, near Dantzig, and Schönsee, West Prussia. The Berlin Central Post-Office authorities are also experimenting with the new light. But it is in railway-station lighting that the new illuminant has made most progress in Germany. According to Rose‡ the stations at Schlacktensee, Olivia, Albona, Frankfort, Hanover, Mainz, Augsburg, Munich, Kleinen, and Asperg, have all their own acetylene generating plants. Wilhelmsbad, Heilsburg railway station, and a factory in Luxemburg, are some of the latest places where acetylene lighting has been adopted. With regard to the actual extent of acetylene lighting in Germany, Erdmann, in the article more than once alluded to, has stated that in July, 1898, 62,000 acetylene burners were in use, and that by January, 1899, the number had risen to 170,000. Another authority states that 6,450 generators had been sold, up to the commencement of 1899, and that 112,350 burners were in use. The consumption of carbide is given as equal to 14,570 tons, and the average value of this as $84 per ton.§ These estimates must be received with caution, since they are obviously founded upon the number of generators *sold;* and the experience in Canada shows that this number is no guide to the number in *actual use*, at a later date.

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* *Industries and Iron*, Sept. 22, 1899.
† *Le Genie Civil*, July 1, 1899.
‡ *Journal of Acetylene Gas Lighting*, July–October, 1899.
§ From *Das Acetylen*, in *Journal of Acetylene Gas Lighting*, July, 1899.

Outside France and Germany the principal progress appears to be occurring in southeastern Europe, where coal gas and electricity are comparatively unknown. Thus at Syra, in Greece, an installation of acetylene gas for public uses is being carried out, oil being the only competing illuminant. The carbide is to be imported from Belgium, and the acetylene is to be charged at the rate of 4 dr. 50 l. per 1,000 cu. ft. This is equivalent to 86c. per 1,000 cu. ft., and the estimate is evidently erroneous.*

At the town of Tata-Tovaros, in Hungary (population, 12,000), 158 public and 250 private lights have been installed, and 8 km. of gas mains have been laid down. The gas is generated in a central works by complete immersion of the carbide in water, and it is afterward purified. The installation has cost $28,800; $18,240 being spent upon the distributing system. The 158 public lights bring in an income of $1,152 per year, or $7·3 per lamp per year. The actual cost of the gas is stated to be $13·6 per 1,000 cu. ft., while it is supplied to the consumers at $14·11 per 1,000 cu. ft. Swiss and French carbide is used at the generating works.† According to Pfeiffer, acetylene is being used as an illuminant on the Hungarian State Railway.

The use of acetylene in conjunction with other gases is considered by many to be the most promising field of exploitation, and mixtures of acetylene and oil gas are being used upon some branches of the German State Railways. Wedding, of Berlin, has recently published details of experiments made with mixtures of acetylene and air, and with mixtures of acetylene and coal gas;‡ while Guilbert, at the recent Buda-Pesth Congress, read a paper on the use of mixtures of acetylene and oil gas.§ Mixtures of these two gases in the proportions 25:75 gave double the lighting value of oil gas alone (not *treble* the intensity, as sometimes stated); but the most favorable results were obtained with mixtures containing 40 to 50% acetylene.

Taking carbide at $124·8 per ton in Paris, the cost of 1 cu.m. (35·3 cu. ft.) acetylene gas will be 47c., including 5·7c. for generating costs. The same volume of oil gas costs 15·3c. Using these costs as the basis of calculation, the costs per cu. m. of various mixtures of acetylene and oil gas work out as follows:

75% oil gas.... } 25% acetylene. } ....23·21 cents.	60% oil gas..... } 40% acetylene.. } ....27·98 cents.	50% oil gas..... } 50% acetylene.... } ....31·15 cents.

The first mixture (25% acetylene) is that known as the Pintsch mixture, and is used under compression for railway carriage lighting on the German State Railway.

Hartman has more recently been investigating the lighting value of mixtures of acetylene and hydrogen, and has published his results in the form of curves in the *Physical Review*. The article is not capable of useful abstraction, and those interested in the subject with which it deals, must refer to the original, or to its reprint in the paper named below.‖

Explosions.—Although the dangers of acetylene are now more clearly recognized, explosions of apparatus for making or utilizing the gas still too frequent-

* *British Foreign Office Report*, No. 2,262, May, 1899.
† *L'Eclairage Electrique*, Oct. 21, 1899.
‡ *Dingler's Polyt. Journal*, May 13, 1899.
§ *Zeitschrift für Angewandte Chemie*, July 11, 1899.
‖ *Progressive Age*, Oct. 16, 1899.

ly occur, and this review of progress in 1899 would be incomplete were these entirely passed over.

The writer has record of six explosions during 1899, causing two deaths and three cases of injury. The most disastrous of these occurred at Wabash, Ind., on August 7, 1899, when the municipal acetylene gas plant at that place was entirely destroyed. The generator had been charged with calcium carbide a few minutes before the explosion, and it is supposed that this was caused by a too rapid evolution of gas. Happily, no one was injured on this occasion. Dr. Munsterberg, at the Buda Pesth Acetylene Congress, gave an address, in the course of which he stated that since April, 1897, he had recorded 49 acetylene gas explosions. Over one-half of these accidents had been due to the over-production of gas by the so-called automatic generators, and to the consequent escape and ignition of the surplus gas. Automatic apparatus is, therefore, condemned when charges of carbide over 6 lb. in weight are to be used.*

The Canadian Electrical Association reported early in 1899 that no fewer than 30 accidents had occurred in Canada in connection with 90 installations of acetylene lighting. These figures show how necessary it is that only qualified engineers should plan and control installations of acetylene gas. Ordinary coal gas is an explosive under certain conditions; but if the proportion of accidents were as high as in the acetylene gas industry, thousands of lives would be lost each year through explosions of coal gas. Acetylene gas, under normal temperature and pressure, is only slightly more dangerous than coal gas; and the application of a small amount of scientific knowledge on the part of designers, and of common sense on the part of the users of acetylene apparatus, should bring the percentage of explosions within the limits of that found in the older industry.

Doubtless, in time this comparative immunity from accident will be attained; but in the meantime the death-roll mounts up, and acetylene gas is maintaining its unenviable reputation.

Conclusion.—This review of recent progress in acetylene generation and utilization, shows that advancement is occurring, and that in certain directions the acetylene gas industry is likely to undergo permanent and successful expansion.

This expansion will not occur, however, in districts where coal gas and electricity are already established, or where they can be favorably installed as illuminants; and those who desire the establishment of the carbide and acetylene gas industries upon a sound basis, should recognize this limitation to their field of operations, and should, above all, discourage the circulation of false information concerning the relative costs of acetylene, coal gas, and oil, when used for illuminating purposes.

* *Journal of Acetylene Gas Lighting*, September, 1899.

CEMENT.

THE production of natural, Portland and slag cement showed a great increase during 1899 above the production in the previous year, although 1898 itself was the record year up to that time. Pennsylvania, as before, leads the States, with a production of 3,250,209 bbl. of Portland cement* (of 400 lb.) valued at $5,687,865, and New Jersey has nearly doubled its production, producing more than double that of New York.

PRODUCTION OF PORTLAND CEMENT IN THE UNITED STATES. (IN BARRELS OF 400 LB.)

States.	1898.			1899.		
	Barrels.	Value at Works.		Barrels.	Value at Works.	
		Total.	Per Bbl.		Total.	Per Bbl.
California	40,586	$92,130	$2·27	48,000	$112,000	$2·35
Michigan	81,500	162,125	1·99	402,000	701,000	1·74
New Jersey	585,000	994,500	1·70	960,000	1,776,000	1·85
New York	529,830	900,694	1·70	507,931	989,672	1·85
Ohio	277,805	472,368	1·73	482,542	916,330	1·90
Pennsylvania	2,017,088	3,428,965	1·70	3,250,209	5,687,865	1·75
South Dakota	30,000	60,000	2·00	34,000	68,000	2·00
Texas	(e) 8,000	(e) 20,800	2·60	(e) 10,000	(e) 25,000	2·50
Other States (a)	14,837	29,534	1·99	110,988	215,064	1·94
Total	3,584,586	$6,168,106	$1·72	5,805,620	$10,441,431	$1·80

(a) Includes Arkansas, Illinois, Indiana, Maryland, North Dakota, and Utah. (e) Estimated.

PRODUCTION OF NATURAL HYDRAULIC CEMENT IN THE UNITED STATES.
(In barrels of 300 lb.)

States.	1898.			1899.		
	Barrels.	Value at Works.		Barrels.	Value at Works.	
		Total.	Per Bbl.		Total.	Per Bbl.
Illinois	531,000	$160,000	$0·30	548,256	$137,275	$0·25
Indiana and Kentucky	1,929,018	482,254	0·25	3,534,344	883,586	0·25
Kansas	180,000	67,000	0·37	160 000	64,000	0·40
Maryland	235,000	99,640	0·42	325,000	132,250	0·41
Minnesota	98,436	49,218	0·50	103,986	51,993	0·50
New York:						
Ulster County	3,245,225	2,103,554	0·65	3,425.560	2,808,959	0·82
Onondaga County	122,300	62,700	0·51	139,586	68,672	0·49
Schoharie County	61,272	27,572	0·45	85,000	43,350	0·41
Erie County	605,000	211,750	0·35	695.000	347,500	0·50
Ohio	27,144	19,301	0·79	21,597	19,437	0·90
Pennsylvania	694,152	347,076	0·50	700,034	420,020	0·60
Virginia	5,491	3,430	0·62	12,064	7,206	0·60
Wisconsin	(e) 400,000	(e) 160,000	0·40	411,000	173,450	0·42
Other States (a)	27,040	26,410	0·98	25,000	25,800	1·03
Total	8,161,078	$3,819,995	$0·47	10,186,447	$5,183,500	$0·51

(a) Includes Georgia, Florida and Texas. (e) Estimated.

* For an estimate which has proven almost exact in 1899 as well as for 1900, see THE MINERAL INDUSTRY, Vol. VII.

Michigan appeared for the first time as a large producer, and it is thought that the production of this State will become a constantly increasing factor in this industry. The production of slag cement in 1899 was 244,751 bbl. of 400 lb. ($360,800) against 157,662 bbl. ($235,721) in 1898.

Market Conditions.—The brisk demand for building and structural work maintained the price at an average of 10c. per bbl. above that in the previous year. Advance orders and work under way would indicate that equal prices can be depended upon in 1900 despite the predicted increase of production.

CEMENT PRODUCTION, IMPORTS, EXPORTS AND CONSUMPTION IN THE UNITED STATES.
(In barrels of 300 lb.)

Year.	Production.				Imports.		Exports. (b)		Consumption.	
	Natural Hydraulic.	Portland.	Total Barrels.	Value.	Barrels.	Value.	Barrels	Value.	Barrels.	Value.
1895.....	7,694,053	998,745	8,692,798	$6,027,374	3,996,527	$3,873,123	123,437	$131,541	12,565,888	$9,768,956
1896.....	7,407,311	a 2,103,044	9,510,355	6,888,441	3,558,166	3,394,426	87,910	103,315	12,980,611	10,179,552
1897.....	7,890,573	a 3,294,537	11,185,110	7,760,955	2,787,766	2,688,122	79,583	103,389	13,893,293	10,345,688
1898.....	8,161,078	a 4,989,664	13,150,742	10,223,822	2,685,092	2,624,228	70,892	98,121	15,764,942	12,749,929
1899.....	10,186,447	a 8,067,161	18,253,608	15,985,731	2,810,951	2,858,286	147,029	213,457	20,917,530	18,630,560

(a) Includes slag cement. (b) Includes re-exports of foreign.

IMPORTS OF CEMENT INTO THE UNITED STATES ACCORDING TO SOURCE.

Year.	Belgium.		British North America.		France.		Germany.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1895.....	141,775	$965,687	2,083	$18,894	4,567	$27,842	259,984	$1,737,442
1896.....	140,197	813,827	2,096	17,777	6,531	38,317	246,666	1,629,785
1897.....	109,531	649,675	981	9,311	7,545	44,574	221,856	1,467,845
1898.....	133,806	80,949	955	8,852	3,459	22,234	206,982	1,394,551
1899.....	124,830	747,448	880	71,676	3,130	22,362	238,764	1,693,722

Year.	United Kingdom.		Other Countries.		Total Imports.		Exports. (a)	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons	Value.	Short Tons.	Value.
1895.....	161,377	$1,014,316	29,693	$208,942	599,479	$3,873,123	18,566	$131,541
1896.....	126,268	808,550	12,056	86,615	533,725	3,394,426	13,186	103,315
1897.....	68,869	451,256	9,380	65,461	418.165	2,688,122	11,937	103,389
1898.....	47,212	333,405	10,350	64,247	402,764	2,624,228	10,633	98,121
1899.....	39,927	309,614	14,112	76,372	421,643	2,858,286	22,054	213,457

(a) Estimated from number of barrels reported at 1 bbl.=400 lb. Includes re-exports of foreign.

THE HYDRAULIC CEMENT INDUSTRY IN THE UNITED STATES IN 1899.

BY FREDERICK H. LEWIS.

Portland Cement.—The Portland cement industry has maintained the interesting activity of industrial progress noted in previous years. Old plants have been enlarged and modernized, numerous new plants are projected or in the course of construction; the output has greatly increased and the market has absorbed it all at satisfactory prices. The general excellence of the present American product is making an increased demand for cement so that consumption is keeping pace with output, and this increasing use of Portland cement is undoubtedly working to the disadvantage both of the natural cement and lime industries. The Lehigh Valley district has easily maintained its supremacy as a center of production. The Atlas, Saylors, Bonneville, Lawrence and

American companies have each enlarged their plants with a large increase in aggregate output. No less than 35 new rotary kilns have been installed. New plants in this district have been in course of construction during the year by Wm. Krause & Sons' Cement Co., Martin's Creek, Pa.; the Whitehall Cement Co., Cementon, Pa.; the Dexter Portland Cement Co., and the Phœnix Portland Cement Co., Nazareth, Pa. These plants will all be in operation in 1900. New construction during 1900 is planned by the Edison Portland Cement Co., (a large plant at Stewartsville, N. J.); the Alpha Portland Cement Co. (10 new kilns at Alpha, N. J.), and the Lehigh Portland Cement Co. (8 new kilns at Ormrod, Pa.). In New York State, the Glen's Falls Cement Co.'s plant was destroyed by fire, but has been in course of reconstruction. A new plant of two rotary kilns has been under construction at West Camp on the Hudson River. Parties said to represent the German firm of Alsen & Co., have purchased property here and expect to build a Portland cement plant. The raw materials at this point are a fossiliferous limestone, known locally as the Beecraft limestone, and the brick clay of the Hudson River Valley. The Virginia Portland Cement Co. had under construction a plant of four rotary kilns at Craigsville, Va., and the Buckhorn Portland Cement Co. began the construction at Rowlesbury, W. Va., of six rotary kilns. In Michigan, considerable new construction is planned by the Peninsula Co., near Jackson, by the Great Northern Co. at Baldwin, and by a third company at Jonesville.

A new plant that will use natural gas as fuel is projected at Iola, Kan. Various schemes and organizations have been noticed at many other points, but so far as actual work is concerned the results of the year have emphasized the Lehigh Valley district as the great producing center of the nation.

The results of the year's business may be said to confirm the rotary kiln as the American type of plant, and considerable development is claimed in respect to increased output, for which larger figures have been quoted than were previously considered practicable. Wherever fuel is cheap in America, the rotary kiln is quite sure to predominate, and it is only in districts remote from the coal producing centers that the best type of continuous shaft kilns may be found cheaper under good management. The production of Portland cement in America was at increased cost last year by reason of an increase in the price of fuel and labor; conditions that will doubtless prevail in 1900. The price of oil as fuel for kilns became prohibitive during the year, and the general substitution of coal is certain.

Nothing new or notable has transpired either in the process of manufacture or in the technical and theoretical part of the industry. More thorough laboratory organization has resulted in a better and more uniform product.

Portland cement became a factor in financial circles during the year; several schemes were on foot to organize the industry into a trust, but without success. The American Cement Co. was organized as a joint stock company and was listed on the Philadelphia Stock Exchange at $10 par with a total capital exceeding $3,000,000. The Edison Portland Cement Co. has been organized as a joint stock company with $11,000,000 capital. Other companies with capital exceeding a million of dollars are also being floated.

CHROMIUM AND CHROME ORE.

THE chrome ore production of California has been at a standstill, but the Pennsylvania mines have produced their usual amount. No new occurrences have been announced in this country.

The greatest demand for chromium is for the manufacture of iron and steel alloys. It is used both in the United States and in Germany for the manufacture of armor plate.

Although the formula is a secret one, yet it is reasonably certain that chromium is used extensively, as the Krupps have interested themselves in the manufacture of chromium free from carbon, and according to a statement made to the Naval Committee of the United States Senate by the Ordnance Bureau of the Navy Department, the composition of their plate is as follows: Nickel, 3·5%; chromium, 1·3%; carbon, not less than 0·2%; manganese, not over 0·4%; copper, not over 0·07%; phosphorus, not over 0·03%; sulphur, not over 0·03%; silicon, not over 0·15%. In the Creusot works, in France, it is stated that chromium is replaced by vanadium, molybdenum, or tungsten.

PRODUCTION, IMPORTS AND CONSUMPTION OF CHROME ORE IN THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Quantity. Long Tons.	Value Per Ton.	Value.	Quantity. Long Tons.	Value Per Ton.	Value.	Quantity. Long Tons.	Value.
1895.....	1,553	$10·81	$16,795	5,230	$15·84	$82,845	6,783	$99,640
1896.....	702	11·07	7,775	8,869	21·13	187,400	9,571	195,175
1897.....	150	10·33	1,550	11,566	16·11	186,313	11,716	187,863
1898.....	100	10·00	1,000	16,304	16·70	272,234	16,404	273,234
1899.....	100	10·00	1,000	15,798	18·03	284,825	15,898	285,825

According to a decision in the United States Circuit Court of Appeals, in the case of the United States vs. Dana & Co., ferrochrome is to be classified at the rate of $4 per ton, a tariff similar to that for ferromanganese. The Government had contended that 20% ad valorem should have been paid.

Canada (By J. Obalski).—The Colraine Mining Co. operated regularly during the year has now installed a concentrating plant. Another jig has been added and an annex has been built containing a drier and an electric light plant with boiler and engine. The company is now prepared to furnish a quantity of high grade concentrate besides the ordinary crude.

They have reported the following analysis of concentrated product: Cr_2O_3, 53·64%; FeO, 11·47%; Al_2O_3, 14·02%; MgO, 15·75%; CaO, 2·81%; SiO_2, 2·31%; total, 100%.

The total shipments from that district (practically all to the United States) represent 1,768 gross tons of a value at the mine of $20,867; of this, 500 tons are of first class concentrate. An increased output is expected for next season. The ore in hand at present amounts to a few hundred tons of second class product.

Both the New Caledonian and Turkish productions seem to be falling off. A considerable amount of chrome ore is exported from Canada to the United States both for the manufacture of ferrochrome and chromates, and for furnace lining. The grade of the ore shipped to the United States is not announced, but it is understood that 40% ore, or under, cannot be economically used for furnace lining, while it may find profitable market for the production of ferrochromium.

THE WORLD'S PRODUCTION OF CHROME ORE. (*a*) (IN METRIC TONS.)

Year.	Bosnia.	Canada.	Greece.	New Caledonia (*b*)	Newfoundland.	New South Wales.	Norway.	Russia.	Turkey. (*b*)	United States.
1894.....	1,808	907	6,000	1,042	*Nil.*	3,083	*Nil.*	7,537	20,250	3,388
1895.....	707	2,882	2,740	6,014	44	4,297	190	21,014	21,050	1,578
1896.....	443	2,124	1,600	16,018	1,081	3,914	*Nil.*	6,682	20,137	713
1897.....	396	2,392	563	9,054	3,084	3,433	*Nil.*	13,433	11,551	152
1898.....	458	1,833	1,367	14,300	657	2,145	(*c*)	(*c*)	(*c*)	102

(*a*) From the official statistics of the respective countries, except for the United States, which are our own. (*b*) Exports. (*c*) Statistics not yet published.

North Carolina.—J. H. Pratt, who has made a thorough study of the occurrence of corundum and chromite in the peridotites of North Carolina has adopted the theory that the chromite was crystallized out from the molten peridotite as the mass cooled. In the peridotite rocks of North Carolina the chromite occurs more commonly as scattered grains or crystals, but it also is found in masses or pockets, which apparently have no relation to each other. The largest deposits occur in the peridotite rock near the contact with the inclosing gneiss. The formation is similar to that in which corundum is found, but where large deposits of chromite occur there is little or no corundum, and where corundum is abundant there is a scarcity of chromite.

Although prospecting for chrome ore was first undertaken over seven years ago and has been continued spasmodically ever since, there has never been any systematic development of the locality. When found, the ore is of high grade, though the existence of large bodies has never been conclusively proven. Small shipments have been made. In Yancey County a small deposit was opened up near Burnsville and near Webster, in Jackson County, some pockets have been found.*

The Wilson Aluminum Co. produces 60 tons monthly of ferrochromium at Holcombs Rock, W. Va. The alloy contains 68 to 71% Cr, with carbon in the proportion of one part to ten of chromium.

France.—Ferrochrome is manufactured at Boucau in the S. W. portion of

* *Engineering and Mining Journal*, March 4, 1899.

France. Shipments amounting to 40 or 50 tons were made to England and 400 or 500 tons to Germany in 1898.

New Caledonia.—The shipments of chrome ore in 1899 were 12,480 tons, a large increase over the exportation of the previous year.

New South Wales.—In 1898 the production of chrome ore fell off in New South Wales, the Gundagai District having produced only 2,111 tons valued at £6,301 against 3,397 long tons valued at £10,269 in 1897. The district will produce much larger quantities of chrome ore when the price increases.

New Zealand.—Chrome ore is known to exist in N. W. Otago, New Zealand, but it has not yet been proven to be in quantities sufficient for economical working.

Turkey.—Through the courtesy of W. Thomae we are indebted for the following report from Paterson & Co., of Smyrna, who practically control the industry in Asia Minor. Turkey almost exclusively supplied the requirements of the bichromate trade up to 1895, which then amounted to an average of about 25,000 tons per annum. For the last four or five years the demand has been gradually increasing and they believe that 35,000 to 40,000 tons are now annually required by the bichromate makers and for metallurgical purposes. They consider that New Caledonia competition is responsible for the decrease in price from £5 10s. to £4 10s. for 50% Cr_2O_3, delivered in England. Preference is, however, always given to Turkey mineral, the bichromate makers finding that it is softer and contains less silica and other objectionable matter than the New Caledonia ore. Were it not for the ever increasing difficulties thrown in the way of all mining enterprises in Turkey by corrupt officials, they believe it would be possible to supply the whole of the requirements of chrome ore from there. The principal deposits are in the Hudavendighiar district, of which Makri is the port, and recently some other important discoveries have been made in the Vileyet of Aidin. There are some deposits being worked in the neighborhood of Salonica (European Turkey). The percentage of the ore from these various sources runs from 45 to 52% Cr_2O_3.

Manufacture of Metallic Chromium and Chromium Salts.

The Goldschmidt process described in The Mineral Industry, Vol. VII., has produced a considerable quantity of chromium during the year, as has the Aschermann thermo-electric process described in The Mineral Industry, Vol. VI. Chromium free from carbon is also manufactured by the Elektrochemischen Werke, Bitterfeld, Saxony. Sherard Cowper-Coles, in a paper read before the Institution of Mining and Metallurgy,* treats upon the electro deposition of chromium. The Placet-Barnet process has been worked on a commercial scale. The electrolyte is made by dissolving 100 parts of chrome alum in an equal quantity of water with 10 to 15 parts of barium bisulphate, but Mr. Cowper-Coles was unable to get a deposition from this solution when worked at various temperatures with carbon and platinum anodes. The current

* December 20, 1898.

densities varied from 15 to 19 ampères to the sq. ft., and the voltage at the terminal of the cells was about 4 volts.

Bunsen succeeded in getting a good deposit of chromium, which had the appearance of iron though less alterable in moist air.' It resisted the action of boiling nitric acid, but was acted upon by hydrosulphuric and hydrochloric acid. E. Placet claimed to have obtained excellent deposits of chromium by electrolyzing an aqueous solution of chromic alum, to which alkaline sulphate and a small quantity of sulphuric acid had been added. Krupps at Essen have electrolytically obtained chromium free from carbon, though not necessarily free from iron or ferrochromium. Double chlorides in the metals are used as an electrolyte. The anodes are ordinary carbonized ferrochromium, porous diaphragms being used to separate the anodes and cathodes. A patent was obtained for depositing an alloy of chromium. The alloys in this case were formed by heating chromium compounds with charcoal in a closed crucible, and pouring 2·5 parts of molten copper upon the reduced mass followed by 1 to 1·5 parts of molten tin; this alloy was then granulated, re-fused and cast into the desired forms and used as anodes in a solution of 1 lb. of potassium cyanide, 1 part of ammonium carbonate, and 1 gal. of water heated to 150°F.

T. Feree has used mercury as a cathode, obtaining chromium amalgam by electrolyzing a solution composed of 160 g. of crystallized chromium chloride, 100 g. of concentrated hydrochloric acid and 740 g. of water, the current being 22 ampères. When a solution of chromium chloride alone is electrolyzed with a negative electrode of mercury, the mercury takes up but little of the chromium, a black chromium oxide being formed in the liquor. Mr. Cowper-Coles succeeded in obtaining bright deposits from a solution of chromium chloride. Twenty-five parts of the salt were used to 975 parts of water. It was worked hot at a temperature of 190°F. with a current of between 40 and 50 ampères to the square foot. In the coal no deposit of metal was obtained, but a black non-adherent deposit, the voltage at the terminal of the cells being about 4 volts. Gas was evoked at both anode and cathode during the deposition. Before good deposits could be obtained it was found necessary to add an excess of hydrochloric acid to dissolve the precipitated oxide and form a clear solution. The deposit was found to have a hardness greater than palladium, cadmium and silver, but less than nickel, Sheffield plate, antimony or bright deposited palladium. Its reflecting power as compared with silver was equal, but after tarnishing it showed greater diminution.

A. Goldschmidt* has devised an improved method of manufacturing sodium bichromate, which does away with the excessive moisture and the sodium sulphate present in the chemical as now manufactured. He takes advantage of the fact that sodium sulphate is quite insoluble in a bichromate lye of 1·795 sp. gr. at 15°C. The weak lyes are concentrated until the normal chromate of soda begins to crystallize out. Sufficient sulphuric acid from 60 to 65°B. is then added to form the bichromate, thus yielding a lye of the required specific gravity. The sodium sulphate separates at once and is removed, and the lye evaporated to produce the solid bichromate.

* English Patent No. 25,020, Nov. 26, 1898.

Analytical Methods.

An error in the estimation of the chromium in ferrochromium is pointed out by a correspondent of the *Chemical News*. The general method of analysis is to heat the finely powdered alloy at a high temperature with basic reagents, in order to form chromates. The sintered or fused mass is placed with the crucible in a beaker with sulphuric acid, and finally the chromic acid is titrated with ferrous sulphate and potassium bichromate. As incomplete oxidation of the powdered alloy creates a double loss by reducing, when dissolved in the dilute sulphuric acid, some of the already oxidized portion, and as the amounts acted upon are small, the error is of importance. A precaution recommended is to boil the solution with permanganate of potash, destroying the excess by dilute hydrochloric acid.

Determination of Chromium in Steel.—R. W. Mahon* has somewhat modified McKenna's method of titrating with ferrous sulphate and potassium permanganate. The chromium is oxidized to chromic acid by potassium chlorate in a nitric acid solution. Mr. Mahon's method is as follows:

Dissolve 3 g. of the sample in 50 c.c. of concentrated hydrochloric acid, in a 400 c.c. lipped beaker, covering with a (five-inch) watch-glass. Boil down until little more than a moist cake is left. Add 50 c.c. of concentrated nitric acid, and boil for a few minutes when the copious evolution of nitrous fumes will have largely ceased. Remove from the light, and when somewhat cooled, add 4 g. of potassium chlorate, and resume the boiling, continuing until the solution is reduced in volume to from 25 to 30 c.c. Dilute to 300 c.c. with cold water, and add 15 c.c. of ammonia of 0·90 sp. gr. Mix thoroughly by stirring and filter the solution (first cooled if necessary) through a ribbed double paper, washing with cold water. Dilute the filtrate and washings to about 450 c.c., and titrate with standard solutions of ferrous ammonium sulphate, and potassium permanganate. The best, that is, the most accurate and simplest, way of standardizing these solutions is by means of a standard solution of potassium bichromate.

In testing solutions of known strength the maximum error was 0·03%. To avoid reduction of the chromic acid by the filter paper, a dilute and cold solution should be used. Excess of nitric acid should be avoided.

Estimation of Silicon in Chrome Alloys and Silica in Chrome Ore.—George Tate† describes a method for the estimation of silicon in ferrochromium and of silica in chrome ore:

One to two grams of ferrochromium or of ore are fused with about five times their weight of sodium peroxide in a nickel crucible. The crucible and contents, after cooling, are brought into water contained in a capacious nickel dish. The alkaline liquid so obtained containing chromate and silicate, together with oxides of iron and nickel in suspension, is treated with HCl in quantity insufficient to neutralize the alkali, and then evaporated to dryness in the nickel dish. The residue is scraped, as far as is practicable, from the sides to the bottom of the dish, and 40 c.c. of strong pure H_2SO_4 rapidly poured on. Fumes of hydrochloric and chlorochromic acids are evolved, and completely expelled by applying heat. The temperature is raised until the sulphuric acid begins to

* *Journal of the American Chemical Society*, November, 1899, p. 1057. † *Chemical News*, Nov. 17, 1899.

fume, thereby insuring the dehydration of silicic acid. After removal of the flame and cooling for a few minutes, water is cautiously added. To prevent undue corrosion of the nickel vessel, the turbid liquid is transferred to a porcelain dish, and, after making the bulk about 0·25 liter, boiled for 15 to 30 minutes, or until the sulphates have completely dissolved. The silica is filtered off, washed thoroughly, ignited in a platinum crucible, and weighed. The silica so obtained should be white, but, as a few milligrams of metallic oxides usually accompany it, the real weight of SiO_2 (from which the silicon is deduced) is obtained by determining the loss in weight occasioned by evaporation with HFl and one drop of H_2SO_4, followed by strong ignition.

It is advisable to test the purity of the reagents by conducting a blank test. The results are very concordant and apparently exact.

Impurities in Chrome-Yellow.—M. Willenz* points out the numerous adulterants of chrome-yellow. Not only lead sulphate, which is probably necessary for shading, but barium sulphate, calcium carbonate, calcium sulphate and clay are used in quantities. Some of the commercial chrome-yellow contained as much as 65% of lead sulphate. To determine the impurities in this pigment Willenz has found the following method simple and sufficiently accurate:

Assuming the most complicated instance, *viz.:* a lead chromate containing lead sulphate, barium sulphate, calcium carbonate, and clay, he proceeds by weighing out one gram of the finely ground pigment and treating it, at a gentle heat, with 100 c.c. of dilute (1·20) HCl added in three portions. After leaving to settle for a moment, the clear liquid is decanted carefully on to a filter, the residue and filter being washed with hot water.

The solution contains, in the state of chloride, all the calcium, the metal being then estimated in the condition of oxide. The sulphuric acid from the calcium sulphate, being also present, is thrown down as barium sulphate.

The residue is digested, at the ordinary temperature, with 50 c.c. of ammonium acetate solution (sp. gr. 1·04)—either neutral or slightly alkaline—and the clear liquid is carefully decanted, the residue washed with hot water, and the solution evaporated to dryness in a platinum capsule in presence of an excess of H_2SO_4. After driving off the ammonia the solid residue represents the totality of the lead sulphate contained in the chrome.

The other residue, consisting of the lead chromate, barium sulphate and clay in the sample is shaken up in 50 c.c. of water and boiled for 10 minutes with 25 c.c. of caustic potash solution (112 g. of KHO per liter). This converts the lead chromate into soluble potassium plumbite, leaving the barium sulphate and clay to be separated by the usual methods. The chromate acid is determined by the Bunsen iodimetric method, from a separate sample.

Recovery of Chromium from Waste Solutions.

F. Regelsberger† describes several means of utilizing waste products containing chromium, by means of electrolysis. (1) Oxidation in alkaline solution. By the action of hypochlorites or chlorine gas in the presence of alkalies, the oxides of chromium are converted almost completely into bichromate, but

* *Bulletin Association Belge des Chimistes.* † *Zeitschrift für Angewandte Chemie*, 1898, pp. 1123-1128.

electrolysis brings about the same conversion at one operation, provided an alkaline chloride is present in solution. The yield of chromic acid is not quite equivalent to the current used, but is highest when the solution contains a chromic salt, much lower when the chromium is present as hydrate, and still lower when free alkali is present. No diaphragm is necessary in the electrolytic cell, which may be a box lined with metal to serve as cathode, while an anode formed of woven wire is suspended in the electrolyte. The best results cited gave a yield of chromic acid equivalent to 80 to 88% of the current used. This method might form an advantageous after-treatment of the product obtained by calcining chromic oxide and lime, but it has the disadvantage of requiring costly platinum anodes. The solution must be free from organic matter, which would consume the current.

(2) Oxidation in acid solution: Instead of platinum, the anodes are formed of lead, and it is in fact the lead peroxide formed by the current on the anode surface which takes the place of the hypochlorite as the intermediate agent in the oxidation of chromic oxide to chromic acid. There is an advantage in working without a diaphragm, as any local accumulation of free acid or alkali is avoided, and by working with hot solutions an economy of current tension is obtained, as well as an apparently higher yield of product. A lead lined box suffices as an electrolytic cell and the concentration of the electrolyte may be equivalent to 100 g. of Cr_2O_3 per liter. The author thinks that the cost of regeneration of 100 kilos of Na_2CrO_4 by this process will not exceed 20 marks. It may serve for the profitable working up of many waste liquors and may influence the use of chromic acid batteries.

NOTES ON THE METALLURGY OF CHROMIUM.

BY W. BORCHERS.

THE metal chromium is not found native, but in combination with oxygen and other metals it occurs as crocoite, a lead chromate of the formula $PbCrO_4$, and chromite, $FeCr_2O_4$, a salt of metachromous acid. The latter mineral, in the form of irregular and crystalline masses varying from brown to black in color, is the only ore of chromium and its compounds are of economic importance. It occurs in Silesia, Styria, Moravia, Russia, Norway, New Zealand and the United States.

There is considerable variance in the chemical composition of these ores, the limits of which are approximately defined in the following analyses:

	%	%	%		%	%	%
Cr_2O_3	39·60	51·50	51·20	MgO	9·60	16·30	12·55
Fe_2O_3	.....	.....	1·45	CaO	1·30	.....	3·15
FeO	21·20	14·80	13·33	SiO_2	4·50	3·80	4·95
Al_2O_3	22·80	13·20	12·80	CuO	0·20	.....	.....

Treatment of Chromite for Ferrochromium.—The different methods of treating chromite are based upon the condition in which it is desired to have the final product, whether in the form of pure chromium or as the alloy ferrochromium. In preparing ferrochromium, the ore is first mixed with the following substances: Charcoal, 12 to 15%; resin or pulverized pitch, 6 to 7%;

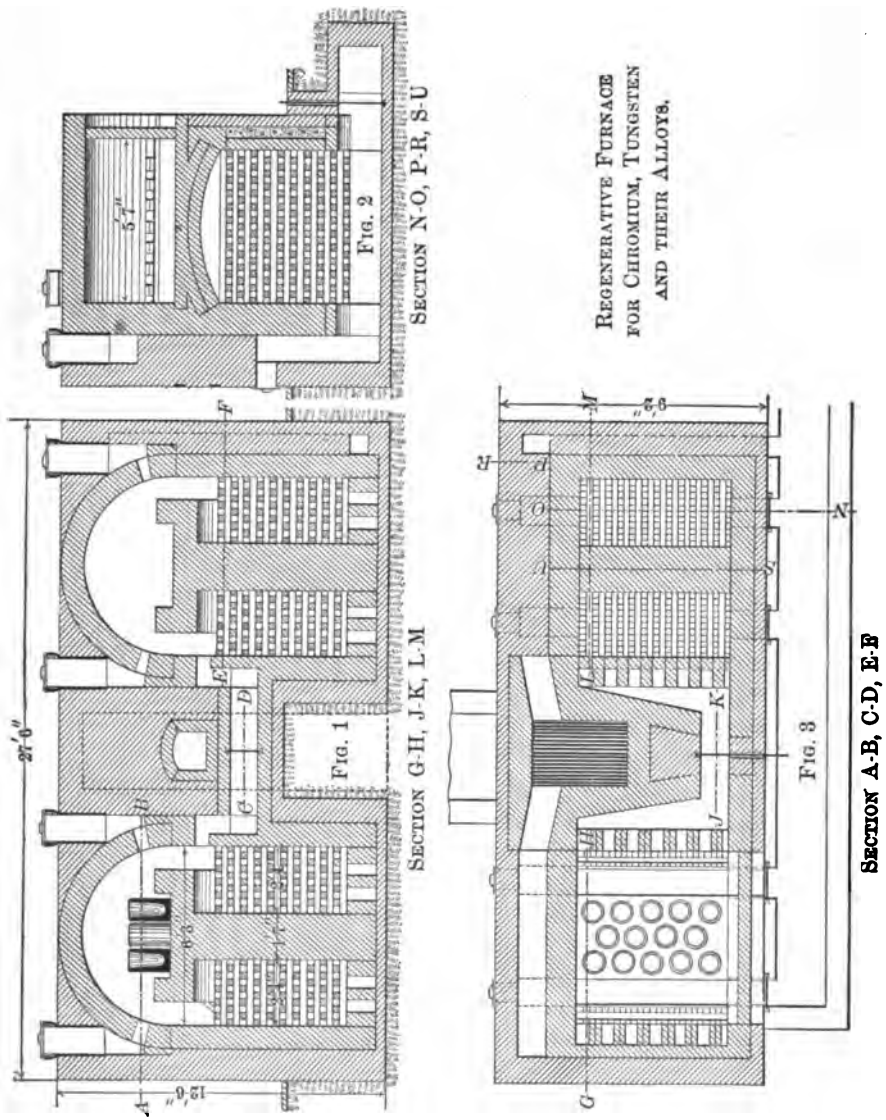
glass scrap, 5%, and quartz sand, 10 to 12%. An intimate mixture is desirable and can be best secured by piling the ingredients in a heap, each forming a separate layer, and then removing it in vertical sections to form a new heap. After repeating this process several times the mass will have a uniform appearance. The ore is then fused in a graphite or strong clay crucible, the bottom of which is covered with a thin layer of fine glass and coarse charcoal. The cover of the crucible is sealed with clay; only a small opening should be left for the escape of gases. The ore can be reduced by a crucible furnace with good draught, but to melt the metal a gas regenerative furnace will be required. Where a Siemens furnace for crucible steel is at hand this may be employed to advantage for smelting ferrochromium. A furnace of special construction for turning out chromium and tungsten, as well as their alloys, has been designed by the writer and has been in successful operation for several years past. This furnace, which is shown in Figs. 1, 2 and 3, consists of a generator in which coke is used, and two combustion chambers, each of which is connected with a pair of regenerators. The gas produced in the generator passes into a channel, whose position may be seen in Figs. 2 and 3, into which funnels are let down from above. When in operation, three of the funnels are closed by iron caps, while the fourth is connected by a U tube with one of the four funnels which are set in the gas channels along the sides of the combustion chamber. By this U tube one of the channels and, at the same time, one side of the chamber, is provided with gas. When air is allowed to enter from the regenerator, after having been heated there sufficiently it will kindle the gas, and a constant flame will be maintained around the crucibles. A second regenerator on the opposite side is meanwhile heated by the escaping gases.

When the furnace is started fresh, it is necessary at first to heat the chamber to such a temperature as to make the kindling of the gas certain. This can be best effected by placing a small amount of fuel in the generator and allowing the gas to enter the combustion chamber from one side only until the chamber begins to redden. More fuel may then be filled into the generator and after the gas has raised the opposite side of the chamber to a red heat, the flow of air and gas is reverted. The passage between the first regenerator and its flue is opened by turning a valve, and cold air is allowed to enter through the channel in the other regenerator. After closing the cold air entrance of the first regenerator, the U tube is placed on the opposite gas channel and the flue damper of the second compartment is closed. The funnels from which the U tube has been removed are then covered over. Within an hour at the most the flow must be again reversed to the first direction and the process is to be repeated every hour, or in shorter intervals. If the furnace was previously warmed, it will reach the melting point in a space of six to eight hours, which is the end of the operation, as the reduction takes place before the oxide comes to a melt.

After this the second chamber may be started. This is accomplished by opening the valves in the further compartment and in the flue connecting the two chambers, setting the U tube on the inside channel, closing the flue and the valve in the first chamber and opening the air channel of the latter. When a new chamber has warmed sufficiently, the gas must be reversed and the valve in

the connecting flue closed again. It is apparent that by this arrangement either chamber of the furnace may be charged or discharged without disturbing the work of the generator.

When the operation is completed the ferrochromium will be found as a cake



of gray metallic luster somewhat impure from inclusions of slag. This alloy is used almost exclusively for making chrome steel; some varieties of steel, however, require that the chromium be freer from carbon.

Treatment of Chromite for Metallic Chromium.—To prepare the metal, a long and roundabout process is involved, by which a separation of the iron and chromium must be effected before the chromic oxide can be reduced.

Breaking the Ore.—In Germany, the ore is first broken in a rock crusher and then further treated in a ball mill of the Grusonwerk pattern. This type of mill is to be preferred, as it separates the coarse and fine material and returns the former to the drum for further crushing. In chromate works of the United States, according to Lunge,* the so-called "Pneumatic Pulverizer" is used. The ore, after being broken to a coarse size in a rock crusher and rolls, is fed through small hoppers into tubes, which receive a jet of steam of 12 atmospheres pressure. From the tubes the ore is blown into a chest, first passing through blast pipes which are provided with disks pierced by holes of 3 mm. diameter. By the friction of the particles in the narrow chest a very fine pulverization is effected. The fines are carried out by the steam into a receiver, while the coarser material falls to the bottom and is again caught up by the jets. The final product is very even and needs no further treatment.

Oxidizing Chromite by Roasting with Basic Flux.—Although roasting with potash or soda is sometimes recommended in literature, it is not in accordance with actual practice. The purpose of roasting is to change the iron into insoluble oxide and at the same time to bring the chromium into a soluble condition as chromate. A fundamental principle in this process is, that there must be free access of air during the entire operation. Care must also be taken to avoid the presence or formation of easily melting products, although a sintering of the ore, so long as its porosity is not lessened, is considered beneficial. The use of lime alone as flux has met with some success, but more often it is mixed with soda. Potash is no longer employed for this purpose. The lime, after being pulverized finely is added partly in solution and partly as a powder. The roasting is carried out in a long-bedded reverberatory furnace or in the terraced-hearth Le Blanc furnace. The mixture of ore and flux is placed in the heating chamber as far as possible from the flames until thoroughly warmed, after which it is drawn to the hotter parts. A charge consumes about eight hours in roasting, during which a temperature of about 1,200°C. should be maintained. To heat the floor of the hearth from beneath, as has been sometimes recommended, is bad practice and greatly shortens the life of the hearth. Great care should be exercised in constructing the hearth floor, if interruptions are to be avoided during practice. The bricks intended for the floor should be laid dry on a bed of fire clay. On the upper terraces the floor need not have a thickness of over 125 mm., but for the lower terraces the bricks should be laid on edge so as to give double that width. The joints should be filled with fire clay.

Various proportions of flux and ore are used. The lime sometimes reaches as high as 150% of the ore, but may be as low as 50%; the alkalies are never more than 50%. The following formula shows the change which the ore undergoes during the roasting:

$$2FeCr_2O_4+4Na_2CO_3+7O=Fe_2O_3+4Na_2CrO_4+4CO_2.$$

Lixiviation and Separation of Chromium and Iron.—The product is now treated in iron kettles with hot water to which sometimes a little soda is added to convert the calcium chromate to sodium chromate. The sodium chromate goes into solution, while the iron oxide and lime, being insoluble, remain behind.

* *Zeitschrift für Angewandte Chemie*, 1894, p. 101.

Conversion of Chromate to Bichromate.—By evaporation the solution is reduced until it reaches a density of 1·5; it is then converted into sodium bichromate by adding sulphuric acid: $2Na_2CrO_4+H_2SO_4=Na_2Cr_2O_7+H_2O+Na_2SO_4$. It is customary not to carry the conversion to completion, 1 to 2% of chromate being allowed to remain neutral, as an excess of sulphuric acid would be injurious to the kettles. The solution is now evaporated to the point where the chromate hardens on cooling and is then poured out into shallow pans.

Reduction of Chromate to Chromic Oxide.—The hardened chromate is crushed and mixed with a sufficient quantity of sulphur and then placed in small cast-iron kettles of about 400 mm. diameter and depth arranged in series of six or eight over a simple grate. Iron pipes leading from the tops of the kettles serve to carry off the sulphur dioxide fumes. A small heat suffices to accomplish the reduction and to bring the mass to a melt. It is now scooped out with iron ladles and allowed to cool.

Separation of Chromic Oxide and the Alkali Sulphates.—By treating the melt, after pulverization, with hot water the alkali sulphates are dissolved; the insoluble chromic oxide is then recovered by decantation and filtration.

Treatment of Chromic Oxide for Chromium.—The chromic oxide after drying and grinding is ready for the final process by which it is reduced to metallic chromium. The reduction may be carried out by one of the following methods: (*a*) about 45% of pulverized charcoal is mixed with the oxide, and the mass then heated in crucibles in the previously described regenerative gas furnace or in an electric furnace. If the former be used, the metal is recovered as a fine powder, which contains some carbon both chemically combined and mechanically mixed. The melting point of chromium cannot be reached in the gas furnace, and if this is desired some form of an electric furnace must be employed. The writer first succeeded in melting chromium by means of a furnace which is described in Vol. IV. of THE MINERAL INDUSTRY. A process of freeing chromium from carbon, as proposed by Moissan, consists in melting the chromium with sufficient chromic oxide or calcium chromite to convert the carbon to carbon monoxide: $3Cr_4C+Cr_2O_3=14Cr+3CO$. The same principle is the basis of the Siemens-Martin process for making malleable iron. (*b*) An interesting method by which the oxide may be reduced without the aid of a furnace has been proposed by Goldschmidt. The chromic oxide is mixed with a quantity of aluminum sufficient for the reaction $Cr_2O_3+Al_2=Cr_2+Al_2O_3$; the mass is then placed in a crucible lined with magnesia, and ignited with some compound giving a high ignition temperature, such as barium peroxide and aluminum in the proportions: $3BaO_2+Al_2=3BaO+Al_2O_3$. Upon igniting with a magnesium band a violent reaction takes place throughout the entire mass and continues until, by adding successive portions, the crucible is filled with molten metal and aluminum oxide. After cooling the chromium can be recovered as regulus of great purity. (*c*) A process proposed by Aschermann is based on the fact that the oxide may be reduced by the sulphur contained in metallic sulphides. It requires a quick and high heat, which can best be secured in an electric furnace. If antimony sulphide is used, the melt will be composed of chromium, the antimony being volatile, $2Cr_2O_3+Sb_2S_3=2Cr_2+Sb_2+3SO_2$. With the fixed metals there will be an alloy, $3FeS_2+4Cr_2O_3=Fe_3Cr_8+6SO_2$.

CLAY.

THE past year has been one of the most successful in this great industry. Not only has production increased through extensive building requirements, but prices generally, in spite of competition, have remained firm. In common with other industries attempts have been made, some successful, others abortive, to consolidate producing interests in various localities. The consolidations which have been effected have doubtless tended to a better understanding between producers, and have helped to maintain prices at a profitable figure, if not at a high one. The production of brick and clay in the United States in 1898 and 1899 is summarized in the following tables:

SUMMARY OF PRODUCTION OF BRICK AND CLAY WARES IN 1898 AND 1899.

Kind.	1898.			1899.		
	No. of M.	Value.	Per M.	No. of M.	Value.	Per M.
Common brick	5,186,232	$26,235,626	$5·08	7,172,262	$37,642,105	$5·25
Front brick	401,691	4,158,067	10·35	886,448	3,261,717	8·49
Fire brick (a)	345,291	4,167,108	12·07	368,427	5,345,730	14·71
Paving and vitrified brick	571,542	4,635,736	8·11	658,005	5,637,571	8·68
Other clay building material (b)	.....	5,439,914	.....	.....	5,714,784	.....
Sewer pipe and drain tile	.....	7,493,428	.....	.....	e 8,242,771	.....
Crude clay, stoneware and misc. man'f'res (c)	.....	e7,250,000	.....	.....	e 8,200,000	.....
Totals	.....	$59,379,874	....	.....	$74,064,628	.....

(a) Not including silica brick. (b) Including terra cotta lumber, hollow building tile or blocks, roofing tile, floor tile and all other clay building material. (c) Including the value of common stoneware and various miscellaneous clay manufactures and crude clay used in pottery, for laying fire brick, in paper making, as burnt clay railway ballast, for the manufacture of gas retorts, glass pots zinc retorts etc. (e) Estimated.

The difficulties which attend a compilation of statistics of brick and clay can be well understood. To obtain those of the large producers is a comparatively easy task, but since it is one of the simplest and yet one of the widest in range of the mineral industries, it is most difficult to obtain returns from every individual producer. This difficulty is increased, moreover, owing to some of the individual producers not keeping records which will enable them to make accurate reports. At times, a brick making establishment produces only sufficient to complete certain contracts and then relinquishes business.

The production of brick and clay building material by States in 1898 and

PRODUCTION OF BRICK AND CLAY BUILDING MATERIAL IN THE UNITED STATES IN 1898.
(In Thousands.)

States.	Building Brick. Common.		Building Brick. Front.		Fire Brick. Alumina.		Paving Brick.		Other Clay Building Material.
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Value.
Alabama. (e)	50,000	$255,000	1,000	$11,500	4,000	$80,000	3,000	$45,000	$500
Arizona	10,623	69,062	e 200	e 2,500	.....	.....	.....	.....	.....
Arkansas	22,808	147,102	2,060	20,418	997	14,330	.....	.....	e 500
California	a 100,102	a 511,362	e 4,000	e 80,000	e 500	e 10,000	e100	e 1,000	.....
Colorado (e)	29,224	146,120	7,306	73,060	2,400	48,000	1,500	15,000	20,000
Connecticut (e)	125,000	657,750	2,500	20,000	3,500	70,000	.....	.....	78,812
Delaware	11,500	74,750	.....	.....	.....	.....	208	2,080	.....
Dist. of Columbia	34,580	197,791	.....	.....	.....	.....	100	900	.....
Florida	27,978	167,608	169	1,490	.....	.....	.....	.....	.....
Georgia	106,456	465,941	6,801	54,676	2,062	27,491	.....	.....	55,000
Idaho	6,467	38,692	2,456	24,560	.....	.....	.....	.....	.....
Illinois	481,778	2,312,974	22,105	221,869	8,269	86,422	88,366	719,496	e 750,000
Indiana	259,546	1,297,730	19,244	194,589	202	1,616	41,072	414,477	e 370,000
Iowa (b)	178,518	1,049,222	6,722	54,752	276	5,525	38,478	313,551	22,006
Kansas (b)	23,157	146,765	6,575	48,300	611	4,888	26,182	190,735	.....
Kentucky	77,624	319,250	90	900	17,180	251,517	1,899	18,629	e 90,000
Louisiana	81,052	408,609	e 5,000	e 42,500	.....	.....	.....	.....	1,500
Maine	61,493	279,097	874	5,244	.....	.....	.....	.....	.....
Maryland	96,355	537,449	4,651	55,251	24,455	231,606	5,227	48,233	38,695
Massachusetts	221,589	1,189,339	2,190	27,651	2,594	79,617	.....	.....	249,058
Michigan	120,626	467,118	15,726	82,533	.....	.....	84	510	9,787
Minnesota	120,797	571,417	5,917	65,482	.....	.....	169	2,028	39,989
Mississippi	37,452	201,257	416	2,915	.....	.....	.....	.....	e800
Missouri	121,882	530,918	9,439	79,097	11,703	146,738	22,558	182,243	e 93,166
Montana (b)	41,541	272,805	625	17,132	2,363	91,866	.....	.....	.....
Nebraska	65,719	405,364	e 2,500	17,500	.....	.....	2,380	25,752	e 5,000
New Hampshire	78,752	275,858	881	8,813	1,459	12,156	.....	.....	17,512
New Jersey (b)	344,387	1,578,963	35,963	824,820	21,620	461,127	200	2,800	1,910,857
New York	317,481	3,786,833	18,326	179,132	e 19,000	380,000	18,233	177,968	e 625,000
North Carolina	61,337	317,113	923	8,695	.....	.....	1,921	20,900	e 3,029
North Dakota	33,413	204,672	138	2,070	.....	.....	.....	.....	.....
Ohio	311,436	1,407,517	54,629	479,097	76,975	808,882	157,251	1,188,903	518,741
Oklahoma (c)	e10,000	66,200	.....	.....	.....	.....	.....	.....	.....
Oregon	10,049	64,499	867	9,825	.....	.....	.....	.....	.....
Pennsylvania	527,787	3,272,024	84,404	773,100	116,848	1,139,857	69,965	624,946	e 400,000
Rhode Island	e 30,000	e 157,800	1,000	8,000	.....	.....	4,000	48,000	12,000
South Carolina	e 50,000	e237,500	287	1,722	719	5,752	.....	.....	e 1,200
South Dakota	7,175	47,779	.....	.....	.....	.....	.....	.....	.....
Tennessee	74,536	371,848	4,005	28,643	e 3,000	e 24,000	e 3,200	24,512	.....
Texas	83,847	478,009	e 22,000	e 170,500	e 2,000	e 20,000	.....	.....	e 15,000
Utah	17,380	104,661	19,400	159,620	1,291	12,910	.....	.....	.....
Vermont	11,437	53,774	60	241	.....	.....	.....	.....	264
Virginia	96,208	579,688	20,097	215,354	567	5,803	11,215	108,142	e 40,000
Washington	11,799	73,985	e 1,000	e 12,000	.....	.....	.....	.....	e 2,500
West Virginia (b)	7,250	43,500	350	3,500	20,500	143,500	24,264	509,771	e 5,000
Wisconsin	87,749	438,901	9,685	69,616	e 200	e 4,000	20	160	3,500
Wyoming (b)	400	4,000	.....	.....	.....	.....	.....	.....	.....
Totals	5,186,232	$26,235,620	401,691	$4,158,067	345,291	$4,167,103	571,542	$4,635,736	$5,439,416

(a) Statistics reported by the California State Mineralogist. (b) Statistics of State Geologists; the figures for West Virginia differ widely from those for 1897, which is probably due to a change in the classification. (c) Includes production of Indian Territory. (e) Estimated.

1899 is presented in the tables under the caption of "Production of Brick and Clay Building Material."

Illinois.—In the vicinity of Chicago, where operations have been active until interrupted by the late strike, there are over 40 complete brick making plants with an annual capacity of not less than 1,000,000,000 brick. While the demand has been fairly active it has been less than the capacity of the works. Prices generally have ruled higher, as they have in the smaller towns of the State.

Minnesota.—At St. Paul, Minneapolis and Duluth the demand has increased more than double that of the previous year, and the brickyards had great difficulty in meeting orders. Stocks at the end of the year were almost completely exhausted; a change from previous years when from 12,000,000 to 15,000,000 brick had to be carried over. Activity in building and construction should

PRODUCTION OF BRICK AND CLAY BUILDING MATERIAL IN THE UNITED STATES IN 1899.
(In Thousands.)

States.	Building Brick. Common.		Building Brick. Front.		Fire Brick. Alumina.		Paving Brick.		Other Clay Building Material.
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Value.
Alabama (e)	57,000	$313,500	1,400	$16,800	7,000	$154,000	3,000	$45,000	$1,000
Arizona	10,450	72,120	125	1,750					
Arkansas	42,466	302,375	4,000	50,000					
California	a 125,950	a 754,730	e 3,000	e 66,000	e 8,500	e 20,000			
Colorado	35,068	245,476	11,652	97,667	3,503	71,882	6,125	60,250	43,000
Connecticut	104,561	505,325	5,023	47,300					57,327
Delaware (e)	12 500	87,500	250	2,250			40	400	
Dist. of Columbia	69,020	500,395	230	3,720			e 150	1,500	
Florida	38,157	209,477							
Georgia	145,000	580,000	17,558	307,649	2,200	26,400			e 60,000
Idaho	6,205	53,315							
Illinois	587,072	3,110,802	16,267	134,328	8,625	90,590	e 125,000	1,012,500	e 775,000
Indiana	437,206	2,615,224	7,334	39,382	9,364	154,582	e 45,000	270,000	e 375,000
Iowa (e)	200,000	950,000	8,500	771,400	880	8,800	40,000	320,000	25,000
Kansas (e)	45,000	283,500	7,000	56,000	650	5,500	30,000	247,500	
Kentucky	96,268	505,875	6,084	30,420	25,844	282,930	2,575	13,900	125,000
Louisiana	100,088	431,842	e 5,000	e 45,000			5,500	27,000	e 2,000
Maine	118,471	621,240	1,216	13,675			100	600	
Maryland (e)	140,000	714,000	5,500	60,500	7,000	84,000	6,000	54,000	53,499
Massachusetts	255,611	1,374,873	1,544	32,947	5,400	190,000			65,000
Michigan	178,000	729,800	16,500	99,000			3,500	38,500	e 7,000
Minnesota	155,608	856,987	12,050	80,600					37,000
Mississippi	25,220	163,832	2,140	10,554					
Missouri	228,062	698,417	21,474	76,820	e 15,000	e 180,000	e 30,000	e 300,000	e 275,000
Montana (b)	32,340	199,575	859	23,440	2,022	54,204	350	13,700	
Nebraska	115,206	738,840					5,000	37,500	
New Hampshire	69,000	356,250	875	8,750	1,500	30,000			
New Jersey (e)	375,000	1,737,500	37,000	603,000	30,000	690,000	250	3,500	2,200,000
New Mexico	2,128	14,231	1,333	10,640	266	2,793			
New York	1,221,398	5,865,115	32,097	264,276	e 30,000	400,000	e 30,000	e 360,000	e 675,000
North Carolina	75,890	308,320	e 950	8,550	800	7,200			e 1,500
North Dakota	23,220	141,750	3,780	37,800			1,425	38,262	250
Ohio	570,678	3,323,571	61,883	653,855	97,773	1,189,779	200,380	1,847,780	374,718
Oklahoma (c)	2,889	16,983							
Oregon	16,254	101,416	1,022	27,139	140	1,680			
Pennsylvania	760,206	4,330,618	71,823	1,073,418	115,236	1,651,596	76,809	916,266	e 500,000
Rhode Island (e)	e 35,000	e 192,500	1,200	10,400			4,500	54,000	15,000
South Carolina	76,550	331,036	1,316	8,360	600	6,000	500	2,500	
South Dakota	5,200	32,800	400	4,400			10	120	
Tennessee	119,161	639,881	3,980	30,500	110	4,240	10,532	e 150.160	1,600
Texas	126,644	602,507	2,922	22,702	714	26,404			840
Utah	27,295	191,625	2,000	16,000					
Vermont (b)	27,840	125,360							
Virginia	99,522	555,592	2,246	104,477			1,219	13,107	e 40,000
Washington	29,480	205,800					40	560	
West Virginia (e)	28,000	154,000			300	3,600	25,000	222,500	5,000
Wisconsin	124,923	795,750	4,915	33,240					
Wyoming (e)	450	3,600							
Totals	7,172 262	$37,642,105	386,448	$3,281,717	363,427	$5,345,730	653,005	$5,637,571	$5,714,734

(a) Statistics reported by the California State Mineralogist. (b) Statistics collected in co-operation with the Montana Geological Survey. (c) Includes production of Indian Territory. (e) Estimated.

increase the demand and production in 1900 to more than double that in the previous year. Tile and sewer pipe manufacturers also report a successful year's business and prophesy an increased demand in the current year.

Missouri.—All producers including brick, pottery, stoneware, sewer pipe and vitrified brick, report increased demand and production, although the demand has not been exceedingly rapid. Prices, however, have not advanced much above those ruling in the previous year. An agreement was brought about between the Hydraulic Pressed Brick Co. and other producers which gave stability to the market. In the State of Missouri there are about 350 brickyards, which have a capacity of fully 1,000,000,000 brick per annum. One company alone can produce, it is claimed, 155,000,000 brick per annum. Much sewer pipe is produced. Fire brick is of lesser importance, although the produc-

tion is considerable. Minor productions are stoneware, paving brick (the production of which is growing larger annually), terra cotta, drain tile, roofing tile and flower pots.

New Jersey.—One of the most important industries in the State of New Jersey is that of manufactured products of clay. Numerous deposits are found in various portions of the State. Although not all of a superior quality there are sufficient deposits of excellent clay in accessible localities. The year's business has been profitable and the fair prices have stimulated production. In addition to ordinary building brick a considerable amount of refractory brick is produced, which has a good demand in the various iron producing and manufacturing industries of the State.

New York.—The greatest producer of clay products in the United States is New York, and there are few counties which have not some representative of this class of industry. The Hudson River valley, however, contains deposits of clay suitable for brick and other manufactures, combined with good facilities for transportation. In this section of the State there are over 100 manufacturers, many of whom have extensive plants. Stocks at the beginning of the year were small, owing to the strike which suspended operations in the Haverstraw yards in the previous year. The demand was exceedingly active and perhaps stands without a precedent in the trade. From 15,000,000 to 25,000,000 brick a week were brought down during the season and found a ready market in New York and environs. It is strange considering the active demand that prices fell off, reaching nearly the minimum of preceding years. On the whole, however, business was fairly profitable. At the close of the season prices ranged from $5 to $5·50 for tide water brick (Haverstraw, Newburg). Later in the year a consolidation under the name of the American Brick Co. was effected of about 60 companies, doing business along the Hudson River, and it is expected that this combination will keep prices at a higher level than before.

Ohio.—At Cleveland, where the product amounts to about 80,000,000 brick a year, trade was satisfactory during 1899, but at Cincinnati business was dull, though showing an improvement at the end of the year. The numerous sewer pipe and tile manufacturers throughout the State reported increased demand and active business as well as firm prices for their products.

Pennsylvania.—At Pittsburg an association of brickmakers under the title of the Western Pennsylvania Brick Co., was formed during the year. The plants included in the consolidation, some 25 in number, have a capacity of 150,000,000 brick per annum. Beaver and Cambria counties showed an increased production, both of bricks and fire bricks. Fayette County also showed an increased production, particularly of paving brick. At Philadelphia the 90 yards in the vicinity of the city showed little progress during the year, owing to the slackness in building. The output in 1899 fell off, it is estimated, from 350,000,000 to 200,000,000 brick.

Notes on the Coarse Pottery Production of Great Britain.

By Wilton P. Rix.

The rapid increase of the manufacture of the coarser varieties of pottery in Great Britain has been due to a number of causes. The development of sanitary science, the rapid progress of chemical research and consequent large increase of chemical industries, the introduction of electricity for traction and lighting, have all called for an extensive use of pottery as a suitable material for various related purposes. In addition to these, the unprecedented activity in the construction of railways and dwellings has added to the demand for heavy goods, especially glazed bricks, to be used in the construction of subways, tunnels and other underground work. Simultaneously with this industrial activity, there has been a large increase in the output of coal and iron, with the effect of a gradual increase in the supply of fire clays which lie immediately beneath and around the Coal and Iron Measures. Many enterprising mine owners have thus been induced to encourage ceramic research in directions which tend to utilize an otherwise waste product, both cumbersome and costly to remove. Hence in some form or other, many mining enterprises have a clay-producing plant for the conversion of the troublesome spoil heaps into profitable ceramic products. For this reason the large increase of production is chiefly located in districts contiguous to the coal and iron-bearing beds.

The object of this article is to give the reader a rapid review of the various sections of this part of the ceramic industry. No description has been attempted of the finer classes of pottery which involve the use of ceramic compositions as usually understood, nor of the more extensive manufacture of ordinary building bricks from surface clays and marls in almost every district.

Clays.—While the extension of fine pottery manufacture has been due chiefly to the excellence of the clay deposits of Dorsetshire, Devonshire and Cornwall, the great addition to the output of coarse pottery was caused by the development of the fire clays of the Midlands and Northern districts. In earlier years it was the custom to get both coal and iron ore without regard to the accumulation of material in spoil heaps. Most of the important collieries now possess their own brick works, and work these waste materials into products of high merit. The chief material for the manufacture of coarse goods, such as glazed bricks, sanitary fittings, baths, sinks and mangers, is a fire clay of a refractory or semi-refractory nature, and it is only from this material that these wares can be made in large sizes that will not warp. The best deposits in England are in Stourbridge, Newcastle, Leeds, Durham and Halifax, all connected with the Clay Measures in the coal-bearing districts. These beds of clay are highly silicious and refractory. They differ entirely in constitution from those of the Midlands and Northern districts. The clays from the strata of Leicestershire and South Derbyshire are somewhat less refractory. Near Glasgow and Kilmarnock are fire clays, in most cases of highly refractory qualities, which have become the foundation of important manufactures.

Stourbridge.—The Stourbridge clays have the preference for the manufacture of glass-house pots and other refractory products. The fire bricks made from

these deposits are used throughout the iron-producing districts, and have maintained their superiority for a long time. The clay is also suitable for glazed work, although this output is less than in the Leeds and Yorkshire districts. The refractory nature of the clay is chiefly due to the complete absence of alkalies and the low percentage of iron oxide. (See Analyses 1 and 2.) The color of the clay is a dark slaty gray, inclined sometimes toward drab. It has some carbonaceous matter, with shells and much free silica. In some parts the texture is coarse. An inferior quality of this clay is almost black in appearance, of very plastic nature, and vitrifies at a low temperature. These very plastic clays burn to a glassy and vitreous body, and are unreliable as they tend to crack and crush.

Leeds.—The Leeds fire clays are found in a district south from Leeds, which extends from 4 to 5 miles wide and 8 to 10 miles long. The quality varies greatly in the different beds and some are well suited to the manufacture of bricks. The darker clay is obtained at a depth of 360 to 420 ft., and requires careful admixture with other materials to produce satisfactory results. Tender clays near the surface are of an entirely different nature, and in some cases are treacherous when worked at a high temperature from the liability of their form to collapse rapidly at the vitrifying point. (See Analyses 3, 4 and 5.) The lower portion of this bed contains an excess of iron stone. The iron nodules from which the Yorkshire cold-blast iron is produced, are usually found near these Measures and some of the clays are worked for glazed purposes, as tiles, faience, and art pottery, at the same time that the iron seam is worked. These clays are very free from iron stone and extraneous matter and the development of the glazed ware industry has been rapid and important. The output in the Leeds district is estimated at from 700,000 to 900,000 per week. The tender clays of this area shrink and warp considerably and require much care and practical experience in their manipulation. From 40 to 50% of sand or calcined clay must be added to give a suitable mixture. (See Analyses 6, 7 and 8.)

To show the variation of the material of this district see Analyses 9 and 10.

Halifax Bed.—The deposit is extensive and the main bed is hard. It is worked near Halifax and Elland and its character changes greatly toward the N.W. portion of the district, where it is made into salt glazed ware as pipes and sinks, while at the opposite end of the district it is of a highly silicious character similar to ganister. Its composition (see Analysis 11) shows its excellent quality.

Newcastle-on-Tyne.—A large quantity of fire bricks and fire clay is supplied from this district. The clays are not equal in refractory qualities to those of Stourbridge, as they contain more alkali and generally more ferric oxide. As these bricks are low in silica they are more suitable for use in kilns, where salt glaze products are manufactured. The quality of the clay varies considerably in the district, and even with the best it is customary to blend clays from other sources. Some varieties are used with success for white glaze sanitary ware The analyses of clays from Newcastle and Glasgow are given below. (See 12 and 13.)

Glasgow.—Many of the clays of this district are highly refractory, and their extreme hardness necessitates long "weathering" to bring them into condition for working. Toward the south the seams are more easily worked as the clay is more plastic, though less refractory. These clays are largely used for glazed brick purposes, while those of Glenboig are manufactured into fire bricks of well-known excellent quality. (See Analysis 13 for the less refractory seam.)

Teignmouth Clays.—The deposits of South Devon are of an entirely different character from those of the fire clay district. The refractory clay is found in small crevices and has been brought from the highlands of Dartmoor by the torrent streams to its present position, where it is generally found in conjunction with plastic clays. The refractory nature of this clay is chiefly due to the presence of a large proportion of free silica, and the absence of lime and alkali. (See Analysis 14.)

ANALYSES OF CLAYS.

	Stourbridge.		Leeds—Ordinary.			Leeds—Tender.			Leeds—Special.	
	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.
Silica	78·51	65·10	70·55	70·03	60·82	64·92	51·11	61·10	81·50	80·95
Alumina	21·89	22·22	20·77	16·55	26·45	25·53	34·6	28·75	11·07	7·31
Iron oxide	0·53	1·92	1·45	1·18	1·94	2·14	3·75	1·43	2·12	6·65
Lime	1·59	0·14	0·75	0·67	0·63	0·33	0·56	0·27	0·40	0·58
Magnesia	.....	0·18	0·24	.....	0·55	.....	.....	.....	.....	.....
Alkali	.....	9·86	.....	.....	.....	0·50	0·03	.....	1·30	1·39
Water etc.	2·48	.....	6·52	5·25	0·68	6·71	9·95	8·55	3·61	3·12
Organic matter	.....	.....	0·22	0·32	11·02	.....	.....	.....	.....	.....

	Halifax.	Newcastle-on-Tyne.	Glasgow.	Teignmouth.	Dorsetshire.	Devonshire.	Torrington.	Stourbridge.	Brierley Hill.	Glasgow.	Leicester.
	11.	12.	13.	14.	15.	16.	17.	18.	19.	20.	21.
Silica	79·6	65·00	49·51	77·30	48·99	52·06	74·3	65·10	51·80	66·16	70·58
Alumina	18·21	30·02	28·91	16·60	32·11	29·38	21·8	22·22	30·40	22·54	17·87
Iron oxide	0·53	1·92	1·59	1·90	2·34	2·37	0·7	1·92	4·14	5·31	1·33
Lime	0·54	0·33	0·36	.....	0·43	0·43	0·5	0·14	0·40	1·42	0·43
Magnesia	0·44	0·47	0·50	.....	0·22	0·02	.....	0·18	0·50	.....	1·04
Potash	0·69	1·29	1·22	.....	3·31	2·29	2·5	0·18	.....	.....	0·07
Soda		0·30									
Water, etc.	.....	.....	2·64	4·20	11·96	12·88	.....	10·07	13·11	3·14	8·68
Organic matter	.....	.....	14·02	.....	.....	.....	.....	.....	.....	.....	.....
Barium	.....	0·75	.....	.....	.....	.....	.....	.....	.....	.....	.....

STONE WARE.—The production of stone ware is of two kinds, the "salt glazed ware," and the "enamel" or "feldspathic glazed stone ware." The former is used chiefly for the manufacture of drain pipes, bricks, coarse sanitary ware, bottles, jars, and large pans or other apparatus for chemical manufacture, mangers, troughs, and street gullies. Insulators and some few forms of electrical apparatus are also produced, but against its use for this purpose some apparently unreasonable prejudice has of late existed.

The method of salt glazing has also been adapted to decorative purposes at the Lambeth art potteries, and has been brought to a high standard of perfection.

The enamel or feldspathic glazed stone ware, generally known as "Bristol ware," is largely used for bottles, jars, culinary and domestic wares, large pans and vessels for dyes and acids, kitchen sinks, sanitary fittings, electrical appar-

atus, channels and water filters. The best qualities of stone ware, both salt glazed and Bristol, are made from the vitreous stoneware clays of Devonshire, termed the "Bovey deposits," located mainly at Newton Abbott and Kingsteighton, and shipped thence from the River Teign. A similar deposit of nearly equal quality is obtained in the neighborhood of Torrington, North Devon, about 10 miles from Bideford, at which port it is usually shipped.

These clays produce a highly vitreous ware of good color, impervious to moisture when fired at Seger Cone No. 8 or 10, and are as a rule sufficiently free from iron and foreign matters to admit of use without a preliminary washing. The clays of Dorsetshire obtained in the district of Wareham, in the vicinity of Corfe Castle near Poole, are somewhat inferior in color, but produce a good vitreous ware. (See Analyses 15, 16, and 17.)

Some portion of the silica is uncombined and in a very fine state of division. In this form it lends itself admirably to the process of salt glazing; the high percentage of free silica enables the salt vapors to attack the surface of the ware readily, and thus produce a superior glaze, intensely hard and yet free from all danger of disintegration. These clays are shipped to Lambeth, Bristol, Liverpool, Glasgow, etc., for the manufacture of stone ware.

The districts of Chesterfield and Nottingham also furnish clays of high iron content and of inferior color, which although suitable for salt glaze stone ware, have not been used to any great extent.

Clays obtained mainly from the Coal Measures of the Midland and Northern counties of England and from the south of Scotland are used for the manufacture of the coarser articles, as drain pipes, bricks, etc. A few analyses of clays from various districts are quoted above. (See 18, 19, 20 and 21.)

By far the largest proportion of the capital employed is for the manufacture of drain pipes, sanitary fittings, and salt glazed bricks from the silicious clays of the Coal Measures. Many of these clays contain a sufficient amount of alkali and ferric oxide to produce a fairly impervious ware if properly manipulated. Of late considerable attention has been given to the fine grinding and careful tempering of these clays, and the demand of engineers for a minimum of porosity has tended to improve greatly the quality of these products.

To prepare the clays for manufacture into these wares, they are first thoroughly dried and passed though a sieve of 6 to 8 mesh to the inch; sufficient water is then added to the fine material to form a plastic mass, which is passed through a pug mill, once for ordinary wares and twice for special wares.

When the clays from the Coal Measures are hard they are weathered for some weeks or even months and usually tempered in a pan for 15 to 20 minutes before working.

These clays to be moulded on a wheel are blunged,* sifted and then brought to a plastic condition by means of a slip kiln or a filter press. The latter method is fairly economical, as the clays are less plastic than the ball clays and give up their moisture more readily with a consequent reduction in the cost of pressing.

The machines used for grinding are of three types:

* The term "blung" is applied to a levigating treatment of the material in a vat with mechanical stirrers.

(1) Geared rolls set in pairs over each other, generally two tiers, sometimes three.

(2) Mortar mills or revolving pans for dry grinding with perforated plates at the outer rim. The pans are stationary or revolving and are fitted with revolving rolls. The gearing is overhead or underneath.

This type of machine, with a solid bottom, is often used for the tempering pan. The clay when sufficiently tempered is removed through an opening in the side, or by means of a scoop mounted on a swivel which is so inclined that it delivers the contents into the corve or tip wagon placed at one side.

Stone Ware Pottery.—Bottles and smaller vessels are made on the throwers' wheel, jam jars and open ware are made on the jolly in plaster moulds. Large quantities of ink bottles and ginger beer bottles are made on the wheel, and have a salt glaze on the exterior and a feldspar glaze inside, or they may be glazed entirely with feldspar. A good thrower can make and finish 80 to 100 half-pint bottles per hour, or 15 to 20 1-gal. bottles per hour. The cost of making the gallon bottles varies in different districts, from 3s. 6d. to 5s. 0d. per hundred. In some factories automatic jollies are used, and require no attention except to place and remove the mould, to fill it with clay and to finish the inner edge of the vessel with the moistened finger. By this machine 36,000 pieces a day can be turned out by one maker and two assistants. Larger bottles are occasionally made by the jolly and finished by hand.

The kilns for firing these wares are usually circular in form and from 16 to 22 ft. in diameter. Up-draft kilns are still used in many localities, but are being replaced by the down-draft plan, which give better results, and a very great economy of fuel. When the articles are uniform in size and thickness and particularly when the material is open in texture, the time of firing and cooling is greatly reduced. Salt glazed drain pipes of 9 in. diameter or less are made chiefly of fire clay and may be placed in an 18 to 20 ft. kiln, fired, cooled and removed in seven days' time. The salt glazed wares of the Chesterfield district are generally fired in small wagon-shape kilns from 6 to 8 ft. wide, and from 10 to 15 ft. long. The firing is accomplished with a small access of air and the ware is therefore of a darker color. Much of the ware is glazed with an opaque white inside, and, as a rule, the clay is blunged and sifted before use.

Considerable progress has been made in the Southern districts with the firing of Bristol stone ware, which is now done in down-draft kilns of from 20 to 24 ft. diameter, in many cases without the use of saggers. The flame is allowed to play over to the crown of the kiln and then downward through the perforated bottom. Sufficient space for this purpose is left between the tiers of ware. Saggers are used in the Bristol, Chesterfield and Glasgow districts. The average ware is not, however, of a high standard. The production of 1-lb. and 2-lb. jam jars from this material is now very large in most of the Midland and Northern districts, and has almost replaced the use of glass. The jars are made by girls in jollies and are finished at the upper rim only. The output at some potteries amounts to 40,000 or 50,000 a day. The production of kitchen sinks and slop stones from this material is also very large, and extends

through all the districts. These are finished in one fire, as they are generally made in plaster moulds, although fairly successful attempts have recently been made by one or two firms to press them out by steam pressure.

The production of salt glazed insulators has lessened, as they have been replaced by the superior white iron-stone ware; still a large quantity is made, chiefly in the Derbyshire district, from the local clay which is extremely dense when washed.

A considerable trade is carried on in Bristol glazed stone ware for the more intricate electrical switch boxes and carriers for underground conduits. These are chiefly made from specially prepared mixtures of Devonshire ball clays, sometimes with the addition of flint and Cornish stone.* Vessels from 10 to 200 gal. capacity are made in the Lambeth and Southern districts from mixtures of Dorsetshire and Devonshire ball clays with fire clay from the Newcastle and Stourbridge beds.

Drain Pipes and Sanitary Stone Ware.—The manufacture of this ware originated in the Lambeth district nearly 60 years ago and has been steadily developed in Lambeth and in Devonshire and Dorsetshire. The impervious quality of this make is superior to that of the Midland and Northern counties produced from finely ground fire clay of a more or less refractory nature. Attempts have been made to overcome the porosity of the product by fine grinding and careful tempering. Pipes up to a diameter of 3 ft. are made from a mixture of this material with coarse "grog,"† although the demand for large sizes is chiefly limited to 30 in. diameter. A few of the Halifax bed clays lend themselves specially to the manufacture of large vessels of considerable thickness, as mangers, troughs and tanks, which are produced in salt glaze, 4 ft. to 5 ft. long, 2 in. to 2·5 in. thick and are free from warps and fire-cracks.

Salt Glazed Bricks.—The manufacture of these bricks is steadily on the increase in various localities, although the chief output is in the Leeds and Huddersfield districts, which furnish clays very free from iron. The bricks are chiefly used for plinths and wall-facings, and are more pleasing in appearance than the Staffordshire blue bricks, which they have superseded for many purposes. They are usually pressed by a method similar to that used for making white glazed bricks, and dipped in the green state in a single dip composed of ball clays and fire clays slightly stained with iron oxide. The kilns in which they are fired are, as a rule, similar to those for firing glazed brick. They are rectangular in form and vary in size from 8 to 9 ft. high and 12 to 18 ft. long, with a uniform width of 10 ft. The salting varies according to the character of the clay and dip; in some instances the fires are salted three times and in others five or six times.

Staffordshire Blue Bricks.—The clays suitable for the production of these goods are confined chiefly to the South Staffordshire district, although makers in other localities produce bricks similar in appearance from clays containing less iron oxide and of more refractory character. The color of the bricks is due

* Cornish stone is mined in England, and is a partly decomposed feldspar. It consists of a mixture of kaolin, feldspar, quartz and mica, and is extensively used as a flux for coarse pottery.

† Unglazed body of pottery.

to the strong reducing temperature at the finish of the firing; the large percentage of iron oxide present is converted into a metallic state to a distance of about 1 in. from the surface. The firing is brought to the point when the iron oxide and silica flux and cause the mass to become somewhat vitreous. The fusion on the surface is aided by the addition of salt to the fire-holes which vaporizes and attacks the brick's surface. The fires during this time are choked with a heavy charge of small coal breeze. Some makers dust the bricks with crocus martis or refuse dust from steel grinding, etc. These bricks withstand great crushing weight, and are highly impervious. They are extensively used for heavy buildings and for docks and their resistance to wear and tear has increased their use for railway copings and stable pavements.

The manufacture of roofing tiles and quarries is carried on by most of the local red brick and terra cotta yards. In the Broseley district, Shropshire, a clay occurs especially adapted to this purpose; it is more vitreous when fired at a low temperature, and yet retains its shape without warping. In addition it has the advantage of yielding a pleasing red-brown tint, much approved by architects. The lightness and imperviousness of these tiles permit their use with a light roof support. A machine for plain roofing tiles will produce 30,000 per day of 10 hours. The tiles are cut by wire from a long ribbon of plastic clay formed between two long rolls and supported by a band or webbing. The cutting is effected by a revolving drum which carries a series of iron wires. The tiles set on edge form 8 to 10 tiers in a square kiln, 10 to 14 ft. long by 8 ft. wide. Fire-holes are placed along one side, extending inward about 16 in. The kilns are usually placed in pairs and each pair is connected with one shaft for the draft.

The clays of this district are advantageously located on high ground in the valley of the Severn River, and admit of a gravity transportation of materials after they have been prepared on the upper floors of the works. Muffle kilns are built on the same level as the railway, 20 ft. above the River Severn. Very extensive works for the manufacture of red and buff paving quarries are located in the neighborhood of Ironbridge, and use the local materials for this purpose. In many cases the clay is weathered for the best quality of product.

Wall Tiling.—The process of pressing tiles from moistened clay dust originally introduced by Messrs. Mintons, of Stoke-on-Trent, and Mr. Singer of Vauxhall, has practically replaced the encaustic method of working in plastic material. The output of tiles in this country is increasing daily, and the demand seems to be limitless. The manufacture is largely carried on in the Staffordshire potteries, yet many factories exist in South Staffordshire, Leicestershir, Dorsetshire, Shropshire, Yorkshire, etc. The chief materials used are clays from the upper Coal Measures with mixtures similar to those for earthenware and iron stone. The clays are usually washed and dried upon a slip kiln or passed through a filter press and dried on racks. They are then ground fine enough to pass through a 30-mesh sieve and moistened on wet plaster slabs or by a spray of water or steam. The size of the tile is limited to 12 in. square; the greater proportion, however, is of a smaller size, 6 in. by 6 in. and 0·375 in. thick.

Glazed Bricks.—The material used for the manufacture of glazed bricks varies greatly. In some cases the clays are of a highly refractory nature with very limited shrinkage. When clays of this character are used the manufacture of refractory fire bricks is often associated with glazed brick making. When the less refractory clays are used for glazed bricks a portion of the material is often converted into salt glaze drain pipes or glazed sanitary ware. The methods adopted for the preparation of the material vary in accordance with the condition of the material when raised. In the lower seams the clays are extremely hard and rocky from the heavy pressure of the over-lying strata and require considerable weathering and preparation. Sometimes the excess of free silica reduces the plasticity to such an extent that heavy pressure is necessary to solidify the bricks. Generally the bricks are made by a semi-plastic method, although various processes are adopted in slipping and glazing. Few makers now produce "once-fired" goods as the percentage of loss is increased unduly in proportion to the saving in manufacture. Some makers slip before biscuiting, while others first work their clay into bricks by biscuiting at a moderate heat and then close the pores of the brick by sizing and dipping at once in slip and in glaze, subsequently finishing with a second fire.

In the process of grinding and tempering there is little variation of the methods. As a rule the clay has been weathered to some extent and sorted, if necessary. It is conveyed to the grinding mills of generally 9 to 10 ft. diameter with edge runners for crushing the clay. The crushed clay is sifted through perforations from 0·166 in. to 0·125 in. diameter in the floor of the grinding pans and is lifted into the second pan of similar construction but without perforations. In this pan the necessary water is added to the fine material and it is ground for about 15 minutes, according to the nature of the material and amount of tempering required. The quantity of clay produced by such an arrangement is from 40 to 60 tons per day of 10 hours. When the clay is sufficiently tempered, the swivel scoop is placed in position and the tempered clay is forced outward over the side of the revolving pan and projected into the truck placed in position to receive it. Sometimes the grinding pan is made with an opening in the upright side, which allows the clay to be thrown outward into the truck, placed at the side. The brick is roughly formed by hand and is ready for the press. In most of the larger factories, however, the plastic dust is passed direct to the hoppers of the brick-making machine.

These machines are of various constructions, the most efficient producing from 5,000 to 6,000 per day. Each brick is delivered from the machine and is ready to be dried. When this method is used, plastic material must be added to the mixture so that the brick will bind in a satisfactory manner. Some clays of more open texture cannot be handled by this direct method, but require more tempering and the addition of more water to prepare them for the first forming in the plastic state. In this "alternative" method, the soft clay is passed through a direct-acting pug mill, and is forced out in a long stream of the exact section of the required brick, the bricks are cut off in lengths by wires properly placed in a frame. When cut, the bricks are stacked on the drying shelves to a height of about 3 ft., and, in the case of semi-plas-

tic clays, they are biscuited before they are slipped and glazed. Soft clay bricks require to be placed on a drying floor before pressing, so that they will attain a stiffer consistency. These floors are sometimes heated by flues running the entire length, which carry the hot gases from a fireplace situated outside of the building. Other makers pass dry steam through pipes to the mouth of the flue. The bricks remain upon the floor from 8 to 10 hours, and are then moved and stacked in a cool shed for about 24 hours, and then generally covered for protection. The floors are sometimes formed of stone, sometimes of fire clay slabs, and occasionally of hollow bricks or pipes grouted solid on the upper surface and leveled with cement. After the bricks have stiffened they are conveyed to the press by barrows or by mechanical means and pressed into die boxes by steam presses. Any slight arris is carefully removed, and the finished bricks placed upon shelves, or taken to the dipper.

The process of dipping is carried on in various ways at different works, according to the series of operations entailed. Sometimes two or more coatings of slip in rapid succession are necessary; sometimes the first coat is brushed and the second dipped; at other times both are dipped. If the clays are inclined to be porous and open grained, it is considered desirable to brush the first coat on. After dipping, bricks are steadily dried, and passed to the kilns, which are usually rectangular in shape, and of the following interior dimensions: 8 ft. 6 in. to 10 ft. wide, 10 ft. 6 in. to 12 ft. high, and 30 to 32 ft. long. The bricks are protected from contact with the flame by false walls, which direct it upward toward the crown of the kiln, and then downward between the walls of bricks through openings in the floor, thence to the shaft by underground flues, which pass along the entire length of the kiln. A kiln of this description can turn out from 16,000 to 20,000 bricks at each firing, with a consumption of about 25 tons of best quality coal.

In some works in the North, glazed bricks are fired in Newcastle kilns, in which temporary flues along the floor are built in the setting of the bricks to be fired. These flues carry a portion of the heat, while a limited amount rises and passes over the upper interior surface of the kiln, the bricks being so arranged as to be entirely muffled from the flame. These kilns require a much longer time to fire, and do not give satisfactory results. The cooling occupies from 38 to 48 hours.

Firing by gas is not used in Great Britain for the manufacture of glazed bricks, and only in one or two cases for the production of fire bricks.

Sanitary Ware.—There has been a rapid increase in the number of makers producing this class of goods. The variety of form is steadily increasing, and the quality of ware has improved greatly during the last three or four years. The clays are carefully selected and sorted to free them from iron oxide, coal or shale. They should be highly refractory, and of small contraction. The analyses of these clays vary considerably. Analysis 12 of clay from Newcastle-on-Tyne may be taken as the best type.

When large pieces are to be made, care is used chiefly in the selection of the material; every piece is hand broken several times to insure the absence of all impurities. Some makers weather the clay well, others keep it covered.

The clay is ground in a manner similar to that used for the manufacture of bricks for large articles with a coarser mesh. The ground clay is mixed, tempered and incorporated with other material if necessary. Sometimes the clay is used pure; at others it is mixed with ground, calcined clay, and frequently with calcined bauxite as well. This latter material is highly refractory, and enables the maker to improve his bodies in this respect without the addition of silica, which is already present in excess. The contraction of bauxite is extremely small. The prepared clay is beaten down into a solid mass and allowed to remain in a special shed for some days to mellow. Large articles, as baths, are formed upon a mould or hump of the requisite interior dimensions, and completely covered with slabs of clay of the necessary thickness, carefully joined together. The outer mould or shell is then placed over the clay form, and when sufficiently set, the whole is turned over for the removal of the hump from the interior. The inner surface is then scraped smooth, carefully finished, and allowed to dry. The slip is then applied by hand with a wide and soft brush, and frequently this process is repeated until the desired thickness of body has been obtained. Sometimes each successive layer is burnished. The methods of slipping and glazing vary with the different clays. In some cases the slip is laid on the green ware, in others on the dry ware, while some makers adopt the plan of slipping on soft clay, which is then fired in biscuit, and subsequently glazed.

When set in the kilns all large goods are, as far as possible, set on end to prevent excessive contraction strains and are fired in muffles of special construction to withstand the high temperature and the weight upon the under flues. Owing to the thickness of the ware, great care is used both in drying and firing, and an additional time, generally from seven to eight days, is allowed for cooling.

It is necessary in these kilns to prevent the radiation of heat at the doorway, and for this purpose the door is of double construction with the included space heated by extra fireplaces and flues.

Glazed Sinks.—These are manufactured by several different methods, according to the material used. Originally, in the Lambeth district, they were made from a mixture of South Devon pipe clay with 30 to 35% of grog. The forms were glazed with feldspar, and finished in one fire at a heat corresponding to Seger cone No. 8 or 10. This material produces a good, strong, vitrefied article, but somewhat liable to warp. During the last 10 or 12 years the demand has gradually increased and the production has extended to the Midland and Northern districts, where tender fire clays are used to a great extent. These tender fire clays have an addition of grog, in accordance with their refractory nature, and the mixture or "body" so produced is treated by three methods:

1. The form is biscuited at Seger cone No. 5 to 8, and then glazed with a transparent frit in a muffle kiln at cone No. 03 to 04. The articles produced in this way are usually of a buff or cane tint.

2. The sinks are slipped on the inside with a white slip body, biscuited,

and again glazed with a transparent glaze at a heat corresponding to Seger cone No. 03 to 04.

3. The slip and glaze are both applied to the ware when in the green condition, sometimes on the inside only, sometimes on the outside as well, and the dried ware is usually fired at a heat corresponding to Seger cone No. 7 to No. 9. The glaze in this case is raw and contains much feldspar, with the addition of a small amount of frit to suit the temperature.

These goods are generally formed in plaster moulds, which are filled from the bottom; the article being thus made upside down. In some cases the clay is first formed into flat bats of a proper size, and joined into shape within a wooden frame placed upon a loose board.

When the clay body is coarse in texture the inner surface is usually coated with a finer sifted slip of the same material, which is burnished with leather as soon as it has stiffened sufficiently. A white slip when desired is usually burnished in the same way. The sizes vary from 20×14 in. to 4×2·5×2·66 ft., with a usual depth of from 4 to 6 in., and the output amounts at the present time to many thousand pieces per week.

Architectural Terra Cotta.—The use of this material has been greatly extended during the last 15 or 20 years, and the demand is still on the increase. The dark red color has the preference, and is often combined with red facing bricks. A large number of important buildings in London have been erected, however, in terra cotta of light buff, or reddish salmon tints, chiefly produced from the ball clays of the Poole Basin districts of Dorsetshire. Some manufacturers have lately added a small amount of salt while firing, which produces a tint nearer a drab brown. The clay most suited to the production of red is found in several very fine beds in the districts near Ruabon in North Wales. The buff colors are formed by an admixture of fire clay, and a small proportion of ball clay, with sufficient grog to produce a suitable fracture when fired. The color varies from a strong yellow ochre to a brownish red according to the district. As a rule, the bodies containing lime must be avoided, as the lime salts effloresce upon the surface of the ware while it is being dried and burned. These lime deposits tend to disintegrate the material when exposed to certain climatic conditions, especially when exposed to the sea spray. A small amount of barium carbonate added to the mixture, however, obviates this difficulty, as it causes the lime salts to become insoluble. Generally the blocks are made in small dimensions of a size approximating a brick rather than a stone. The blocks are usually hollow with cells, which are filled after firing with a cement or concrete made with coke breeze or other material. The ware is usually fired in circular, down-draft kilns, with an open fire. The size of these kilns varies from 18 to 24 ft. in diameter. The time of firing depends largely upon the material and thickness of the ware. Some manufacturers use a rectangular kiln of the muffle type. They are similar to those used for burning colored glazes, about 10 ft. wide, 12 ft. high, and 16 to 20 ft. long, with fire holes on one or both sides.

Glazed Terra Cotta or Faience.—Terra cotta is glazed with a colored lead glaze in a second muffle fire, and the use of this ware has been largely adopted

for architectural decoration. While glazed bricks have been preferred for exterior work of simple design, very elaborate schemes of interior decoration have been carried out in glazed terra cotta and tiles, for the complete ornamentation of many large hotels and public buildings; its permanence and cleanliness are in its favor. The material used is chiefly a mixture of various fire clays with 20 to 25% of ball clay, with an addition of grog to aid safe drying.

A considerable range of color is attained in the glazes which are generally added by dipping, although elaborate effects are occasionally produced by penciling with various glazes which harmoniously blend when fused. These glazes are fired at a low temperature about Seger cone No. 010.

A finer body similar to earthenware is used in the Staffordshire districts, and some parts of the Midlands for the production of finer and smaller sections. The use of this fine material, however, increases the liability to warp.

Wall Tiling.—The manufacture of glazed wall tiling has developed in connection with the architectural glazed terra cotta. The material used is either a finely ground tender fire clay or marl of a buff or ivory tint; or an earthenware body composed of ground flint, Cornish stone and ball clay. The glazes used are either soft majolica lead glazes, fired at Seger cone No. 011 to 09, or an earthenware glaze fired at Seger cone No. 05 to 03. The softer glazes are fired in the rectangular kiln, and the earthenware glazes are usually fired in a circular closed oven. The tiles are for the most part made by the dust method, in which the slightly moistened dust is compressed into a mould box by a screw plunger.

Coarse Red Pottery, White Inside.—There are few if any important potteries that produce this ware. Most of the makers are scattered in small establishments of one or two kilns, and work on a small scale. The wares are chiefly made of surface red marls and clays, and are slipped inside with white or ivory body similar to earthenware. The raw, plumbiferous glaze is applied after the ware is biscuited. The biscuiting and glazing is usually done in the same oven. The glazed ware is sometimes placed in saggers, often set open and protected from fire and from the steam which rises from the clay ware, by the mere choice of position.

Yellow Ware.—In the Derbyshire and the Leicester districts, and in a few other localities, a considerable output of the ware exists. It is made chiefly from tender fire clays obtained from the Coal Measures. The clays are thrown loosely upon the ground in 6 to 8 in. layers, and allowed to weather. Though hard when brought to the surface, they disintegrate, and when tempered and pugged yield a fairly plastic mixture. In most cases, however, it is blunged and sifted, and afterward treated similarly to white earthenware. Yellow ware is used for ordinary domestic purposes and for sanitary goods. Many of these culinary vessels are slipped inside with a white earthenware slip before biscuiting.

Art Pottery.—There are other small potteries that produce goods of characteristic peculiarities, which cannot be classified in the previous lists. Their importance, however, is limited.

COAL AND COKE.

THE great industrial prosperity of the United States in 1899 manifested itself in the enormous consumption of coal, the year's output being 228,717,579 metric tons (252,115,387 short tons), the largest on record. Compared with the production in 1898 of 198,071,199 metric tons (218,333,883 short tons) it shows an increase of 30,646,380 metric tons (33,781,504 short tons). The appended chart illustrates clearly the relative positions of the different coal producing countries. During the past decade the United States has gradually

TOTAL PRODUCTION OF COAL IN THE UNITED STATES. (IN TONS OF 2,000 LB.)

States.	1898.			1899.		
	Tons.	Value at Mine.		Tons.	Value at Mine.	
		Total.	Per Ton.		Total.	Per Ton
Bituminous:						
Alabama	6,466,741	$5,496,730	$0·85	7,484,763	$7,484,763	$1·00
Alaska (b)	1,300	6,825	5·25	2,300	12,282	5·34
Arkansas	a1,184,064	1,202,108	1·06	a918,743	1,233,553	1·35
California	135,795	334,999	2·47	167,161	430,631	2·58
Colorado	4,125,206	7,219,110	1·75	4,747,812	8,308,671	1·75
Georgia	240,000	156,455	0·65	203,775	183,081	0·90
Illinois	a18,599,299	14,567,598	0·78	a23,434,445	18,443,946	0·78
Indiana	5,435,896	4,729,230	0·87	6,158,294	5,542,402	0·90
Indian Territory	a1,458,098	1,822,622	1·25	a1,404,442	2,106,663	1·50
Iowa	4,117,359	4,759,967	1·16	4,675,000	5,937,350	1·27
Kansas	3,860,405	4,193,160	1·09	4,096,895	5,124,248	1·25
Kentucky	3,492,243	2,479,493	0·71	4,668,800	3,720,100	0·80
Maryland	4,621,618	3,142,700	0·68	5,060,248	4,318,211	0·85
Michigan	378,541	560,241	1·48	500,000	720,000	1·44
Missouri	a2,888,152	3,148,862	1·10	a3,191,811	3,582,111	1·12
Montana	1,450,971	2,180,390	1·50	1,409,882	2,227,998	1·58
Nebraska	500	1,750	3·50	1,000	3,000	3·00
New Mexico	a863,583	1,416,890	1·64	a1,049,034	1,600,588	1·52
North Carolina	6,144	8,602	1·40	26,994	37,792	1·40
North Dakota (b)	124,526	124,526	1·00	120,597	120,597	1·00
Ohio	14,053,899	10,680,910	0·76	16,695,949	14,191,557	0·85
Oregon	65,871	183,543	2·79	86,886	222,854	2·68
Pennsylvania	64,247,859	38,548,715	0·60	73,066,943	57,722,885	0·79
Tennessee	3,094,748	2,340,346	0·76	3,736,134	3,706,617	0·99
Texas (c)	751,191	1,141,810	1·52	940,622	1,646,088	1·75
Utah	571,417	583,084	1·02	882,496	1,553,193	1·76
Virginia	1,540,000	984,000	0·60	2,111,391	1,372,404	0·65
Washington (d)	1,988,288	3,301,978	1·66	1,917,607	3,355,812	1·75
West Virginia	16,499,840	9,715,754	0·59	a18,201,189	11,830,773	0·65
Wyoming	3,181,905	4,006,756	1·26	4,525,207	5,656,509	1·25
Total bituminous (Sh. tons)	165,435,389	$129,089,154	$0·78	191,501,350	$172,406,679	$0·90
Total bituminous (Met. tons)	150,082,000	.....	0·86	173,728,885	.....	0·99
Cannel:						
Kentucky (Sh. tons)	49,889	$134,700	$2·70	36,639	$91,597	$2·50
Kentucky (Met. tons)	45,259	.....	2·98	33,239	.....	2·76
Anthracite:						
Colorado	48,831	$134,285	$2·75	59,067	$162,434	$2·75
Pennsylvania	52,799,774	81,311,652	1·54	60,518,831	103,486,346	1·71
Total anthracite (Sh. tons)	52,848,605	$81,445,937	$1·54	60,577,898	$103,648,780	$1·71
Total anthracite (Met. tons)	47,943,940	.....	1·70	54,955,455	.....	1·89
Grand total coal (Sh. tons)	218,333,883	$210,619,791	$0·96	252,115,387	$276,147,056	$1.10
Grand total coal (Met. tons)	198,071,199	.....	1·06	228,717,579	.....	1·21

(a) Fiscal year. (b) All lignite. (c) One-third lignite. (d) One-half lignite.

TOTAL PRODUCTION OF COKE IN THE UNITED STATES. (IN TONS OF 2,000 LB.)

States.	1898.			1899.		
	Tons.	Value at Oven.		Tons.	Value at Oven.	
		Total.	Per Ton.		Total.	Per Ton.
Alabama	1,541,250	$3,082,500	$2·00	1,798,612	$4,676,391	$2·60
Colorado	445,925	1,226,294	2·75	455,783	1,253,403	2·75
Georgia	50,000	63,500	1·27	44,529	92,748	2·08
Illinois	3,000	4,950	1·65	2,000	3,300	1·65
Indiana	1,521	2,662	1·75	2,105	4,105	1·95
Indian Territory	*a* 34,810	78,323	2·25	*a* 29,362	95,426	3·25
Kansas	10,000	20,000	2·00	200,000	600,000	3·00
Kentucky	21,394	30,593	1·43	55,580	79,201	1·42
Missouri	5,440	10,709	1·97	4,560	8,808	1·93
Montana	70,235	529,825	7·54	59,072	401,779	6·80
New Mexico	*a* 2,275	5,119	2·25	*a* 33,661	110,046	3·27
Ohio	100,000	250,000	2·50	73,391	183,477	2·50
Pennsylvania	10,671,920	21,663,998	2·03	12,192,570	29,871,796	2·45
Tennessee	394,545	710,181	1·80	440,157	864,073	1·91
Utah	28,327	127,809	4·51	26,700	122,820	4·60
Virginia	525,000	703,500	1·34	555,507	1,027,688	1·85
Washington	62,720	219,520	3·50	31,283	117,311	3·75
West Virginia (*b*)	1,951,801	2,315,823	1·19	*a* 2,004,405	2,512,709	1·25
Wyoming	18,393	64,375	3·50	15,979	55,926	3·50
Total coke { Sh't tons	15,938,556	$31,109,680	$1·95	18,025,256	$42,061,092	$2·33
Total coke { Met. tons	14,459,363	.....	2·15	16,352,405	.....	2·57

(*a*) Fiscal year. (*b*) Includes 85,319 tons ($131,033) made in Wisconsin in 1898, and 40,537 tons ($156,067) in 1899.

UNITED STATES PRODUCTION OF COAL, 1895–99. (IN TONS OF 2,000 LB.)

Year.	Alabama.	Arkansas	California.	Colorado.	Georgia.	Illinois.	Indiana.	Indian Territory	Iowa.	Kansas.
1895	5,680,410	900,671	80,115	3,582,393	414,310	17,735,864	4,312,084	1,228,440	3,995,886	3,190,843
1896	5,747,698	889,750	70,649	3,371,633	246,012	*a*19,786,626	*a*4,068,124	1,225,333	4,115,000	3,191,748
1897	5,893,771	850,000	87,449	3,565,660	196,268	*a*20,072,758	4,439,489	*a*1,343,376	4,523,494	3,672,195
1898	6,466,741	*a*1,134,064	135,795	4,174,037	240,000	*a*18,599,299	5,435,896	*a*1,458,098	4,117,359	3,860,405
1899	7,484,763	*a*913,743	167,161	4,806,879	203,775	*a*23,434,445	6,158,224	*a*1,404,442	4,675,000	4,096,895

Year.	Kentucky	Maryland	Michigan	Missouri.	Montana.	Nebraska.	New Mexico.	North Carolina.	North Dakota.	Ohio.
1895	3,207,770	3,479,499	.....	2,283,081	1,104,854	.....	367,442	15,386	79,850	13,683,879
1896	3,128,818	4,062,716	85,465	*a*2,420,147	*a*1,157,722	.....	655,370	14,970	94,500	12,912,608
1897	3,304,058	4,411,924	152,850	*a*2,429,388	*a*1,607,637	500	*a*729,653	20,611	120,000	12,465,533
1898	3,542,132	4,621,618	378,541	*a*2,838,152	1,450,971	500	*a*863,583	6,144	124,526	14,053,899
1899	4,705,489	5,080,248	500,000	*a*3,191,811	1,409,882	1,000	*a*1,049,034	26,994	120,527	16,695,949

Year.	Oregon	Pennsylvania.		Tennessee.	Texas.	Utah.	Virginia	Washington.	West Virginia.	Wyoming	Totals. (*b*)
		Bituminous.	Anthracite.								
1895	65,918	51,813,112	58,660,657	2,319,720	499,668	530,713	1,083,229	1,184,619	12,355,113	2,197,914	189,053,890
1896	90,715	50,273,656	48,074,330	2,663,714	585,000	508,243	1,018,237	1,295,313	13,509,964	2,273,928	187,657,250
1897	100,929	54,674,272	52,581,036	2,898,849	615,540	506,455	1,418,746	1,489,815	13,950,000	2,744,980	200,857,211
1898	65,871	64,247,859	52,799,774	3,084,748	751,191	571,417	1,640,000	1,968,288	16,499,840	3,181,905	218,333,833
1899	86,886	73,066,943	60,518,331	3,736,184	940,622	882,496	2,111,391	1,917,607	*a*18,201,189	4,525,207	252,115,387

(*a*) Fiscal year. (*b*) Includes from Alaska, 1,300 in 1898 and 2,300 in 1899.

EXPORTS OF COAL AND COKE OF DOMESTIC PRODUCTION. (IN LONG TONS.)

Year.	Anthracite.		Bituminous.		Totals.		Coke.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1895	1,470,710	$5,987,130	2,211,983	$4,816,847	3,682,693	$10,753,977	117,298	$424,174
1896	1,350,000	5,925,506	2,276,202	5,072,818	3,626,202	10,998,324	151,062	553,600
1897	1,298,768	5,836,730	2,399,263	5,326,761	3,698,031	11,163,491	173,034	546,066
1898	1,350,948	5,712,985	3,152,457	6,699,248	4,503,405	12,412,233	199,562	600,981
1899	1,707,796	7,140,100	4,044,354	8,573,276	5,752,150	15,713,376	230,196	858,856

IMPORTS OF COAL AND COKE INTO THE UNITED STATES.

Year.	Coal.					Coke.		
	Anthracite.	Bituminous	Totals.			Long Tons.	Metric Tons	Value.
	Long Tons.	Long Tons.	Long Tons.	Metric Tons	Value.			
1895	141,337	1,229,109	1,370,446	1,392,373	$3,970,761	26,448	26,871	$71,366
1896	101,689	1,246,991	1,348,680	1,370,259	3,725,832	43,372	44,066	114,713
1897	3,251	1,276,963	1,280,244	1,300,728	3,432,154	35,193	35,756	98,558
1898	3,149	1,270,557	1,273,706	1,294,085	3,578,181	41,185	41,844	142,334
1899	61	1,393,579	1,393,640	1,415,938	3,858,544	27,855	28,301	142,504

PRODUCTION AND CONSUMPTION OF COAL IN THE UNITED STATES. (IN LONG TONS.)

Year.	Production.	Imports.	Total Supply.	Exports.		Consumption.	
				Domestic.	Foreign.	Tons.	Metric Tons.
1895	168,797,669	1,370,446	170,168,115	3,682,698	3,440	166,481,982	169,145,694
1896	167,551,116	1,348,680	168,899,796	3,626,202	5,204	165,268,390	167,912,684
1897	179,336,796	1,280,244	180,617,040	3,698,081	5,297	176,913,712	179,744,331
1898	194,940,967	1,273,706	196,214,673	4,503,405	2,890	191,708,378	194,775,712
1899	225,108,094	1,393,640	226,496,664	5,752,150	6,806	220,737,708	224,269,511

approached the foremost place, and has finally taken the lead in 1899. The United States and the United Kingdom together produce about 60% of the world's output of coal.

Of the coal exported nearly all anthracite and bituminous is shipped to Canada. The increase in exportations during 1899 was due to the increasing demand in South American countries, whose usual supply of Welsh coal was shut off by the strikes in that country and by the increased demand of the home consumers. The greater part of the importations was received on the Pacific Coast from British Columbia and Australia, while considerable importation was made to Atlantic Coast ports from Nova Scotia.

The production of coal in 1899 has been but little hindered, as the strikes that occurred were of minor importance and of short duration, except the one in the Southwestern coal field, comprising mines in Arkansas, Indian Territory and southeastern Kansas which interfered materially with the production from this field and had not been settled at the close of the year. There has been a considerable tendency toward consolidation of interests during the year. In the Pennsylvania anthracite trade over 60% of the output is now under the Morgan control.

In West Virginia the principal collieries of the New River field have passed under the direct control of the Chesapeake & Ohio Railway by the formation of the New River Coal Co. In Ohio the Hocking Valley and the Wheeling & Lake Erie roads have been reorganized, while the Pennsylvania has secured control of the Cleveland, Akron & Columbus Railway. By far the most important consolidations of the year, however, have been in the Pittsburg district where the interests of the operators who ship by rail were consolidated into one company and the interests of the river shippers into another. The Pittsburg Coal Co., the railroad consolidation, was formed with a capital stock of $64,000,000; one-half preferred and one-half common. This company is composed of 104

concerns and began operations on October 1, 1899. At that time 1·25-in. coal was selling as low as 85c. per ton at the mines. Since the consolidation the price has advanced to $1·25 per ton. The river coal combination resulted in the formation of the Monongahela River Consolidated Coal & Coke Co. with a capital stock of $30,000,000, divided into $10,000,000 in bonds, $10,000,000 in preferred stock and $10,000,000 common stock. The combination began business October 1, 1899, when the price of 1·25-in. screened coal delivered at Pittsburg was about 95c. a ton. The price has since been advanced to $1·30 per ton.

Purchasers of coal lands were unusually active during the year. The steel and iron companies have acquired reserves of fuel supplies and have bought many thousand acres of coal lands.

Alabama.—Owing to the very prosperous state of the Southern iron industry Alabama has had an exceptionally good year, and its output in 1899 was 7,484,763 short tons against 6,466,741 for 1898.

Illinois.—The second place among the coal producing States in 1899 was held by Illinois. There were some labor disturbances previous to July, which, however, was offset by the heavy production during the last half of the year. The total for the fiscal year ending June 30, 1899, was 23,434,445 short tons, against 18,599,299 tons for 1898.

Ohio.—During a large part of 1898 the mines in the Hocking Valley region were closed. The improvement which came in the last months of that year continued through 1899, and the output has been heavy. There has been no serious disturbance to production except the short supply of railroad cars. The total output for 1899 was 16,695,949 short tons, compared with 14,053,829 short tons for 1898.

Pennsylvania.—This State remains the first in coal production. There was a six-months' strike among the employees of the Susquehanna Coal Co., some short labor disputes in the Pittsburg district and some larger ones in the northwestern field of the State. The only large strike, however, was in the Clearfield region, where, in July, many thousand men were out for a few weeks. The output of bituminous coal in 1899 was 73,066,943 short tons against 64,247,859 in 1898. The anthracite production in 1899 was 60,518,331 short tons, against 52,799,774 in 1898.

PENNSYLVANIA SHIPMENTS AND PRODUCTION OF ANTHRACITE COAL. (IN LONG TONS.)

Companies.	1896.		1897.		1898.		1899.	
	Tons.	%	Tons.	%	Tons.	%	Tons.	%
Philadelphia & Reading	9,019,533	20·9	8,395,411	20·2	8,219,814	19·6	9,688,509	20·3
Lehigh Valley	6,749,128	15·6	6,425,927	15·4	6,885,577	16·5	7,588,932	15·9
Central of New Jersey	4,999,003	11·6	4,730,860	11·4	4,626,387	11·0	5,392,550	11·1
Delaware, Lackawanna & Western	5,627,533	13·0	5,690,684	13·7	5,795,540	13·8	6,372,401	13·5
Pennsylvania Coal Co	1,728,972	4·0	1,777,841	4·3	1,854,515	4·4	2,347,061	4·9
Delaware, Hudson & Lackawanna	5,835,621	13·5	5,646,853	13·5	5,613,186	13·4	6,490,050	13·6
Other lines	9,217,698	21·4	8,970,988	21·5	8,904,733	21·3	9,850,706	20·7
Total shipments	43,177,485	100·0	41,637,864	100·0	41,899,751	100·0	47,665,208	100·0
Used and sold at mines	3,022,424	.....	5,309,490	.....	5,242,904	.....	6,869,021	.....
Total production, long tons	46,199,909	.....	46,947,354	.....	47,142,655	.....	54,034,224	.....
Total production, short tons	51,742,898	.....	52,581,036	.....	52,799,774	.....	60,518,331	.....
Total production, metric tons	47,939,098	.....	47,698,512	.....	47,896,937	.....	54,898,772	.....

West Virginia.—The most striking development of the year has been shown in this State, the older mines which supplied the Atlantic seaboard trade being unable to supply the large demand in the latter part of the year. Many of the small operators of the Pittsburg district after the consolidation of interests there, established themselves in this region and prepared to open new mines. The total production in 1899 was 18,201,189 short tons compared with 16,499,840 in 1898.

Other States.—There were increased outputs in nearly all of the States east of the Mississippi other than those already mentioned, although labor troubles in the mines of Kentucky and Tennessee have interfered to some extent with the production. The States west of the Mississippi show increased outputs due to the general conditions of prosperity except Montana, whose output was a little less than in 1898.

COAL PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD. (IN METRIC TONS.)

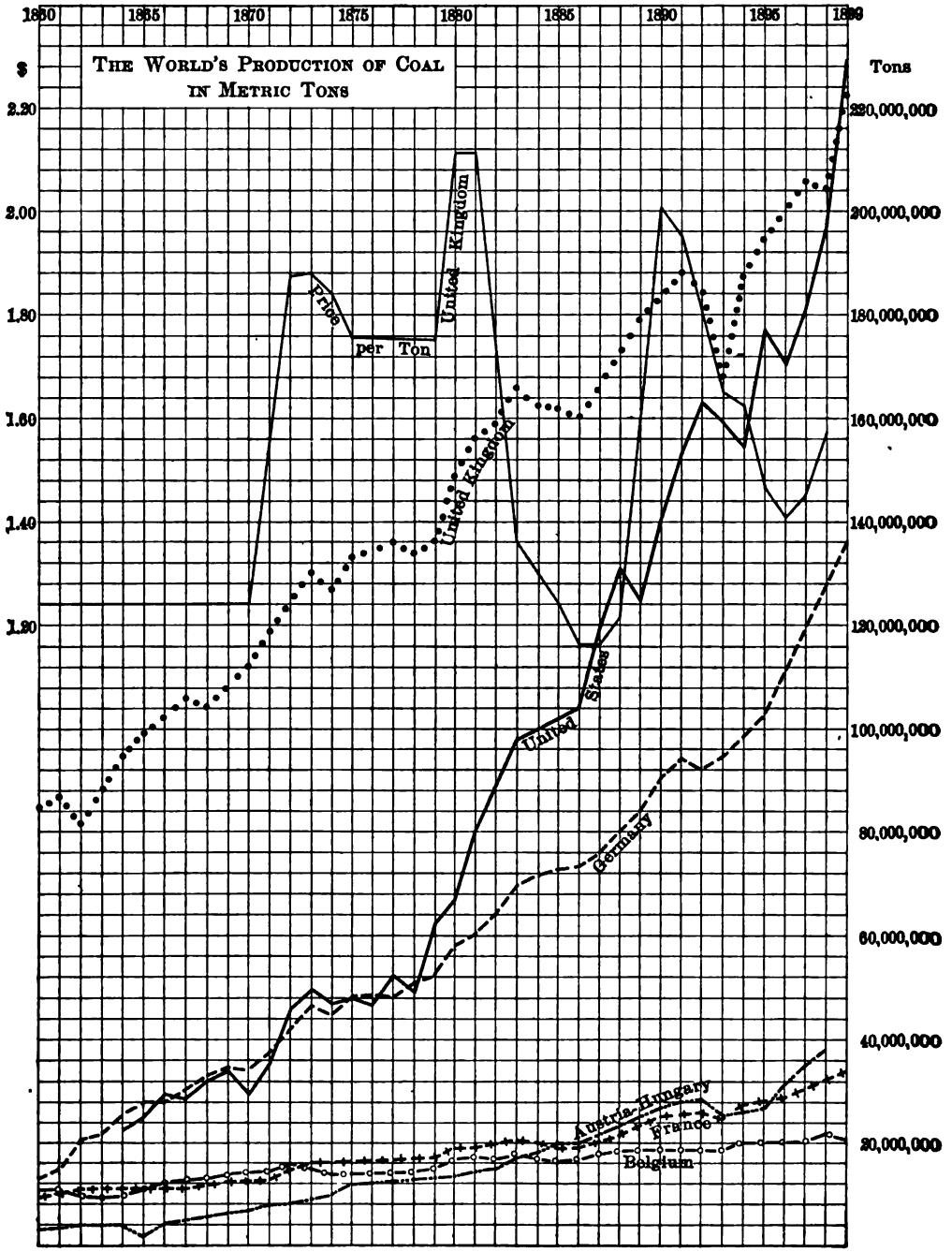
Year.	Africa.	Australasia.					Austria-Hungary	Belgium.	Canada.		France.	Germany.
		New South Wales.	New Zealand.	Queensland.	Tasmania	Victoria.			Alberta, British Colu'bia	N. Brunsw'k, Nova Scotia.		
1894	1,015,653	3,780,829	745,000	275,086	31,052	174,407	26,905,490	20,584,501	1,327,250	2,168,340	27,416,905	98,805,702
1895	1,402,182	3,500,000	760,000	275,000	33,000	204,000	27,250,000	20,414,849	1,408,911	1,782,732	28,236,039	108,957,639
1896	1,787,908	3,972,068	805,537	377,322	44,286	230,187	33,676,411	21,252,370	1,188,682	2,259,168	29,190,000	111,471,106
1897	2,003,174	4,453,728	854,164	364,142	42,910	240,063	35,289,416	21,492,446	1,178,010	2,383,567	30,798,000	120,474,485
1898	a2,550,485	4,781,551	921,546	414,461	49,902	246,845	37,785,902	22,088,335	1,461,882	2,323,546	32,356,107	127,958,550

Year.	India.	Italy.	Japan.	Russia. (f)	Spain.	Sweden	United Kingdom.	United States.	All Other Countries	Totals.
1894..	2,620,910	271,294	4,329,397	8,763,340	1,659,264	213,634	188,277,525	154,211,308	1,600,000	543,811,506
1895..	2,650,000	250,000	4,843,936	9,098,477	1,774,560	205,000	194,350,604	177,595,679	1,750,000	580,186,311
1896..	3,909,582	276,197	5,100,005	9,377,551	1,852,947	225,848	198,487,040	170,242,657	e2,000,000	597,676,866
1897..	4,128,137	314,222	5,647,751	11,208,738	2,019,000	234,343	205,364,010	182,216,466	e2,000,000	633,342,072
1898..	4,678,640	341,327	e6,000,000	13,000,000	2,466,800	236,277	205,297,388	198,071,199	e2,500,000	665,590,783

(a) Includes estimate of 50,000 tons as the output of the Orange Free State, for which no statistics are available. (e) Estimated. (f) 1898 production estimated.

Canada.—The collieries at Fernie, B. C., were opened in 1899. The coal is reported to be of excellent quality for coking and 150 coke ovens have been constructed. The coke is in great demand to supply numerous smelters in British Columbia, Montana, Idaho and Washington. On the west coast of British Columbia and on Vancouver Island the coal mines have had a prosperous year.

At Cape Breton, N. S., the Dominion Coal Co. has reported an increased output for 1899 and the formation of the Dominion Iron & Steel Co. with the erection of blast furnaces at Sydney insures a heavy output in the future. Other Nova Scotian mining companies have increased their output chiefly on account of the high ocean freight rates and the shortage of soft coal along the Atlantic seaboard of the United States in the latter half of the year, which led to considerable importations of Canadian coal at points east of Cape Cod. The output of the Nova Scotia mines for 1899 was 2,397,109 metric tons, against 2,069,712 in 1898. The total production of coal in Canada in 1899 was 4,141,424 metric tons.


THE WORLD'S PRODUCTION OF COAL IN METRIC TONS
1860
1865
1870
1875
1880
1885
1890
1895
1900
$
2.20
2.00
1.80
1.60
1.40
1.20
Tons
220,000,000
200,000,000
180,000,000
160,000,000
140,000,000
120,000,000
100,000,000
80,000,000
60,000,000
40,000,000
20,000,000
Price per Ton
United Kingdom
United Kingdom
United States
Germany
Austria-Hungary
France
Belgium

Mexico.—In Mexico the Southern Pacific Railroad, which now relies largely on Utah and California coal, has been searching for coal in Sonora, and has reported fairly satisfactory results. Large collieries are now being opened on the Conquista Coal Railway Co., which has recently been constructed to connect the mines, 15 miles distant, with the Mexican International Railway. The production in Mexico in 1899 was 409,125 metric tons, against 367,193 in 1898.

South America.—Good deposits of coal are known to exist in Peru and Colombia, but the existing labor conditions are not conducive to an early development of this industry. In the Argentine Republic and in Chile there has been considerable progress in mining during the year. The production still falls short of the domestic production and large amounts have to be imported—most of it generally has been shipped from Australia, but this year large shipments have gone from the United States. Deposits of coal and lignite occur in Patagonia and Terra del Fuego which, when systematically developed, will doubtless furnish a considerable output for the shipping trade. The imports of coal into Chile are shown in the following table, reported in long tons, for which we are indebted to Jackson Bros., of Valparaiso:

Year.	Steam Coal.						Smelting Coal.		
	Hartley.	Orrell.	Other Classes.	Australian.	North American.	Total.	English.	Australian.	Total.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
1895.....	189,119	16,412	157,498	227,014	.....	590,043	23,809	41,988	64,797
1896.....	151,547	9,254	115,024	290,073	.....	565,898	14,889	38,186	53,075
1897.....	126,909	.....	52,979	296,173	.....	476,061	7,646	50,172	57,818
1898.....	134,177	8,591	100,480	270,085	10,608	523,941	8,599	42,763	51,362
1899.....	125,042	7,633	151,325	307,119	3,200	594,319	24,073	43,277	67,350

Europe.—In England the attempts to extend the limits of coal supply still continue. The most notable development in 1898 was the exploration in southeastern Kent in the hope that coal bearing strata might be found extending across the southeastern portion of the island and ultimately connecting with the French coal field. During 1899 some thin coal seams have been found at a great depth. The results as yet do not warrant any development by sinking shafts, but they serve to show that there is probably an area of workable coal which will ultimately be developed. The larger part of the increase in English coal production was derived from the Welsh collieries. The total production of coal in England in 1899 was 220,085,303 long tons against 202,042,243 tons in 1898.

Belgium's output for 1899 was 21,917,740 metric tons.

France produced 32,779,965 metric tons of coal (including lignite) in 1899, against 32,356,107 tons in 1898.

Germany produced an output in 1899 slightly in excess of the production in the previous year. The amounts in metric tons for 1899 and 1898 were respectively 135,824,427 and 127,958,550.

Russia is passing through an industrial crisis. While the statistics for 1899 are not yet available, it is believed that the production will be increased. The output in 1897 was 11,203,738 metric tons.

Spain has shown an increase both in the production and importation of coal in 1899. The output for this year was 2,742,389 metric tons.

In Asia little has been done in opening up new sources of fuel supply during the year. The Shan Tung mines have seen some active development and have probably made an increased output, but no figures are as yet available to show the amount. In China itself little has been done. The Japanese output in 1899 should show a substantial gain, as improved methods of mining are being introduced in the collieries.

The Australian collieries have shared in the world's activity and all show substantial gains in the production of coal as compared with 1898.

Sumatra.—The Ombilein coal fields are extensive and the coal is of a good quality. The mines, a few miles' distant from Padang, are worked by the Dutch Government; convict labor is employed and the coal is delivered in ship's bunkers at Emma Haven, at a price ranging from $4·42 to $5·63 a long ton. It is expected the production will reach 15,000 tons a month, the greater portion of which will be consumed by the steamship lines.

Africa has suspended mining operations on account of the war between the Boar Republic and the British Empire. What little coal is now mined is consumed by the railroads, and many of the principal mines are closed entirely.

THE ANTHRACITE COAL TRADE OF THE UNITED STATES IN 1899.

The poor condition of the coal trade during 1898 has been more than offset by the excellent showing in 1899. The mining and transportation companies harmonized their interests with very satisfactory results. The firm of Drexel, Morgan & Co. took up its option on the Lehigh Valley Railroad and the Delaware, Lackawana & Western system was overhauled. The good feeling that existed between the Morgan and Vanderbilt interests, together with the friendliness of the Pennsylvania Railroad, resulted in the heaviest year's output yet recorded. Prices were higher than in any year since 1893.

The consumption in January was heavier than at any time in 1898. Coal at New York sold freely. Broken was quoted at $3·20, egg at $3·40, stove at $3·55 and chestnut at $3·50. February brought cold waves which started buying. It was seen that stocks at the head of the lakes and in Chicago territory would be all cleaned up before navigation would open, necessitating a very heavy movement from the collieries. Nut coal was by this time in very short supply in the West, and its price at Chicago advanced 50c. in about two weeks. This shortage was apparently due to its increased use in base-burning stoves. As it is practically impossible for the breakers at the collieries to alter their screens so as to change the relative percentages made of the various sizes, any increased demand for a particular size is pretty sure to make that size scarce and command a premium. This shortage of nut size finally was felt in the East. By the end of the month stove coal was quoted at $3·75 and nut at $3·65.

During March the Temple Iron Co. was formed and some extensive purchases of coal lands were made by the New York, Ontario & Western. The demand for coal in the East was fair, and in the West very good, a considerable

amount of coal going forward to Chicago by rail. The small steam sizes, which had been in excellent demand through the winter, were now quoted $2·40 and $2·60 for pea, $2·10 and $2·40 for buckwheat, and $1·40 and $1·65 for rice. April brought warmer weather, and a consequent fall in demand, particularly in the East. Stocks were allowed to accumulate preparatory to a heavy movement up the lakes when navigation opened. The sales agents of the various companies met and announced a new schedule of prices from May 1st, as follows: Broken $3·15, egg $3·35, stove $3·75, nut $3·75. It was openly stated that the selling price of nut coal in February at New York had been about $3·45, and in March $3·54. May brought the opening of lake navigation. All docks were practically cleaned up, there being less than 1,200 tons on hand at the head of the lakes, and perhaps less in proportion at Chicago. The movement West started strong, the current price of transportation from Buffalo to Duluth being 30c. per ton. In the East consumption fell off.

The situation in the East showed little change during June and July. A new New York price list was brought out July 1st, as follows: Broken $3·50, egg $3·75, stove and nut $4. This schedule, however, was not adhered to.

Freight rates up the lakes continued to get stronger and by the middle of June 40c. per ton to Duluth was freely paid. During this month the movement West continued heavy. The market was unsteady in the East, but the Western demand kept up well. Selling prices for the six months f. o. b. New York were approximately: Broken $3·13, egg $3·27, stove $3·59, nut $3·53. The July prices were not maintained in the East. The collieries were running on short time, working on an average perhaps four days a week. The demand from the West continued good though lake freights from Buffalo to Duluth rose to 45c. and later to 60c. per ton. In September things changed for the better. The movement West was hindered by high freights up the lakes, 75c. being asked from Buffalo to Duluth, but buying East proved excellent and the stocks on hand showed an actual decrease during August. A new price list showing a 25c. advance was issued October 1, 1899. During the last three months of the year everything favored the coal producer. Consumers at both Eastern and Western points cheerfully paid higher prices. By the end of December the market in the East was beginning to show a slight weakness, but the cold wave at the end of the month stirred up retail buying on a large scale and the year ended with even a more prosperous outlook than at the beginning. Prices have been firmly maintained at New York at the October figures. Owing to the great demand for coal by manufacturing concerns in the East, the sizes of anthracite generally used for steam production have been in much better demand than usual. This has kept broken size from being a drug on the market, and has made pea and buckwheat bring excellent prices. Average selling prices f. o. b. New York were, for the last half of the year: Broken $3·22, egg, $3·49, stove $3·92, nut $3·93. Production during the year was much more uniformly distributed than of late years. There have been few serious strikes; the most important was at the collieries of the Susquehanna Coal Co. at Glen Lyon and Nanticoke. There were strikes also at other points in the region, but none were of long duration.

The Bituminous Coal Trade of the United States in 1899.

The manufacturing interests supplied by the seaboard soft coal trade had a year of great prosperity. The prolonged drought which began in April greatly curtailed the water power of many mills and was felt especially by those concerns which in an ordinary season would rely almost wholly upon water power; this condition resulted in an output of at least 25% greater than 1898. Prices, however, were not favorable. Several railroads in business competition lowered freight rates from mine to tide water to such an extent that the prices for bituminous coal at the opening of the year were very nearly the lowest in the history of the trade. The trade conditions during January were fair; the demand for vessels was heavy and their freight rates were extremely high. The Atlantic Transportation Co., which had brought a number of vessels from the Great Lakes to use in carrying coal found these craft unsuited to its needs and went into the hands of a receiver. The better quality coal was quoted at $1·65 and $1·70 Chesapeake Bay ports. During the month rates from Philadelphia to Boston were $1·50 and $1·75, and to Sound ports $1·00 and $1·35. There were a large number of contracts for coal to be delivered at Central and South American ports which resulted from the prolonged strike in the Welsh collieries in 1898. By March the demand for coal increased and prices were quoted on a basis of $1·65 for best grades at Philadelphia and Chesapeake Bay ports. Freight rates were $1·15 to $1·25 from Philadelphia to the Sound ports during the month and $1·30 to $1·45 to Boston. More coal than usual went forward by rail. During April heavy shipments from New York harbor ports led to a break in ocean freight rates which lowered the rate from Philadelphia to Providence to 65c. and to Boston 75c. Trade during May and June was generally quiet. In July the demand for coal was strong. The car supply, however, was only sufficient for about two-thirds of the demand. Vessel freight rates remained high for summer months, being 70c. from Philadelphia to Providence and 80 to 90c. to Boston. By the middle of August the producers found themselves utterly unable to supply the demand for coal and the car shortage was intensified. Coal commanded a premium of 25c. a ton for immediate delivery. In October the pressure on producers became heavier. Vessel owners and captains became very firm regarding freight rates and, as a result, ocean freights went up to exorbitant figures, $3 from Philadelphia to Boston. Naturally the price of coal for prompt delivery rose also and one lot was sold for delivery at a Sound port for $4 a ton. The average price at Philadelphia was over $2 per ton. At many Eastern points bituminous coal actually sold for more than pea and broken sizes of anthracite.

Chicago.—The year began with an excellent trade. The demand for hard coal became so strong that there was an actual scarcity after July. Chestnut size was at a premium during the entire year. The features of 1899 were the advanced price on anthracite, the shortage of vessels and cars and the high freight rates. Prices of hard coal opened the year at $4·50 to $4·75 and advanced by jumps of 25c. per ton; it closed the year selling at $5·75 for egg and stove, while chestnut stood at $6. Coal receipts by the lakes, at Chicago, were very much lower

than in 1898 and the railroads also were unable to bring much coal here, the general demand for boats and cars having been so large that freight rates were at times almost prohibitive against coal shipments. In 1898 the cost of carrying coal by the lakes averaged 28c. per ton, while in 1899 the average cost had increased to 72c.

The activity in bituminous coal has been a feature the entire year, the demand from manufacturers being very great. Prices have jumped up 25c. to $1 per ton, ending the year high and very steady. Car shortage greatly interfered with business, and there was a reported shortage of 200,000 tons of soft coal as compared with last year's figures. Profits to the dealers have been large.

Pittsburg.—The year 1899 was a stirring one in the coal trade of the Pittsburg district. It opened with coal selling at lower prices than ever ruled in this market and closed with the stiffest rates charged for years. The river coal trade was not good during the year owing to the continued low water in the rivers. No coal was shipped South from May 20th to December 13th. The local trade, however, was better than in 1898. About 4,300,000 tons of river coal was sold in the local market during the year. What the operators lost in being unable to ship coal South was partially made up by the December shipment, which netted the combination a profit of over $500,000.

The river shipments, though less than the great total of 1896, exceeded those of any other year since 1891. They show that Pittsburg, in spite of competition, retains a strong hold on the river coal trade.

SHIPMENTS OF COAL FROM PITTSBURG BY THE OHIO RIVER. (IN SHORT TONS.)

Year.	Cincinnati.	Louisville and South.	Total.	Year.	Cincinnati.	Louisville and South.	Total.
	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.
1894.....	1,199,920	1,323,280	2,523,200	1897.....	1,144,568	922,800	2,067,368
1895.....	994,400	1,284,080	2,368,480	1898.....	1,236,800	1,319,120	2,555,920
1896.....	2,029,760	2,578,120	4,607,880	1899.....	1,000,000	1,185,500	2,810,500

The above table shows the annual shipments of coal from Pittsburg by the Ohio River. The largest shipments were in 1896, while the lightest were in 1897, when the low stage of water during almost the entire year caused months of waiting before the coal loaded in Pittsburg could be shipped by water.

Fuel and Its Economical Utilization.

By William Kent.

The question of the economical use of fuel is one of ever increasing importance. The recent rapid development of manufacturing industries and of electric lighting and transportation has been the chief cause of a vast increase in the consumption of coal, the amount of which is indicated by the statistics of coal production in the United States, the quantity produced, in long tons, being in 1880, 65,757,140 tons; in 1890, 139,351,438 tons, according to the figures of the United States census of these years; and in 1899 225,103,024 tons, according to The Mineral Industry.

A fair estimate of the average cost of coal to the consumer, including transportation charges, is $2·50 per long ton, which would make the total fuel bill of the United States, in 1899 approximately $562,757,560. A very large portion of this amount represents a kind of waste that may easily be prevented by means of well-known modern appliances; another portion is waste that is not preventable in the light of our present knowledge; a third is waste that might be saved by the use of appliances which are too expensive to be economically practicable; and a fourth portion is waste that may be saved under some circumstances and not under others. Examples of the first kind of waste, *i.e.*, that which is easily preventable are: (1) The use of furnaces for burning soft coal under steam boilers, which are not well adapted to that kind of coal; (2) the discharge of exhaust steam into the atmosphere when all or part of it might be utilized for heating purposes. An example of the second kind of waste—*i.e.*, that which is not preventable with our present knowledge—is the heat losses in the condensing water of condensing engines and in the jacket water of gas engines. An example of the third kind of waste—*i.e.*, that which may be saved by the use of expensive appliances—is that part of the heat lost in the chimney gases of steam boilers which might be saved by the use of an economizer. An example of the fourth kind of waste, which is preventable under some conditions and not under others, is that of exhaust steam from engines in a factory or other building, which may be utilized for heating purposes in cold weather but for which there is no use in warm weather.

In addition to these four general classes of waste, may be added a fifth class, a very important yet difficult one to deal with; namely, the waste that may be saved by the substitution of one kind of apparatus for another, or of one arrangement of plant for another. Usually each particular case under this head requires to be studied separately by a skilled engineer in order to determine whether the saving which may be accomplished will warrant the expenditure of money in extra plant required. For example, such questions as the following frequently arise: Shall we use a condensing or a non-condensing engine? Shall we use a compound or a triple expansion engine? Shall we use subdivided steam power, separate engines in each department of a plant, or shall we concentrate the power plant and transmit the power to the departments by electricity? Shall we use steam engines or gas engines? Shall each heating furnace have its separate coal fire, or shall we have a gas producer, and gas-fired

furnaces? Each particular case requires special investigation, and no general rules can be laid down for the treatment of all.

In order to confine this article within practicable limits, we shall avoid all discussions relating to the waste of steam after it passes from the boiler, and the waste of gas after it enters the pipe leading to the gas engine, and limit ourselves to the question of the economical use of fuel in the production of steam and in the production of gas for heating or power purposes.

Beginning with the steam boiler and its furnace, the first question to be considered is: How shall the fuel be burned? The answer to this is that it depends upon the nature of the fuel; and as coal is the fuel most universally used, we shall give it chief consideration.

In studying the qualities of different coals with regard to their use for generating steam, it is convenient to divide them into the classes given in the following table:

CLASSIFICATION OF COALS.

	Moisture. %	Ash. %	Volatile Matter. %	Fixed Carbon. %	Heating Value, B. T. U. per Lb. Combustible.	Relative Value of Combustible, Semi-bit.=100.
Anthracite and Semi-Anthracite	1 to 3	3 to 20	3 to 12	97 to 88	14,700 to 14,900	94
Semi-bituminous	1 to 3	3 to 10	15 to 25	75 to 85	15,600 to 16,000	100
Bituminous, Eastern	1 to 3	3 to 15	25 to 40	60 to 75	14,800 to 15,200	95
Bituminous, Western	4 to 14	5 to 25	35 to 50	65 to 50	13,600 to 14,800	90
Lignite	12 to 18	5 to 25	Over 50	Less than 50	11,000 to 13,000	76

The anthracite coals are mined in a small area of territory in eastern Pennsylvania, and their relative importance as steam coals is rapidly decreasing, on account of the competition of the cheaper semi-bituminous coals. The semi-anthracites are found in a limited portion of the western part of the anthracite field. The semi-bituminous coals are mined in a narrow strip of country, about 20 miles wide, on the eastern slope of the Appalachian range, over a length of about 250 miles, running from Clearfield County, Pa., S. W to the southern boundary of Virginia. They include the steam coals of Clearfield, Cambria, and Somerset counties, in Pa., the Cumberland, Md., the New River, W. Va., and the Pocahontas, Va. and W. Va., coals. It is a curious fact regarding these coals that their combustible portion is of remarkably constant quality throughout the whole range of territory in which they are mined, the percentage of volatile matter in the combustible scarcely ever varying beyond the limits of 18 and 22%. The moisture also is usually very low, but the sulphur and ash are apt to vary in different portions of the territory. Their heating value per pound of combustible also varies within very narrow limits, usually less than 1·5% on either side of an average value of 15,800 British thermal units per pound of combustible. They are clearly distinct from the semi-anthracite coals mined east of their territory and from the bituminous coals lying a short distance to the westward, so that it is rare to find a coal on the dividing line of quality on either side; that is it is rare in the United States to find a coal which has 14 to 16% of volatile matter or one which has 24 to 26%, while there is a great abundance of coal having from 18 to 22%, and a still greater quantity having from 30 to 40%.

Passing to the westward from the narrow area of the semi-bituminous coals, we have the bituminous coals which are mined all over the coal-bearing territory of the country west of the summit of the Appalachian range. These coals may be divided into three classes, which tend to overlap one another, there being no clearly marked dividing line between them. The first of these classes, called the eastern bituminous coals, comprises those of western Pennsylvania—the Pittsburg coal being the type—and those of the western part of West Virginia, eastern Kentucky, eastern Tennessee, and Alabama. They are distinguished from the coals mined further west by their low percentage of moisture, and usually also by their low percentage of ash and sulphur. They contain a much larger amount of volatile matter than the semi-bituminous coals, ranging from 30 to 40% of the combustible, but are of lower heating value, for the reason that their volatile matter is higher in oxygen.

The next sub-class of the bituminous coals, called the western, is distinguished from the preceding by containing a higher percentage of volatile matter, by having the volatile matter higher in oxygen, and therefore of less heating value, and by a higher percentage of moisture. The Illinois coals typify this class, which is exceedingly variable in the percentages of ash and sulphur, some samples having over 20% ash and 5% S.

There is no strict line of demarcation between the western bituminous coals and the lignites, which are still higher in volatile matter, oxygen and moisture, and consequently of lower heating value.

The above table shows that the heating value of the lignites per pound of combustible averages only 76% of that of the semi-bituminous coals. That is their theoretical value as determined by analysis or by a calorimetric test; but their practical relative value is very much lower, as it is not possible to burn them with an efficiency equal to that obtained with the semi-bituminous coals. Their content of moisture and ash is a greater detriment to their practical heating value than the figures indicate, as the moisture tends to make a lower temperature in the furnace, and the ash gives trouble in handling the fire, with the additional disadvantage that there is usually a loss of carbon in the ash and clinker removed from the fire.

A rough estimate of the relative practical values of the several classes of coal may be calculated as follows:

	Moisture. %	Ash. %	Combustible. %	B. T. U. per Lb. Combustible.	Theoretical Heating Value.	Efficiency of Boiler.	Relative Practical Value.	
							B. T. U.	Semi-bit. =100.
Anthracite	2	13	85	14,800	12,180	77	9,379	88
Semi-bituminous	2	8	90	15,800	14,220	75	10,665	100
Bituminous, Eastern	2	8	90	15,000	13,500	70	9,450	89
Bituminous, Western	10	15	75	14,200	10,150	65	6,598	62
Lignite	15	20	65	12,000	7,800	60	4,680	44

The figures in the column headed "combustible" are obtained by subtracting the sum of the percentages of moisture and ash from 100. These multiplied by the average heating value per pound of combustible give the theoretical heating value of the coal. The figures for efficiency of boiler are high as com-

pared with ordinary practice, but they may be realized with properly designed furnaces and with skillful firing. These multiplied by the theoretical heating values give the figures for relative practical value of average coals of the several classes, which are converted into percentages in the last column.

This table shows us that coal is a substance so varied in quality that a ton of one kind may be worth to the steam user only 44% as much as a ton of another; furthermore, the figures of the table really understate the truth, as a ton of lignite is worth to the steam user 44% as much as the ton of semi-bituminous coal only when his boiler furnace is well adapted to burn lignite. If he attempts to burn lignite in a furnace adapted for semi-bituminous or for anthracite coal only, he will have to burn nearly 2·3 tons of lignite to obtain the same result as from a ton of semi-bituminous coal, and in addition he may find that his boiler furnace is of too small a capacity to burn the quantity required. Since in common practice the furnace is limited by the boiler, as it is a part of it, he may have to buy an additional boiler as the alternative to paying a high transportation charge on a better quality of coal. It is common throughout the West to have larger boilers for generating a given amount of steam than those used in the East, since it is quite easy, with semi-bituminous coal to drive a boiler far beyond its rated capacity, while in the West, with poor coal, it is difficult to obtain even the rated capacity. This is not the fault of the boiler, considered as an apparatus for absorbing the heat produced in the furnace, but is the fault of the coal and of the furnace which is designed with insufficient coal-burning capacity.

Since coal varies so much in different sections of the country great care should be used in its purchase, but as a fact there is nothing that is bought so carelessly by the ordinary steam user. Coal is not sold on analysis nor on guarantee, as is iron ore, or steel, but on so-called "reputation." The writer recently had occasion to make inquiries concerning coal for shipment to Italy. The coal shipping agents furnished printed analyses which were evidently of selected samples obtained for advertising purposes. Upon being asked if a fairly sampled cargo of their coal would be guaranteed to show ash not over 3% in excess and volatile matter not over 5% in excess of the published analysis, they replied, "No, we never give any guarantees; our coal sells on its reputation." As long as a good mining company confines its operations to a few mines in a limited district, and exercises great care in the preparation of its coal for market, its customers may be reasonably sure of getting coal of about the same quality, but in these days of consolidation of companies, and of operations over a wide territory, together with the opening of new mines of no reputation, there is every chance that sometimes the customer will receive coal of inferior quality. The remedy lies with the customer himself. When he learns how to sample and test each cargo or trainload of coal he receives, and to keep records of the quantity of water evaporated in his boiler room as well as the quantity of coal used in it, he will soon find a way to buy coal on guarantees of quality similar to those obtained with other materials used in his factory.

Eight years ago the writer published in Vol. I. of THE MINERAL INDUSTRY a

study of Mahler's analyses and calorimetric tests of European coals, in which he drew the conclusion that the heating value of a coal per pound of combustible might be estimated within quite narrow limits from the proximate analysis. The relation between the heating value and the percentages of fixed carbon and of volatile matter in the combustible was shown by a table and also by a plotted curve. More recent study of Mahler's tests in comparison with the tests of American coals reported by Profs. Lord and Haas, of Columbus, O.,* and by Messrs. Hale and Williams† while confirming the writer's original conclusions, have led to a slight modification of the figures and the curve and to a restatement of the limits within which they are approximately accurate. The revised figures and the curve are reproduced herewith.

APPROXIMATE HEATING VALUE OF COALS.

Per Cent. of Fixed Carbon in Coal, Dry & Free from Ash.	Heating Value.		Per Cent. of Fixed Carbon in Coal, Dry & Free from Ash.	Heating Value.	
	Calories.	British Thermal Units.		Calories.	British Thermal Units.
97	8,300	14,940	68	8,400	15,120
94	8,450	15,210	60	8,200	14,760
90	8,600	15,480	57	7,900	14,220
87	8,700	15,660	55	7,700	13,860
80	8,800	15,840	53	7,400	13,320
73	8,700	15,660	51	6,900	12,420
68	8,600	15,480			

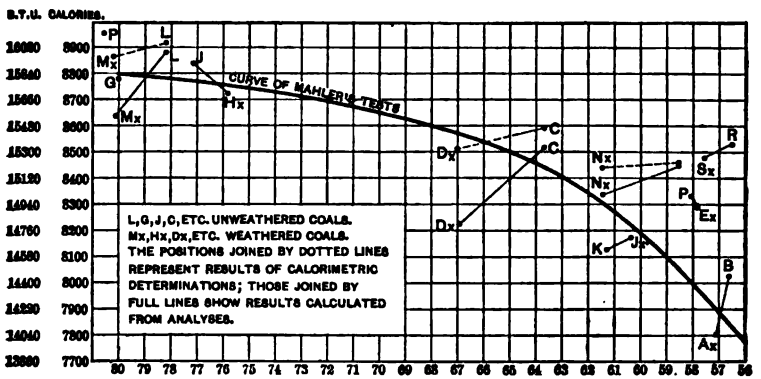

B.T.U. CALORIES.
CURVE OF MAHLER'S TESTS
L,G,J,C,ETC. UNWEATHERED COALS.
Mx,Nx,Dx,ETC. WEATHERED COALS.
THE POSITIONS JOINED BY DOTTED LINES REPRESENT RESULTS OF CALORIMETRIC DETERMINATIONS; THOSE JOINED BY FULL LINES SHOW RESULTS CALCULATED FROM ANALYSES.

FIG. 1. CALORIFIC POWER OF COALS, WEATHERED AND UNWEATHERED.

The original claim made for the curve in 1890 was as follows: Knowing the percentage of fixed carbon in the dry coal free from ash, we may in the case of all coals containing over 58% fixed carbon predict their heating value within a limit of error approximating 3%.

In 1897 a study of the work of Lord and Haas led to the following statement: Excluding the coals that have less than 58% fixed carbon in the combustible, the variation of any one of Lord and Haas's coals from the Mahler line does not exceed 320 calories (576 B.T.U.) or about 4%. Taking the average figure for each class of coals (there were seven classes, 40 coals in all, *viz.*: Pocahontas, 5;

* *Transactions of the American Institute of Mining Engineers*, Vol. XXVII., p. 259.

† *Transactions of the American Society of Mechanical Engineers*, Vol. XX., p. 337, 1899.

Hocking Valley, O., 5; Thacker, W. Va., 4; Pittsburg, Pa., 7; Middle Kittaning, Pa., 7; Upper Freeport, Pa., and O., 11; and Mahoning, O., 1), it falls in all cases within the limits of 3%. It is to be observed, however, that the Mahler line falls rapidly with percentages of fixed carbon below 62, and it is therefore to be expected that below this point there will be a greater range of variation in heating value than above it. When the volatile matter exceeds 38% an increasing proportion of it is oxygen, and the relative proportion of oxygen in the highly volatile coal varies in different districts, as is shown by the analyses of Lord and Haas. Thus the Upper Freeport coal, dry and free from ash, averages only 9·58% O, while the Hocking Valley coal averages 16·10%, although both have the same percentage (58%) of fixed carbon. Full credence, therefore, is to be given to the conclusions drawn from Lord and Haas's tests, "that when the fixed carbon is less than 62% of the combustible, each class of coal has a law of its own, and that coals of any one class may differ in heating power from the coals of another class containing the same percentage of fixed carbon, as much as 5%, as exemplified in the case of the Upper Freeport and the Hocking Valley coals.

The work of Lord and Haas also confirms the conclusion drawn from Mahler's tests, *i.e.*, that the relation of the heating value of coal to its ultimate analysis is expressed by Dulong's formula with remarkable accuracy, that is, within a limit of error of usually less than 2%, which percentage is approximately the limit of error of analysis and of calorimetric determination. This formula with recent average figures for the constant is: Heating value per pound in B.T.U. equals

$$\frac{1}{100}\left[14{,}650\ \mathrm{C}+62{,}000\left(\mathrm{H}-\frac{\mathrm{O}}{8}\right)+4{,}000\ \mathrm{S}\right];$$

in which C,H,O and S are respectively the percentages of carbon, hydrogen, oxygen and sulphur in the coal. Their work, as well as that of Mahler and of Williams, discredits the results obtained by Scheurer-Kestner and Meunier-Dollfus, in 1868, which indicated that the actual heating value of coal was much higher than that calculated by the Dulong formula, and also discredits the results obtained later by some American chemists, which indicated that the heating value was less than that calculated by this formula.

Another most important conclusion from the tests of Lord and Haas is that the coals mined from one seam over a considerable area of territory have a nearly uniform heating value per pound of combustible. This is true of the seven classes of coal above mentioned, but how much farther it extends is a question for future research. It seems to be true of the anthracite coals, of the semi-bituminous coals and of the bituminous coals mined in horizontal seams over wide areas in parts of western Pennsylvania, Ohio and Illinois, but more extended research in other territory may show exceptions. It is greatly to be desired that the work so well begun by Lord and Haas shall be extended to cover the coal areas of the entire country. If a map of these coal areas could be drawn on a moderately large scale, and on it be marked symbols representing the class and the average heating value of the combustible of the coal in each district, and of each seam in the district, it would add vastly

to the value of the present fragmentary and imperfect knowledge of American coals. In constructing such a map, the results of analyses and calorimetric tests made earlier than 1897 should be rejected on account of inaccuracies so frequent as to cast a doubt upon all. With the Mahler and other forms of the Berthelot bomb calorimeter in the hands of skilled chemists, and with modern methods of sampling, of moisture determination* and of analyses, there is no longer an excuse for inaccurate work.

How to Burn Coal.—The quality of American coals having now been discussed at sufficient length, we return to the question of how they should be burned under steam boilers in order to obtain the most economical result.

Two fundamental propositions should always be borne in mind when the question of steam boiler economy is considered.

(1) The coal should be burned in such a manner as to obtain the highest temperature possible.

(2) The boiler should be designed so as to absorb the greatest proportion possible of the heat generated, and to discharge the waste gases into the chimney at the lowest temperature possible.

An equation for the efficiency of a steam boiler is: Efficiency$=\frac{T_1-T_2}{T_1}$, in which T_1 is the temperature of the fire and T_2 the temperature of the chimney gases. It is evident from this equation that the efficiency is a maximum when T_1 is a maximum and T_2 a minimum. The first of these propositions relates to the boiler furnace; the second to the heating surface of the boiler.

The furnace and the boiler are two different pieces of apparatus, having different functions, and each should be considered separately. In designing a boiler capable of evaporating a given quantity of water into steam, (10,000 lb. per hour, or say 300 commercial horse-power, as stationary boilers are rated,) the chief questions to be considered are: (1) the extent of the heating surface, so that it may absorb as large a percentage as possible of the heat supplied to it; (2) the arrangement of this heating surface, so as to provide the proper area of gas passages, to insure the circulation of the water, to be structurally strong and durable, to be easily cleaned and repaired, etc.

In designing a furnace for the same boiler the chief considerations are: (1) that the grate surface shall be large enough for the burning of the greatest amount of coal that will ever be required to furnish the heat for the boiler; (2) that the furnace chamber, or the combustion chamber attached to the furnace, shall be of such size and form as to insure that the gases produced by the burning coal shall be thoroughly burned before they reach the relatively cool heating surfaces of the boiler.

The old practice of designing the furnace as an integral part of the boiler, with the grate placed immediately below (the common forms of shell and water-tube boilers) or in a flue or fire-box inside of the boiler structure (locomotive, marine and vertical tubular boilers) neglects entirely the two prime considerations of good furnace design mentioned above, except under the condition that the fuel is anthracite coal, natural draft being used with large sized coal and

* See Report of the Committee on Boiler Tests *Transactions of American Society of Mechanical Engineers*, Vol. XXI.

forced draft with small. Whenever the fuel is soft coal the requirements for good boiler and furnace designs are incompatible with one another if the designer is required to make the furnace with an ordinary grate an integral part of the boiler. In other words it is not possible to design a furnace properly for burning soft coal on an ordinary grate, if it has to be placed in the limited space within the flue of an internally fired boiler, or directly underneath the heating surface of an externally fired boiler.

With egg or larger sizes of anthracite coal, and with good chimney draft, it is easy to burn on an ordinary grate, of an area equal to one-fortieth of the area of the heating surface of the boiler, an amount of coal sufficient to drive it to more than its rated capacity. With coal of this character a large combustion chamber is not needed, as the coal contains so little volatile matter that it makes a very short flame, and consequently there are no unburned gases of combustion to be chilled and thus prevented from burning when they touch the heating surface above the grate. The ordinary design of a furnace as part of a boiler is therefore a good one for large sizes of anthracite.

With small sizes of anthracite, as pea, buckwheat and rice, the draft through the bed of coal on the grate is obstructed by the fine particles of coal and still more by the ash and clinker, which forms rapidly, and consequently either the grate has to be enlarged beyond the limits of the space provided for it in the design of the boiler, or else forced draft must be used to overcome the resistance to the passage of the air through the bed of coal and ashes. Even with forced draft, it is rare that the relative economy or efficiency of a boiler is as high with small sizes of anthracite as with large sizes. This may be explained by the fact that the small sizes are apt to contain a higher percentage of surface moisture. When fresh wet coal is thrown on a white-hot fire, the draft is temporarily obstructed by the extra layer of fine coal, causing a deficiency in the air supply, with the result that the moisture is decomposed at the point of contact between the hot coal and the fresh coal, and forms water-gas (CO+H) which escapes unburned, as there is not sufficient hot air to burn it.

When soft coal is used in the ordinary boiler furnace, having the grate near the heating surface, all of the proper conditions for a good design of furnace are violated and a great waste of fuel results from the unburned combustible gases distilled from the coal and the ever increasing smoke nuisance.

Smoke.—The following reasons are given for the production of smoke in an ordinary hand-fired steam boiler furnace in which the grate surface is directly below the shell or tubes of the boiler:

(1) When fresh coal is fired on the bed of very hot coke lying on the grate, the moisture and volatile matter are distilled rapidly, yielding great volumes of gases.

(2) Instead of these gases being brought at once into intimate contact and thoroughly mixed with the hot air required to burn them, the air supply is at first almost entirely shut off, both by the mechanical action of the fresh coal choking the air passages through the hot coke, and by the generation of great volumes of steam and gas from the fresh coal; these two results destroy the partial vacuum due to the chimney draft which is required to draw the air through the coal bed.

(3) The production of the steam from the moisture in the coal and the distillation of the gases is a cooling process, which chills the furnace just at the time when it should be the hottest.

(4) The dull, smoky flame made by the partial combustion of the gases, instead of being given time and space in which to complete its combustion, is instantly brought in contact with the comparatively cool surfaces of the boiler, causing the deposition of soot and the production of black smoke.

Time, space and air, even when all are present, are not the only requisites needed to burn the gases distilled from coal or other fuel. In addition there should be an intimate mixture of the gases and air at the point where ignition is taking place, that is at the place where the temperature is hot enough to cause combustion, or the rapid chemical union of the combustible gas with the oxygen of the air. To illustrate this point suppose a can of kerosene oil to be set on fire in the open air; it will not burn with a clear white flame, as in an argand burner, nor with a transparent blue flame, as in a blow-pipe, but will show a dark reddish flame, giving off dense clouds of smoke. In this case there is plenty of time, space and air available, all out doors, in fact, but these are not sufficient to produce complete combustion. However, if the oil be transferred to an oil stove or to a Rochester lamp with central draft, and be fed slowly upward through the wicks, the oil itself or the vapor distilled from it at the top of the wick will be brought in contact with air which has been heated to a very high temperature by radiation from the flame and perfect combustion will ensue. Should the wick be turned a trifle too high the lamp or stove will smoke, for although there is plenty of air present, it is not hot enough to cause combustion.

The following is the chemical reason for the production of smoke and soot when a transparent gas is burned with an imperfect admixture of air. As there is not enough oxygen at the point of ignition to combine with all the hydrogen and all the carbon in the gas at that point, one of these elements must be slighted, and as the affinity of hydrogen for oxygen is greater than the affinity of carbon for oxygen, the hydrogen is first satisfied and the remaining carbon, without the aid of oxygen to convert it into invisible gas, carbon dioxide, is set free in the form of a black smoke which, in reality, consists of fine particles of carbon or soot.

The erroneous statement is frequently made that smoke cannot be burned after it is once formed, and consequently the existence of an apparatus that will burn smoke or a "smoke consumer" is impossible. Smoke and soot can be burned in a similar manner to other carbonaceous substances if they are carried into an intensely hot chamber, or passed through a bed of white-hot coal or coke.

In order to burn coal without smoke under a steam boiler the four conditions above stated should be avoided or neutralized, thus:

(1) The gases should be distilled from the coal slowly.

(2) The gases when distilled should be brought into intimate contact or thorough admixture with very hot air.

(3) They should be burned in a hot fire-brick chamber.

(4) While burning, they should not be allowed to come in contact with comparatively cool surfaces, such as the shell or tubes of a steam boiler; this

means that the gases shall have sufficient space and time in which to burn before they are allowed to come in contact with the boiler surfaces.

The production of smoke and the concomitant loss of economy in the common style of steam boiler furnace increase proportionately with the amount of volatile matter in the coal. With semi-bituminous coals low in both volatile matter and moisture, the smoke and the loss may be appreciably lessened by carefully firing small quantities at a time, and by the introduction of heated air above the fire or at the bridge-wall. At best, however, these remedies are imperfect and with bituminous coals containing over 25% volatile matter they are failures. With these coals a partial diminution of the smoke may be effected by diluting it with air and steam, blown in over the fire, but this remedy is exceedingly wasteful. In order to suppress the smoke entirely and at the same time to increase the economy, the common style of furnace must be replaced by improved designs or by other and better means of burning the coal.

Various methods of burning coal without producing smoke are now in use, and are more or less successful. That these methods are not more commonly employed is due to several causes: (1) They generally entail increased first cost of apparatus, and some require a greater space than the common furnace. (2) The failure of many so-called smoke consumers has led to a popular belief in the impracticability of all of them. (3) The conservatism and prejudice on the part of steam users lead them to continue the use of old methods for years after they have been proven wasteful. Only such conservatism and prejudice can account for the present use in Pittsburg and along the Ohio and Mississippi Rivers, of the ancient two-flue boiler with its large grate surfaces burning double the amount of coal that should be burned under its limited heating surface. Common practice with this boiler yields a temperature of 800 to 1,000°F. in the chimney, and the money value of the waste thereby occasioned amounts in the aggregate to millions of dollars a year.

The two-flue boiler, however, is gradually being superseded by boilers of a modern type, such as the water-tube boiler, but this latter is still commonly set in the soft-coal districts with a furnace that is adapted for anthracite only. A long campaign of education is necessary before steam users will learn that improved furnaces are as essential to economy as improved boilers.

The one wrong way to burn soft coal is to burn it on an ordinary grate, set immediately beneath the boiler, the coal being fired by hand. There are many better ways—and some of these are better than others.

From a theoretical standpoint, the ideal way is first to convert the coal into gas in a producer, and then to burn this gas with very hot air in a fire-brick chamber. The Siemens gas producer and the Siemens regenerative furnace constitute a nearly perfect apparatus for smokeless combustion and they are economical when used for heating and melting steel and glass. For steam boilers, however, they are as yet impracticable as their initial cost is great and the subsequent cost of maintenance and attendance is high. Numerous attempts have been made to adapt the Siemens gas producer and regenerator to steam boilers, but without commercial success. The regenerator, of the ordinary form, is not well adapted for steam-boiler practice, as the waste gases from the boiler are at so

low a temperature that they cannot impart a sufficient amount of heat to the regenerator through which they pass on their way to the chimney and, as a consequence, the air subsequently passed through the warmed regenerator is not heated to the temperature required for the ignition of the producer gas. Modifications of the producer and regenerator will doubtless be devised that will accomplish the desired result; as for instance, the use of a blast-driven producer making a gas high in carbonic dioxide and therefore high in temperature, together with a regenerator that is heated by the combustion of a portion of this gas, but as yet no apparatus of this description has been introduced in practice. A broad field for experiment in this line is still unexplored.

Another method of burning coal which is excellent in theory is to grind it to dust and then inject it into the furnace with the air required for its combustion. This method was used nearly thirty years ago in the Crompton puddling furnace at the Woolwich Arsenal,* and numerous attempts have since been made to apply it to steam boilers. An experiment with it was made in New York about two years ago, but it does not appear to have been economically successful, although it prevented smoke. Its chief objection is the cost of grinding the coal and the high degree of skill required for the management of the furnace so as to obtain the right proportions of air and of coal.

A third method of burning soft coal without smoke is to reverse the ordinary operation of a gas producer, and burn the coal downward, fresh coal and the air both being supplied at the top of the bed of coal. The gas generated in this way travels downward through the bed of hot coke, which is being burned by a supplementary supply of air. No gas producer of this type is in use, as far as the writer is aware, although a similar device is extensively used in the United States, known as the Hawley down-draft furnace. This furnace has two grates one above the other; the upper one is formed of water-tubes, and the lower is an ordinary grate. Coal is fed on the upper grate and is burned by air, which is introduced above it; the gases pass downward through the bed of coal to the bed of coke on the lower grate formed by the pieces of coke that have dropped through the wide spaces of the upper grate. An additional supply of air passes upward through this bed of coke, becomes highly heated and burns the gas thoroughly when the furnace is properly managed. This furnace has been very successful in burning many of the poorer western coals. A fourth method is the ordinary furnace, in which the fresh coal is fed beneath the bed of over-lying coke. The gases as they distil from the fresh coal and pass through the coke are met by blasts of air, both being highly heated by the hot coke, and complete combustion takes place. This method also has been very successful in improving economy of coal and in preventing smoke. It has recently been extensively introduced by the American Stoker Co.

A fifth method, more or less successful under different circumstances, for diminishing smoke and increasing economy, is the use of mechanical stokers in which the coal is gradually carried from the front toward the rear of the furnace, either by being pushed over a series of rocking or step grates, or by traveling on chain grates. The theory of these stokers is that the gases are distilled slowly from the coal as it is very gradually fed in at the front of the furnace,

* *Journal of the Iron and Steel Institute*, I., 1873, p. 91.

and that at the rear they meet hot gases charged with an excess of air rising through the hot coke. When these stokers are placed directly under the ordinary form of water-tube boiler, with transverse gas passages across the tubes, they do not prevent smoke, for the reason that the vertical currents of smoky gas from the front portion of the grate, and of clear gas from the rear portion, both pass directly up into the passages between the tubes, where they are chilled, and never become mixed until they have cooled below the point of ignition. With horizontal shell boilers, the two currents are forced to flow horizontally for some distance, and there is therefore a better chance for them to mingle; even here the currents are very apt to stratify and refuse to mix before they enter the tubes of the boiler. Another difficulty in securing smokeless combustion and good economy with this type of stoker is the fact that it is difficult to regulate the rate of feed of the coal and the air supply through the coked coal in the rear of the grate; one or the other is apt to be in excess, and a deficiency of air causes imperfect combustion, while an excess of air causes a waste due to the highly heated surplus air which passes out of the chimney.

In order to make this type of stoker successful as a smoke consumer, it is necessary to inclose it entirely in a fire-brick chamber and to provide passages from it to the heating surfaces of the boiler of such form that the gases distilled from the coal and the hot air pressing through the coke are compelled to mix together before they touch the heating surfaces. That this plan, whose merit seems to be almost self-evident, has not generally been followed, is due to the survival of the old idea that the boiler furnace is part of the boiler, and that the only space allowable for it is the limited portion immediately beneath the front, or in case of internally fired boilers, the space inside of the fire-box or furnace-flue. When designers of steam boilers and boiler plants will allow a proper space for the furnace, either by raising the boiler 8 to 10 ft. above the floor, or by building a fire-brick oven 10 or 15 ft. long in front of the boiler, for use with automatic stokers of the type mentioned, and will so design the furnace that it will cause a thorough mixture of hot air with the distilled gases, these stokers will become not only labor-savers, but savers of fuel and smoke preventers as well.

The simple device of building a fire-brick oven in front of a boiler to contain the grate is of great advantage in economizing fuel, even with hand firing, and especially with water-tube boilers. In some experiments made a few years ago by the writer with Illinois coal, a 17% gain in economy, over the ordinary setting, was obtained with this device. Its function is simply to give the gases an opportunity to burn under the fire-brick roof, instead of being chilled immediately upon their distillation by the cool tubes of the boiler, as in the ordinary setting. With this oven furnace the smoke was of much shorter duration than with the old setting, but it could not be prevented for the first two or three minutes after firing, as the fresh coal chilled the fire, and choked the air supply, and great volumes of smoky gas were distilled immediately after firing.

The writer has recently designed and patented a furnace which possesses the advantages of the oven furnace and has a device for mixing the gases and the hot air. It is illustrated in Figs. 1 and 2, as applied to a water-tube boiler.*

* United States Patent No. 604,158, May 17, 1898. *Stevens Institute Indicator*, 1899.

Referring to the cuts, *C* is a fire-brick chamber or oven, which extends out in front of the boiler. In it the fuel is burned, either on the ordinary grate-bars or by means of a mechanical stoker. *D* is the ordinary bridge wall. *E E'* are two tall vertical walls built some distance in the rear of the bridge wall. *G* is a combustion chamber. *H H* are several piers of fire-brick, projecting into the chamber *G* from the rear wall *J*. *K* is the ordinary partition wall built

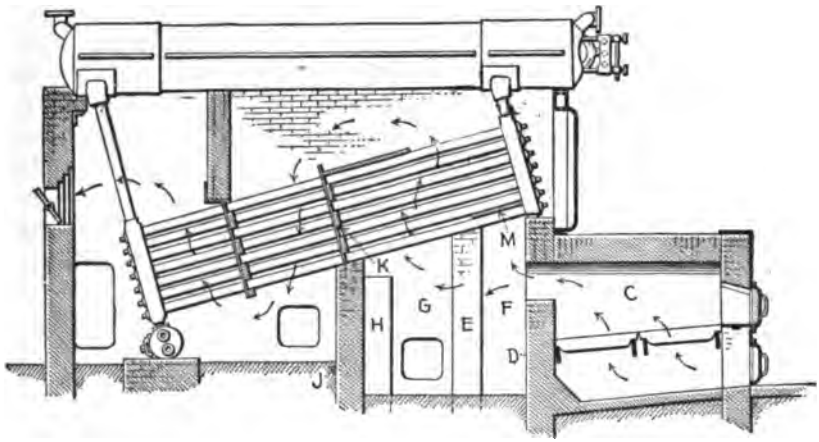


FIG. 1.—VERTICAL SECTION THROUGH FURNACE AS APPLIED TO A WATER-TUBE BOILER.

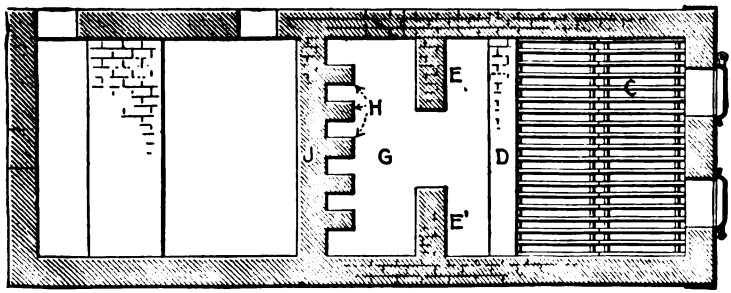


FIG. 2.—HORIZONTAL SECTION THROUGH FURNACE.
KENT'S PATENTED FURNACE.

across the tubes, and *M* is a tile roof to the chamber *F* to prevent the gases in that chamber from reaching the tubes until after they have passed through the narrow vertical passages between the wing-walls *E E'*.

In operation, the fresh coal is spread alternately on the right and left sides of the grate at equal intervals of time. Immediately after firing on one side, dense smoky gases arise on that side, while on the other side an excessive supply of very hot air is passing through the bed of partially burned coal or coke. These two currents, one of cool, smoky gas and the other of clear hot gas with a large excess of air, pass side by side over the bridge wall *D*, but they are compelled to change their direction and mingle together on passing through the tall

narrow passage between the wing-walls *E E'* and by so mingling, the gases are burned and smoke is prevented.

The combustion is assisted by the heat radiated from the walls of the combustion chamber *G* and the piers *H*, which absorb heat during the time when the fire is hottest—that is, just before fresh coal is spread on the grate—and give out heat to the gases in the chamber *G* when the fire is coolest—that is, just after firing, when the smoky gases are escaping.

Economy of the Heating Surface.—Having the boiler furnace designed of a size sufficient to burn the maximum quantity of fuel that will ever be required, and of such a form that it will cause, when the proper amount of air is supplied, the complete combustion of the gases distilled from the fuel, the next question to be considered is the design of the boiler itself.

The function of the boiler is to absorb as much as possible of the heat generated in the furnace, allowing as little as possible to escape in the chimney gases and by radiation. By proper covering of the boiler, the radiation loss can be reduced to as low as 2 or 3% of the heat generated. The loss of heat in the chimney gases depends on their quantity and their temperature. It is rarely below 20% with well-designed boilers and furnaces driven with the minimum air supply which will just maintain complete combustion, and may reach 40% or more, with boilers having insufficient heating surface and driven with excessive air supply, increasing the quantity of the chimney gases.

There is one rule for proportioning the extent of heating surface to secure maximum economy of fuel which should always be followed when economy of fuel is more important than first cost of boiler. This rule is: Provide one square foot of heating surface for every three pounds of water to be evaporated per hour, reduced to the "equivalent evaporation from and at 212°F." Taking a boiler horse power at 34·5 lb. of water per hour evaporated from and at 212°F. the rule is equal to 11·5 sq. ft. of heating surface per horse power. This area, however, will not suffice for maximum fuel economy unless the following three conditions are observed:

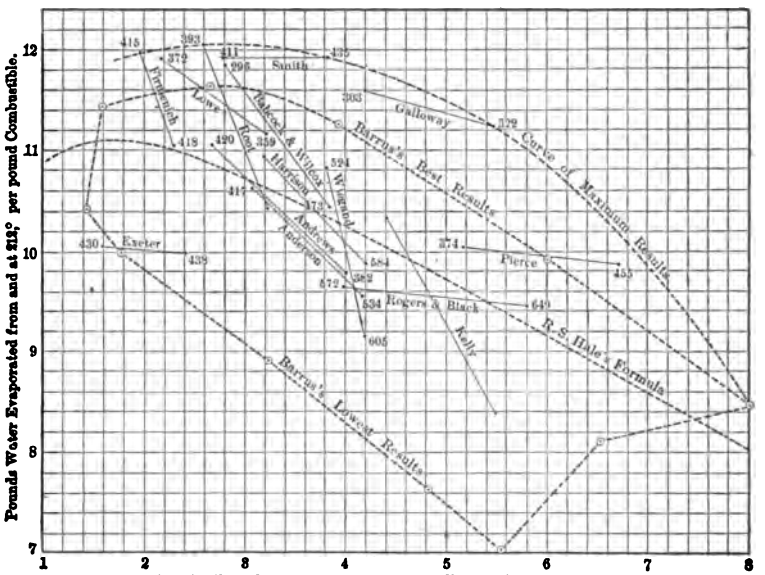
(1) The coal should be thoroughly burned in a proper furnace, with a minimum excess of air, insuring a high furnace temperature. (2) The gas passages through or over the heating surface should be so designed that nearly the entire heating surface is effective—this avoids the short-circuiting of the gases. (3) The heating surface should be kept clean both inside and out. When these conditions are obtained the boiler may be driven at a rate of 4 lb. of water per hour per sq. ft. of heating surface, corresponding to about 8·5 sq. ft. of heating surface per horse power, with very slight decrease of economy, otherwise the economy is apt to fall off rapidly.

The relation of the rate of evaporation to the efficiency of heating surface, as shown by tests with anthracite coal, is given in the accompanying diagram, which was originally contributed by the writer to a discussion of Mr. R. S. Hale's paper on "Efficiency of Boiler Heating Surface."*

The several inclined straight lines in this diagram indicate the results obtained from the economy and the capacity tests of the boilers tested at the Centennial

* *Transactions of the American Society of Mechanical Engineers*, Vol. XVIII., p. 854.

Exhibition in 1876. The rapid decrease of economy with increased rate of driving with some boilers, and the small decrease with others, is quite remarkable, and the causes of the difference have never been explained. The decrease of economy may have been due partly to imperfect furnace conditions under rapid driving, and partly to such arrangement of the gas passages that the gas was short-circuited, making a large portion of the heating surface ineffective. The great difference between the maximum and minimum results is no greater than that often found in boiler tests. There is evidently a wide field here for further experiment.



RELATION OF RATE OF EVAPORATION TO EFFICIENCY. ANTHRACITE COAL.
Centennial Tests, Barrus's Tests and R. S. Hale's Formula Compared.
The figures are the temperatures of uptake in the Centennial Tests.
Hale's Formula is $Ea = 13 - \frac{.6W}{S} - \frac{1.5}{\frac{W}{S}}$, in which Ea = evaporation per pound combustible, and $\frac{W}{S}$ = evaporation per square foot heating surface per hour.

The curve of maximum results of the Centennial tests shows that the highest economy was obtained between the limits of 1·9 and 3 lb. evaporation per sq. ft. of heating surface per hour, by five boilers of different construction, *viz.:* the Root, the Firmenich, the Babcock & Wilcox, the Smith and the Lowe boilers. These yielded an evaporation of between 11·8 and 12·1 lb. of water from and at 212°F. per pound of combustible—figures which have never been exceeded in reliable competitive tests with anthracite coal. The curve marked "Barrus's best results" is plotted from the highest figures recorded in Geo. H. Barrus's book on "Boiler Tests," for tests with horizontal tubular boilers. There is a variation of more than 20% between the highest and the lowest results for the same rate of evaporation.

These tests teach us that economy of fuel does not depend upon the type of boiler, as boilers of widely different forms give the highest results under favor-

able conditions, and lowest results when they are unfavorable. No improvement in economy is to be expected from inventions of new forms of boiler. The standard forms of water tube boilers with inclined tubes and transverse passages for the gases across the tubes, offer as good conditions for the absorption of heat and the avoidance of short-circuiting of the gases as are ever likely to be obtained by any form of boiler, and no greater economy is to be expected from these or any other form of boiler than was shown by them in 1876.

The proper selection of the form or type of boiler for any given purpose is therefore independent of the question of fuel economy or heating surface efficiency as maximum economy may be obtained by any type. Other considerations, as space available, safety, durability, facility for cleaning and repairs, cost of repairs, first cost, etc., must be considered. In the design of the furnace, however, economy of fuel is the chief consideration, and the shape, size and style of furnace will be governed by the kind of coal to be burned, as has already been discussed under the subject of furnaces.

Economizers.—The use of fuel economizers in connection with steam boilers is now becoming quite common in large plants for a three-fold reason, *viz.:* (1) The use of higher steam pressures and temperatures necessitates a higher temperature of the waste flue gases leaving the boiler; hence there is a greater amount of waste to be saved, and also a higher temperature for the economizer to work with, which increases its effectiveness. (2) The use of condensers with the steam engines furnishes colder water for the economizers than that furnished by feed water heaters supplied with exhaust steam from non-condensing engines. (3) The increased tendency to invest the necessary capital in expensive apparatus, such as an economizer, providing it will effect a saving in cost of operation.

A calculation of the possible saving to be accomplished by an economizer may be made as follows: Assume a boiler evaporation of 8 lb. of water per pound of coal, and a production of 20 lb. of chimney gas per pound of coal, or 2·5 lb. of gas per pound of water evaporated. If the temperature of the furnace is 2,150°F. (a theoretical figure, assuming that there is no direct radiation from the fire to the boiler) and the flue gases are 600°F., the heat wasted in the flue gases will be 600÷2,150=28%. If by an economizer the temperature of the gases can be reduced to 300°F., half of this waste, or 14% of the total heating value of the fuel will be saved. The efficiency of the boiler alone will be 72% and of the combined boiler and economizer 86%. The gain in economy is 14÷72=19·3%. The gain of heat per pound of gas is 300×specific heat 0·24=72 B.T.U., and per pound of water evaporated 72×2·5=180 B.T.U. Also, suppose the feed water is supplied to the economizer at 100°F. and the steam pressure is 150 lb. gauge, corresponding to a temperature of 358°F., and 1,213 B.T.U. per lb. above 0°F. The heat furnished to the water by the boiler and economizer will be 1,213—100= 1,113 B.T.U., of which 180 B.T.U. is supplied by the economizer and 933 by the boiler. The gain in economy is 180÷933=19·4%—a percentage quite possible in practice provided that there is sufficient heating surface in the economizer and that the feed-water temperature is 100°F., with the gas as hot as 600°F.

Forced Draft.—An economizer generally obstructs the available draft to a

considerable extent, and when one is used, forced draft is advisable. This system is also desirable for burning small sizes of anthracite coal, and in all large plants where the load is variable. In the latter case it allows the maximum load to be carried with a smaller number of boilers than would be required with natural draft, although usually at a slight sacrifice of fuel, which, however, is offset by the decrease in the first cost of boilers and by a saving of space.

The use of *induced draft,* produced by a large fan placed between the economizer and the chimney is becoming quite common. This arrangement has usually the advantage of greater convenience than the forced draft system.

Steam Superheating.—The next important improvement in steam engine practice, which may be largely introduced in the near future, is the use of superheated steam. It was used to some extent more than fifty years ago and its benefit was then apparent although many difficulties were met in its application. The greater gains in steam economy due to the use of higher steam pressures, more rapid running of engines, automatic cut-off and compounding, caused superheating to be neglected for many years, but it is again coming into use especially in Europe. Among the difficulties encountered in the use of superheated steam, the location of the superheater caused considerable trouble; if it were placed in the flue between the boiler and the chimney, the temperature of the gases would be too low to render it efficient; if a special furnace were used for the superheater it would be not only expensive and troublesome to manage, but the superheater itself would be easily burned out. These difficulties have been recently overcome by locating the superheater midway between the boiler furnace and the flue, where the temperature of the gases is about 1,000°F., a heat high enough to make the superheater efficient, and yet not burn it out.

The common form of water tube boiler with inclined tubes has a large triangular-shaped space between the steam and water drum and the tubes, which is well adapted for the location of the superheater. The Babcock & Wilcox Co. has built a superheater so located, which has given excellent results. A detailed discussion of the subject of superheating, by Prof. R. H. Thurston, will be found in *Transactions of the American Society of Mechanical Engineers,* Vol. XVII.

The economy gained by superheating is said to be from 15 to 20% with the most economical forms of engine, when the steam is superheated 100 to 150°F.

Gaseous Fuel.—For many purposes the best method of utilizing coal is to convert it into gas and to convey the gas into the furnace in which it is to be burned. The ideal fuel is natural gas, but this is obtainable only over a limited area of the earth's surface. On account of its high heating value it may be transported in pipes to greater distances than any other kind of gas. Next in value is illuminating gas, made either by distilling highly gaseous coals or by enriching water-gas with oil or other enrichers. The kinds of gas that may be obtained from coal vary greatly in their composition and heating value, according to the nature of the coal and the process of manufacture.

1. *Carbon Monoxide Gas (Carbonic Oxide) CO.*—This gas, which may be formed by burning perfectly dry charcoal with a deficient supply of oxygen, is

never obtained in a pure state in any industrial process. When charcoal or coke is burned with air—the fuel being in a deep bed and the air supply restricted—the carbon monoxide formed is diluted with several times its volume of nitrogen; as both the air and the fuel contain moisture; some hydrogen will also be found in the product, and as the carbon dioxide, CO_2, formed by the complete combustion of the carbon at the bottom of the bed or at the point of entrance of the air, is apt to be not entirely reduced to CO in its passage through the fuel, there will likely be some carbon dioxide in the final product.

One lb. of carbon burned to CO produces 2·33 lb. of CO. The 1·33 lb. of O required for the combustion, if supplied by air, carries with it $1{\cdot}33\times77\div23$ lb., or 4·39 lb. of nitrogen. The product then will consist of 2·33 CO and 4·39 N, equalling 6·72 lb. or 85 cu. ft. of mixed gas, measured at 32°F. and atmospheric pressure. The heating value of CO being 4,400 B.T.U. per lb., that of the mixed gas will be $4{,}400\div6{\cdot}72=1{,}525$ B.T.U. per lb., or 52 B.T.U. per cu. ft.

2. *Blast Furnace Gas, and Anthracite Producer Gas.*—These gases approximate the diluted carbon monoxide gas above described. They contain some carbon dioxide due to an excessive supply of oxygen in the furnace or producer, and also some hydrogen, due to the decomposition of moisture which is always present in both the fuel and the air. The composition of blast furnace gas varies considerably, especially in the relative proportion of CO and CO_2, according to the fuel consumption of the furnace, its rate of driving, and other variable conditions. In six analyses made in one day at one blast furnace, reported by Prof. D. S. Jacobus,* the CO_2 varied from 6·6 to 7·7%; the CO from 20·1 to 31·7% and the nitrogen, hydrogen and hydrocarbons (by difference) from 60·5 to 72·2%, by volume. The average was 7·08% CO; 0·10% O; 27·80% CO; and 65·02% N. etc. The heating value of the gas, calculated from the analyses was 1,175 B.T.U. per lb., and at the temperature of 584°F. above the temperature of the atmosphere, at which it entered the steam-boiler furnace it carried 140 B.T.U. additional available heat.

In burning this gas under the steam boiler the combustion was not entirely perfect, 1·8% CO being present in the chimney gases, although they contained also 3% O. The efficiency of the boiler was 61% of the total heat contained in the gas. Of the loss of 39% about 23% was due to heat wasted in the chimney gases and 13% to the incomplete combustion of the CO.

The quantity of gas discharged from a blast furnace is far greater than the amount which is needed for the fuel supply of the steam boilers, furnishing steam to the blowing engines, and for hot-blast stoves, and the difference represents a vast waste of fuel which might be utilized if there were a demand for it for other purposes. In modern iron and steel plants, some of this waste is saved by installing additional boilers to make steam for the rolling mills and Bessemer works. In Europe some progress has been made in using the gas in gas blowing-engines, and it is proposed also to build engines to use blast-furnace gas for the generation of electric currents that will be carried to distant points for use. A fine problem in engineering design is here offered. It is to construct gas engines to use blast-furnace gas which will furnish electricity cheaper—tak-

* *Transactions of the American Institute of Mining Engineers*, Vol. XVII., p. 58.

ing into account all the costs of fuel, interest on investment, labor and maintenance—than it can be furnished by the same quantity of gas burned under steam boilers and making steam to be used in steam engines. For descriptions of two blast-furnace blowing engines using waste gas in the engine cylinder, one at Seraing, in Belgium, and the other at Hoerde, in Germany, see *Engineering and Mining Journal* of Aug. 26, Oct. 7, 1899, and Feb. 10, 1900.

3. *Siemens Producer Gas.*—The old form of the Siemens producer was a simple rectangular fire-brick structure, containing an inclined or "step" grate, which carried a bed of soft coal, 3 or 4 ft. deep, the air being supplied by natural draft. The product was a mixed gas, part being the volatile matter distilled from the freshly fired coal, carrying with it tarry vapors, and part consisting of carbon monoxide and nitrogen, produced by burning the coke in the lower part of the producer. Some carbon dioxide was also present, due to excessive air supply or to imperfect reduction to CO of the CO_2 formed near the grate. Some hydrogen also was formed by the decomposition of the vapor of water in the air and of the moisture in the coal. The composition of Siemens producer gas varies considerably according to the amount of volatile matter in the coal and its quality, to the moisture present in the air and in the coal, to the depth of bed of fuel and to the rate of driving. The following analysis, given by W. H. Blauvelt in *Transactions of the American Institute of Mining Engineers,* Vol. VIII., p. 64, may be taken as a typical one: CO, 25·3%; H, 9·2%; CH_4, 3·1%; C_2H_4, 0·8%; CO_2, 3·4%; N, by difference, 58·2%.

This gas has a calculated heating value of 2,270 B.T.U. per lb. or 155 B.T.U. per cu. ft. The composition of the coal from which it was made was: Water, 1·20%; volatile matter, 36·22%; fixed carbon, 57·98%; sulphur, 0·70%; ash, 3·78%. In the processes of gasification of the coal and purification of the gas there was a loss of 35·2% of the heat energy of the fuel.

Mr. Blauvelt emphasizes the following points as highly important in soft-coal producer practice: (1) That a large percentage of the energy of the coal is lost when the gas is made in the ordinary low producer and cooled to the temperature of the air before being used. To prevent these sources of loss, the producer should be placed so as to lose as little as possible of the sensible heat of the gas and prevent condensation of the hydrocarbon vapors. A high fuel bed should be carried, thereby keeping the carbon dioxide low. (2) That a producer should be blown with as much steam mixed with the air as will maintain incandescence. This reduces the percentage of nitrogen and increases the hydrogen, thereby greatly enriching the gas. The temperature of the producer is kept down, diminishing the loss of heat by radiation through the walls, and in a large measure preventing clinkers.

4. *Water Gas.*—When steam is blown into a bed of white-hot coke it is decomposed into its constituent gases, hydrogen and oxygen, the chemical reaction being $C+H_2O=CO+2H$, and the resulting product a mixture of equal volumes of carbon monoxide and hydrogen, or by weight of 28 parts of CO to 2 parts of H. The heating value of pure water gas is calculated as follows, using Favre and Silbermann's figures for the heat value of the constituents:

28 lb. CO× 4,325=121,100
2 lb. H ×62,032=124,064

245,164÷30 =8,172 B. T. U. per lb.

1 cu. ft. CO=0·0781 lb.× 4,325=338
1 cu. ft. H =0·0056 lb.×62,032=347

685÷2=343 B. T. U. per cu. ft.

The gas being measured at 32° F. and at atmospheric pressure.

The production of water gas by the decomposition of steam is a cooling process. The bed of fuel is rapidly chilled toward the point at which the reaction would cease, and hence the process must be an intermittent one, the bed being "blown up" by a blast of air in order to produce a high temperature, and then, the air being shut off, the steam is admitted to form the water gas.

The following is a statement of the thermal quantities involved in the production and combustion of water gas, assuming, for the sake of simplifying the calculation, that the water and carbon enter the process at atmospheric temperature, and that the products of combustion are finally cooled to the same temperature. To make 30 lb. of water gas=28 lb. CO+2 lb. H,

Heat development, 12 lb. C to CO×· 4,451= 53,412 B. T. U.
Heat absorption, 2 lb. H (from water)×62,032=124,064 B. T. U.

Deficiency, 70,652 B. T. U.

This may be made up by burning 4·86 lb. C to CO_2, making a total of 16·82 lb. C required to produce the gas.

4·86 lb. C to CO_2×14,544=	70,652 B. T. U.
12·00 lb. C to CO_2×14,544=	174,528 B. T. U.
16·86 lb. C burned to CO_2 would give	245,180 B. T. U.

30 lb. water gas burned to CO_2 and H (water)×8,172=245,160 B. T. U.

Therefore the same total quantity of heat is generated, theoretically, by the complete combustion of 30 lb. of water gas, as would be generated by the complete combustion of the carbon used in making it.

Practically, however, the thermal value of water gas is always less than that of the coal used in making it, for three reasons: (1) Some portion of the heat developed in burning the C to CO_2 in the blowing up part of the process is lost in the escaping CO_2 gas, although with a regenerator attachment to the producer much of it may be saved. (2) In the blowing up process some of the carbon is burned to CO, which escapes, causing a considerable additional loss of heat. (3) In practical applications of water-gas in furnaces the waste chimney gases are not discharged at atmospheric temperature, and the water produced by the burning of the hydrogen is not cooled to the liquid state, but escapes in the form of superheated steam.

The following calculation of the waste due to this steam in the chimney gases was given by the writer in 1886, in a paper entitled "Is Water Gas an Economical Fuel?"*

* *Transactions of the American Society of Mechanical Engineers*, Vol. VIII., p. 226.

Suppose a pure water gas to be of the formula

$$CO+H, \text{ made from } C+H_2O$$

[— or, pounds, 12+16+2] 12+18,

the escaping chimney gases from its combustion would contain, in addition to nitrogen and excess of air:

$$CO_2+H_2O$$

or, pounds, 44+18

The 18 lb. of water being the same from which the water gas was made with 12 lb. carbon. The C, if completely burned to CO_2 would generate

$$12\times14{,}500=174{,}000 \text{ B.T.U.}$$

Suppose that the chimney gases are cooled, by regenerators or other means, down to 212°F., that the water escapes in the form of steam of that temperature—its total heat including latent heat of evaporation, being 1,178 B.T.U.—and that the water entered the system at 78°F. The heat carried away by this water escaping in the shape of steam, in excess of that which it introduced into the system, would be $1{,}100\times18=19{,}800$ heat units equal to about 11·4% of the whole amount of heat produced by burning the 12 lb. carbon from C to CO_2. This is the most favorable case, for in practice it is never possible to cool the chimney gases of heating furnaces to a temperature as low as 212°F. Such a loss is not met with in the Siemens system, nor in the ordinary system of direct burning of the fuel upon a grate, except in so far as moisture in the shape of cold water may be present in the fuel used.

The great merit of water gas compared with carbonic oxide or with Siemens gas, is its relatively high heating value, 343 B.T.U. per cu. ft., as against 155 B.T.U. for average Siemens gas and 52 B.T.U. for pure carbonic oxide diluted with the nitrogen contained in the air from which it was made. This makes it possible to carry it economically in pipes to far greater distances than these other gases. Twenty years ago many hopes were raised by promoters of water-gas processes that this gas would be "the fuel of the future," that it would be generated in large plants in every city and distributed in pipes to consumers who would use it in their houses for heating purposes. Many plants were built with this end in view, but they have all proved commercial failures.

For large heating furnaces, as those used in iron and steel works, it has not proved as economical as Siemens gas; however, it has a field in small furnaces in manufacturing plants, from its convenience of distribution and handling, and from the high furnace temperature which it gives without the use of the regenerators required with Siemens gas.

Recently two new fields have been opened for water gas or its modifications, namely, gas engines and the Welsbach system of incandescent gas burners for illumination.

5. *Enriched Water Gas.*—Water gas has proven a great success as an illuminating gas, when enriched by hydrocarbon gases obtained from petroleum or from gas coal, by the Lowe-Strong gas process, which is now generally used in modern

gas works. The addition of the hydrocarbons not only gives the gas illuminating power but also doubles its heating value.

It is a question of the near future for engineers and capitalists to settle whether it will pay better to have a single system of gas works and distributing pipes in cities, furnishing the more expensive illuminating gas of high heating power to ordinary gas burners, to Welsbach burners and gas engines alike, or to have two systems, one furnishing the illuminating gas, and the other the cheaper and poorer gas, requiring much larger pipes for its conveyance.

Promoters are again at work trying to introduce cheap fuel gas, of low quality, into cities, but they seem to meet with little or no favor among gas experts. A recent scheme of this kind has been made to introduce an alleged "electrical" gas process* in Passaic, N. J., where a company has been organized with $100,000 capital to furnish gas at 50c. per 1,000 cu. ft. The "process" is apparently based on the old "Hall" process that has failed so notoriously in many localities where it has been installed during the past ten years, notably at San Francisco in 1899. One of its claims is that the gas contains a variety of nitrogen which is *combustible and has a heating value!* All stated analyses of the gas made by this process show it to contain over 50% N.

6. *Gases Produced by the Distillation of Coal.*—There are two principal varieties of these gases: (1) Retort gas, commonly called "coal gas," made in the old-style illuminating gas process by distillation in closed retorts, heated by coke burning beneath them, and (2) the waste gas from coke ovens.

A typical analysis of retort gas, or coal gas is given by Dr. Gideon E. Moore, in a report on Granger water gas, in 1885, in comparison with two analyses of water gas, as follows:

Components.	Composition by Volume.		
	Water Gas.		Coal Gas.
	Wo-cester. %	Lake. %	Heidelberg. %
Nitrogen	2·64	3·85	2·15
Carbonic acid	0·14	0·30	3·01
Oxygen	0·06	0·01	0·65
Ethylene	11·29	12·80	2·55
Propylene	0·00	0·00	1·21
Benzole vapor	1·53	2·63	1·33
Carbonic oxide	28·26	23·58	8·88
Marsh gas	18·88	20·95	34·02
Hydrogen	37·20	35·88	46·20
	100·00	100·00	100·00
Density	0·5825	0·6057	0·4580
B. T. U. per cu. ft. (at 62° F.)	650	689	642
B. T. U. per lb	14,650	14,940	18,390

The following analyses of gas from coke ovens is given by M. Ebelmen.† The coking process lasted 24 hours and the gas was collected at three different periods, as below:

* *Engineering News*, Dec. 7, 1899 and March 29, 1900.

† Groves and Thorpe's *Chemical Technology*, Vol. I., Fuels.

Components.	After 2 Hours. %	After 7½ Hours. %	After 14 Hours. %	Mean. %
Carbondioxide.....	10·13	9·60	13·06	10·93
Carburetted hydrogen.	1·44	1·66	0·40	1·17
Hydrogen	6·28	3·67	1·10	3·68
Carbonmonoxide	4·17	3·91	2·19	3·42
Nitrogen.....	77·98	81·16	83·25	80·80
	100·00	100·00	100·00	100·00

The coal from which this gas was made contained 78% fixed carbon, 2% ash and 20% volatile matter. It was therefore a semi-bituminous coal, and the volatile matter in it would be chiefly carburetted hydrogen, or marsh gas. It is evident from the analysis of the gas that only a small portion of it is the product of the distillation of the volatile matter, and that the greater part is the product of combustion of carbon by the air admitted to the oven, such combustion being necessary, in the beehive coking oven, to furnish the heat for the coking process. The calculated density of the gas of the mean analysis is 0·994, and its heating value is only about 35 B.T.U. per cu. ft., or 470 B.T.U. per lb. Only about 8% of its volume is combustible gas, the remaining 92% being incombustible carbonic acid and nitrogen. On account of its low heating value it is not practicable to transport the gas from beehive ovens to any distance for utilization, although it is sometimes, but very rarely, used as fuel for steam boilers located in close proximity to the ovens. In ordinary practice with beehive coke ovens not only is the whole volatile portion of the coal wasted, but the coking process involves a waste of some of the fixed carbon of the coal to keep up the heat of the oven. Both of these wastes, amounting in value to millions of dollars annually, might be saved by the use of the modern retort coke oven.

Mr. J. D. Pennock,* taking the statistics of coke production in beehive ovens in the United States in 1897, states that 22,140,000 tons of coal were used in these ovens, producing 13,289,000 tons of coke; that if retort ovens had been used with apparatus for saving the by-products, an addition of more than 2,600,000 tons of coke would have been made from the saving of coal, amounting to $5,000,000, besides the recovery of tar, ammonia, cyanides, benzene and gas, of a value of nearly $35,000,000. The by-product retort coke ovens are now being rapidly introduced in this country.

The amount of coal used in by-product coke ovens in the United States during the year 1898 was about 400,000 tons. The ovens numbered 280 Semet-Solvay, 180 Otto-Hoffmann and 90 Newton-Chamber ovens; total 550.

7. *Gas from Retort Coke Ovens.*—In the operation of retort coke ovens, such as the Otto-Hoffmann and the Semet-Solvay type, the heat required for the coking process is furnished by a portion of the gas distilled from the coal, the more valuable portion of the gas being utilized for the production of various valuable by-products, for illuminating purposes, for running gas engines, and for other uses as fuel. An excellent article by H. O. Hofman, on "The Production of Illuminating Gas and Coke in By-Product Coke Ovens,"† gives a very complete technical description of the process. A series of tests of Cape Breton coal in

* *Journal of the American Chemical Society*, August, 1899.
† *Engineering and Mining Journal* of Oct. 8 and 15, 1898.

an Otto-Hoffman oven gave the following results per ton of 2,240 lb. of coal used. The coal contained 9·9% of moisture. The average analysis of the dry coal was: C, 75·10%; H, 3·75%; N, 1·51%; O+S, 13·80%; ash, 5·84%; total, 100. A proximate analysis gave: Volatile hydrocarbon, 34·60%; fixed carbon, 59·56%; ash, 5·84%; total, 100.

Products from One Long Ton of Coal.	Pounds.	Per Cent.
Coke, large, 66·69%; small, 1·64%; breeze, 2·80%	1593·4	71·13
Tar	75·7	3·38
Ammonia=1·87% sulphate	7·6	0·34
Gas, total 10,390 cu. ft. of 0.466 sp. gr	368·0	16·43
Sulphur compounds in gas:		
Hydrogen sulphide, H_2S, 0·98 lb. per 1,000 cu. ft.	10·8	0·48
Carbon disulphide, CS_2, 0·13 lb. per 1,000 cu. ft.	1·6	0·07
Gas liquor and loss, by difference	182·9	8·17
	2240·0	100·00

Of the 10,390 cu. ft. of gas, 49·5% was surplus gas, that is, gas not required for the heating of the oven. This had the following composition:

Olefines, C_nH_{2n}, 5·2%; marsh gas, CH_4, 38·7%; hydrogen, H, 38·4%; carbon monoxide, CO, 6·1%; carbon dioxide, CO_2, 3·6%; oxygen, O, 0·3%; nitrogen, N, 7·7%; total, 100. Its calorific power (H burned to liquid water) was 6·86 B.T.U. per cu. ft.; its candle power, 14·7, and its sp. gr. (air=1) 0·512.

The coking process lasted nearly 34 hours. Chemical analyses of the gas were made at intervals, showing changes in the composition of the gas, and diagrams are plotted, from which the following figures are taken: CH_4 during the first 21 hours slowly decreased from 42 to 35%, and during the last 13 hours rapidly decreased to 9%; H during the first 21 hours slowly increased from 33 to 45%; during the next 9 hours it increased rapidly to 66%, and then for 4 hours remained constant; N ranged between 7 and 10% for 29 hours, and then increased during 4 hours to 16%; CO remained nearly constant at from 6 to 7% during the whole operation; CO_2 decreased slowly from 4 to 1%. The olefines, or illuminants, decreased regularly from 6 to 3% during 21 hours, then more rapidly for 7 hours, to less than 1%, and decreased still further for the last 6 hours. The heating value increased during the first 3 hours from 690 to 770 B.T.U. per cu. ft., then decreased for 18 hours to 630 B.T.U., then more rapidly and steadily decreased for 12 hours to 340 B.T.U. The illuminating value decreased from 18 to 10 candle power during the first 24 hours, and then in 5 hours decreased to 4 candle power, and remained below 4 candle power for the remaining 4 hours.

The gas produced during the first 14 hours, 46 minutes was surplus gas, and that during the remaining 19 hours, 10 minutes was oven-heating gas, required to heat the ovens. The quantity and quality of these two gases were as follows:

	Surplus Gas.	Oven-heating Gas.	Average of Entire Gas.
Average calorific value, per cu. ft., B. T. U.	686·0	567·0	626·0
Average illuminating value, C. P.	14·7	9·0	11·6
Average specific gravity	0·512	0·421	0·466
Volume per long ton, cu. ft.	5143·0	5247·0	.....
Average per long ton, per cent	49·5	50·5	.....
Calorific value of gas per long ton of coal, B. T. U.	3,527,920	2,973,680	.....
Calorific value of gas per long ton of coal, per cent.	54·2	45·8	.....

A "heat balance" of the process is given by Mr. Hofman as follows, comparing it with the production of illuminating gas from German coal in retorts:

The Heat contained in 100 lb. Dry Coal is Distributed as follows:	Per Cent. of Total Heat Value of Coal.	
	German Coal in Gas Retorts.	Cape Breton Coal in Otto-Hoffmann Oven
In coke, salable	46·4	72·3
In coke, used for heating retorts	10·1	.....
In tar	5·5	4·1
In gas, salable	21·0	12·7
In gas, for heating ovens	.....	10·7
In ammonia liquor, sulphur in purifiers, and loss	17·0	0·2
Total	100·0	100·0
Heat used and lost in distillation process	27·1	10·9
Heat contained in products	72·9	89·1
Total	100·0	100·0

The figures given by Mr. Hofman show that the surplus gas of the retort oven is of very high heating value, constituting it an excellent fuel gas, of about two-thirds of the value per cu. ft. of natural gas, and double the value of pure unenriched water gas. To make it a good illuminating gas it requires to be somewhat enriched with illuminants. It should be an excellent gas for use in gas engines.

An important practical consideration in connection with the economical manufacture of retort oven gas is given by Mr. Hofman as follows:

"The amount of gas consumed by a city at different periods varies greatly. In making gas in small retorts a larger or smaller number can be put in operation, or the fires heating them can be urged in a greater or less degree. A by-product coke oven, however, has to run uniformly all the year round, which means producing the same amount of gas. If heated exclusively with its own gas it would not be well adapted for furnishing greatly varying amounts of illuminating gas. When the consumption is small the excess of the surplus gas would have to be disposed of at any price it would bring, or go to waste. In order to make the system elastic, the ovens will have to be heated by some other fuel than coke-oven gas during the period of greatest consumption of illuminating gas, and this fuel will be found in producer gas. In times when the consumption of illuminating gas is the greatest the ovens will be heated exclusively with producer gas, and the whole coke-oven gas converted into illuminating gas; in times of smallest consumption the ovens will be heated exclusively with coke-oven gas, illuminating gas being produced the first 14 hours of the coking period and heating gas the rest of the time. . . . As to the matter of cost, it is cheaper to heat with producer gas than with that from the coke oven. . . . The candle power of the coke-oven gas will be reduced from 14·7 to 11·8 if all of it is collected in one holder. The additional cost of bringing it up again to the standard would more than be covered by the saving from the use of producer gas."

A large plant of Otto-Hoffmann ovens is now being built at Everett, near Boston, Mass., for the purpose of supplying illuminating gas, heating gas and coke.

The first section of this plant was started in March, 1889.* When completed it will consist of 400 retort ovens, with a daily capacity of about 2,000 gross tons of coal. It will be equipped with mechanical appliances for the cheap handling of coal and coke and a complete system of condensers, purifiers and ammonia stills.

The saving in coke by the retort oven, as compared with the beehive oven is estimated as follows in a pamphlet issued by the Semet-Solvay Co.

Kind of Coal.	Yield, Bee-Hive %	Yield, Retort Oven. %	Gain. %
Connellsville	66	76	10
Pocahontas	62	88	33
Alabama	60	74	23

The output of a retort oven is claimed to be more than double that of a beehive oven; the quality is better, and the proceeds from the by-products are, under favorable conditions, sufficient to pay for the cost of the operation and interest on the investment. From a ton (2,000 lb.) of West Virginia coal, which would yield in a beehive oven 60%, or 1,200 lb. of coke, the yield from a retort oven is stated to be 80%, or 1,600 lb. of coke, besides 50 lb. of gas tar, 20 lb. of ammonium sulphate and 3,000 cu. ft. of illuminating gas.

The retort coke oven has hitherto had a much more rapid development in Europe than in this country, but it appears now to have gained a secure foothold here, and its rapid introduction within the next few years is to be expected. Its extensive use promises not only to effect a vast saving in coal but also to have a beneficial effect upon other industries. It offers what is probably the cheapest method of producing illuminating gas. By the distillation of tar, one of its by-products, pitch may be made, which may be utilized in the manufacture of fuel briquettes from anthracite culm. The ammonia, another of the by-products, converted into sulphate, will when produced in great quantities, prove one of the cheapest of fertilizers, returning to the soil the nitrogen which is continually being taken from it by the crops.

8. *Producer Gas Made with a Steam Blast.*—It has been stated above that the product of the Siemens producer, when driven by natural draft, is a mixed gas, consisting of the volatile matter distilled from the coal, and of carbonic oxide and nitrogen, produced by burning the coke left after the volatile matter is driven off. By blowing the producer with a mixture of air and steam, either by means of a steam-driven air jet, or by a fan blower supplying air, with a steam jet added, the character of the gas is changed, by diminishing the percentage of nitrogen, by increasing the proportion of carbonic oxide, and by the production of hydrogen by decomposition of steam, as in the manufacture of water gas. It was shown in discussing water gas that its production, by blowing steam into white-hot coke, is a cooling process, and that therefore it has to be made intermittently, the producer being first "blown up" with air, to heat the bed of fuel, and then blown with steam, making water gas, and cooling the producer. By adding steam to the air blast in just the right proportion, the cooling effect of

* *Engineering and Mining Journal*, March 25, 1899.

the steam may be balanced by the heating effect of the combustion with air, and the producer may thus be run continuously. The product is then a mixture of water gas and Siemens gas. The proportion of volatile matter it contains will depend chiefly upon the kind of coal that is used, and the proportion of other constituents will depend on the quantity of moisture in the coal, on the relative quantities of air and steam in the blast, and on the thickness of bed and the rate of driving. A thin bed and rapid driving will tend to produce carbonic acid.

A theoretical computation of the analysis and heating value of steam-blown producer gas may be made as follows, on the assumption that the fuel used is dry coke, the volatile matter in the coal having been driven off, and that the operation, beginning with a hot bed of coke is continued without gain or loss in temperature, radiation being neglected.

Taking the calculation already given under the heading of Water Gas:

To make 30 lb. of water gas=28 lb. CO+2 lb. H.

Heat development,	12 lb. C to CO × 4,451 =	53,412 B. T. U.
Heat absorption, 2 lb. H (from 18 lb. H_2O) × 62,032 =		124,064 B. T. U.
	Deficiency	70,652 B. T. U.

This deficiency may be made up by burning 15·873 lb. additional C to CO, 15·873×4,451=70,651. The burning of this C requires 15·873×1·33=21·164 lb. O, and the air to supply this O carries in 21,164+$\frac{77}{23}$=70.853 N. The gas produced will then consist of

	Lb.	%
CO {28·0 / 37·037}	65·037	47·17
H	2·0	1·45
N	70·853	51·38
	137·890	100·00

The heating value of this gas will be:

65·037 lb. CO burned to CO_2 ×	4,325 =	281,285 B. T. U.
2·0 lb. H burned to H_2O ×	62,032 =	124,064 B. T. U.
		405,349 B. T. U.

The heating value of the carbon used in making the gas is 27·873 lb. C ×14,544=405,385 B.T.U., or practically the same as that of the gas itself, no losses of heat being assumed in the process of gas making.

The density of the gas will be 0·964, and the heating value 2,940 B.T.U. per lb., or 229 B.T.U. per cu. ft., measured at 32°F. The gas is much superior to either mixed carbonic oxide and nitrogen gas, formed by burning carbon with air, and to producer gas of the analysis given by Mr. Blauvelt, quoted above, but is inferior to water gas. The above calculation is a theoretical one, based on carbon as fuel. The actual gas made by this process will contain volatile hydrocarbons distilled from the coal, which will tend to give it a higher heating value, and it will contain some carbonic acid, which will tend to decrease its

heating value. The gas-making process also cannot be conducted without loss of the heat required to volatilize the hydrocarbons and loss by radiation.

Producer gas made from anthracite or from coke, with the addition of steam to the blast, is extensively used, especially in Europe, for gas engines. The Wilson and the Dowson producers, in England, the Lencauchez producer, in France, and the Taylor producer, in this country, are successfully used for this purpose. An illustrated description of a large gas producer plant, with two 7-ft. Taylor revolving bottom producers, recently installed by R. D. Wood & Co., of Philadelphia, at the shops of the Erie Railroad, in Jersey City, will be found in *Engineering and Mining Journal* of November 4, 1899. The fuel used was anthracite buckwheat or rice coal. The manufacturer's guarantee was to deliver in the form of gas an average of not less than 10,000 B.T.U. for each pound of buckwheat coal consumed. This would correspond to an efficiency of 80%, and to produce 80 cu. ft. of gas of a heating value of 125 B.T.U. per cu. ft. per pound of coal used. On a test with fine rice coal, gas of 141 B.T.U. per cu. ft. was made, and the Otto engines run by it gave an indicated horse power from between 11,000 and 12,000 B.T.U. per hour, corresponding to an engine efficiency of about 22%, and a consumption of about 85 cu. ft. of the gas per i. h. p. per hour, which is equivalent to about 1·1 lb. of coal.

From the producers the gas is taken through wash boxes, scrubbers and purifiers, to extract from it so much of the ammonia, tar and sulphur as to leave it clean enough for use in the engine, and it is then taken into a holder, which serves the purpose of balancing irregularities in production, consumption and constitution of the gas.

One of the limitations to the general use of gas engines supplied with producer gas has hitherto been the fact that all the producers designed for making gas for gas engines have been restricted to the use of comparatively expensive anthracite coal or coke, since there was no producer in the market that made a satisfactory gas from bituminous coal. Recently, however, a successful producer using soft coal slack has been erected at the chemical works of Brunner, Mond & Co., at Norwich, England.

9. *Mond Producer Gas.*—A description of this producer is given by Mr. H. A. Humphrey.* It is the result of over ten years of experiment by Dr. Mond. The producer, with its accessories, is shown in Figs. 3, 4 and 5. The freshly fired slack coal is held for a time in the bell-shaped casting hung from the top of the producer, and surrounded by the hot gas. The volatile products distilled from it have to pass downward through the hot coke, and the tar is thus destroyed and converted into a fixed gas. The sloping grate bars do not reach to the center of the producer, and part of the weight of the fuel rests upon the ashes which form a cone filling the central space down into the water, and it is from below the water that the ashes are removed. The air supply, mixed with steam, passes downward between the two shells of the producer on its way to the fuel, so that it is heated while the inner shell is cooled. The gas leaving the producer enters a regenerator, formed of tubes, where it is cooled, giving up some of its heat to the entering air and steam. It is next carried into a washer, where more of its

* *The Proceedings of the Institution of Civil Engineers*, Vol. CXXIX., 1897.

heat is taken in forming steam or water vapor, at a temperature about 190°F. It then passes to an acid tower, where it gives up its sulphuric acid to a flowing stream of acid-liquor, which is circulated by a pump. Finally it is taken to a large tank, filled with wood-packing, to afford a large surface, where it is further cooled by cold water; the clean and cool gas is then taken away for use in furnaces or in gas engines. At the Norwich works, in 1896, more than 24,000,000 cu. ft. of this gas was used daily.

The result of a test showed that for every ton of fuel about 2·5 tons of steam and 3 tons of air are blown through the grate, the mixture being at a temperature of about 480°F. Of this steam 1 ton is furnished by the system of regeneration, and 1·5 ton is added as exhaust steam from various engines and pumps. More than 0·5 ton of steam is decomposed in passing through the hot fuel, and nearly 4·5 tons of gas are formed from a ton of coal, equal to about 160,000 cu. ft. at ordinary atmospheric temperature. The gas has a calorific value of 81%, calculated on the calorific value of the original fuel, and in a Babcock-Wilcox boiler gives an evaporation of somewhat more than 7 tons of water. When used in a gas engine, it is sufficient to give 2,166 i. h. p. hours, being at the rate of 1·03 lb. of slack per i. h. p. per hour, the thermal efficiency of the engine being 23·8%; but better results have been obtained by using the Crossley scavenging system in the gas engine.

The result of the working of the Mond producer with gas engines is considered

FIG. 3

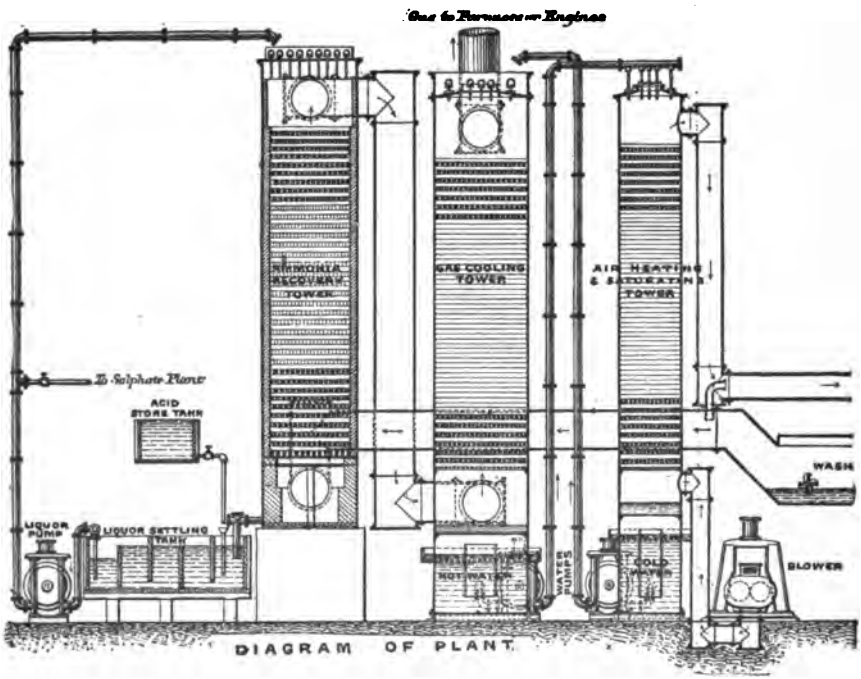


FIG. 3.—VERTICAL SECTION OF TOWERS, SETTLING TANK, ETC.
THE MOND GAS PRODUCING PLANT.

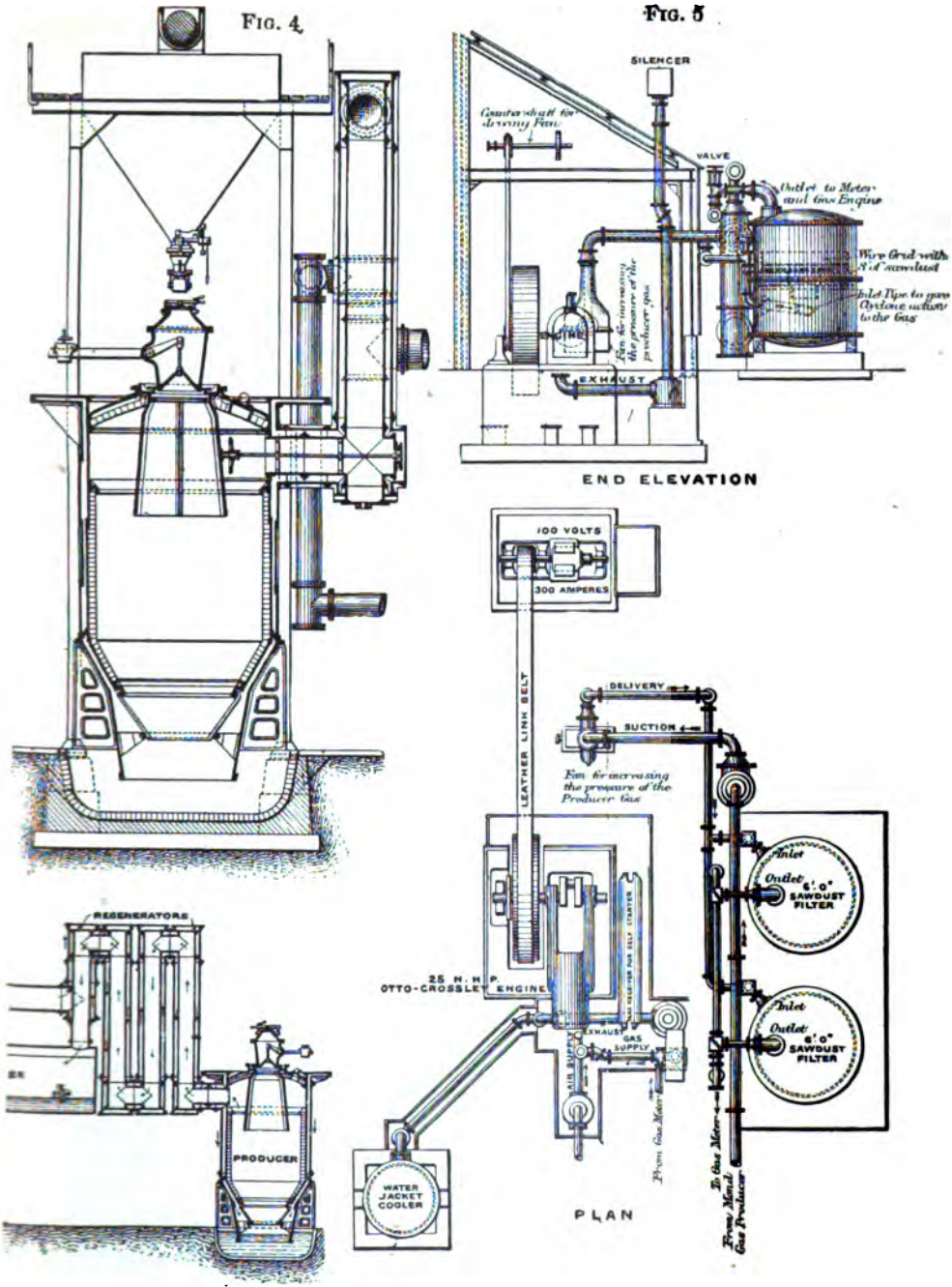


FIG. 4.—CROSS-SECTION OF PRODUCER.

FIG. 5.—END ELEVATION AND PLAN OF ARRANGEMENT.

THE MOND GAS PRODUCING PLANT.

so favorable by Mr. Humphrey as to lead him to make the following statements:

"When the difficulties of building satisfactory gas engines of say 500 h. p. and 1,000 h. p. have been overcome, they will inevitably replace steam engines in large central electric light and power stations, even as they are already doing on a smaller scale. . . . The transmission of power from the South Yorkshire or Midland coal fields to the metropolis, and its subsequent distribution at a very cheap rate, is undoubtedly to be looked forward to by metropolitan manufacturers, as well as by the present users of electric power. . . . It cannot be foretold what will be the ultimate degree of economy realized, but it should certainly be possible to supply every house in London with gas for heating and ventilating purposes, and electric current for lighting, at such a price that no householder would think of consuming coal in an open grate, or polluting the air of his rooms by burning illuminating gas. . . . Also the expenditure of England in nitrogenous compounds for fertilizing agents, amounting to about £2,000,000 per annum, would, as the system of gas producers became general, be changed to an annual income arising from the sale of the surplus sulphate of ammonia in foreign markets."

Mr. Humphrey gives the following table, showing the composition of Mond producer gas in comparison with six other varieties of gas:

ANALYSIS AND CALORIFIC VALUE OF VARIOUS GASES (PRODUCTS COOLED TO 18° C.).
(Volume Per Cent.)

Components.	Mond Gas from Bit. Slack.	Siemens Producer Gas.	Dawson Gas from Anth.	Lencauciez Producer Gas from Anth.	Solvay Coke Oven Gas.	Coal Gas (Illuminating).	Pittsburg Natural Gas.
Hydrogen, H	24·8	8·6	18·78	20·0	56·9	48·0	22·0
Marsh gas, CH_4	2·3	2·4	0·31	.....	22·6	39·5	67·0
C_nH_{2n} gases	*Nil.*	*Nil.*	0·31	4·0	3·0	3·8	6·0
Carbonmonoxide, CO	13·2	24·4	25·07	21·0	8·7	7·5	0·6
Nitrogen, N	46·8	59·4	48·98	49·5	5·8	0·5	3·0
Carbondioxide, CO_2	12·9	5·2	6·57	5·0	3·0	*Nil.*	0·6
Total volume (approximate)	100·0	100·0	100·00	100·0	100·0	100·0	100·0
Total combustible gases	40·3	35·4	44·42	45·0	91·2	98·8	95·6
Theoretical:							
Air required for combustion	112·4	101·4	113·20	154·0	410·0	581·0	806·5
Calorific value per cu. ft., lb. C°. units	85·9	74·7	88·90	115·3	284·0	381·0	495·8
Calorific value per cu. ft., B. T. U. units	154·6	134·5	160·00	207·5	511·0	686·0	892·4

A Mond producer has recently been erected by the Solvay Process Co. at its works in Detroit, Mich. The writer has been furnished with an analysis of the gas made at this plant, as follows: CO_2, 14·1%; O, 0·3%; N, 42·9%; H, 25·9%; CH_4, 4·1%; CO, 12·7%; total, 100. Combustible, 42·7%; calories per cu. m. equals 1,540; B.T.U. per cu. ft., 173.

It will be noticed from the above analyses that the Mond producer gas is very high in carbonic acid. The cause of this is not explained in Mr. Humphrey's paper, but the writer would suggest that it may be due to the low temperature in the producer. Mr. Humphrey states: "The producer is worked at a much lower temperature than usual, so that the resultant ammonia is not decomposed, and the fuel does not cake, and no clinker is made. The low temperature results from and is preserved uniform by the large quantity of superheated steam

introduced with the air, amounting to more than twice the weight of the fuel dealt with."

The low temperature may be useful in preserving the ammonia, a valuable by-product, and also for preventing the forming of cake and clinker, but it is evident that the operation of the process is far from perfect. The primary combustion of the fuel at the grate produces CO_2 which would all be reduced to CO if it was passed through a sufficient depth of highly heated carbon, but the steam may reduce the temperature of the upper portion of the bed of coke so that the reducing action $CO_2+C=2\ CO$ does not take place to the degree that would be desirable in a good gas-making process. It appears that there is room for further improvement of this process, to reduce the quantity of CO_2 in the gas.

10. *The Smith Gas Producer.*—A new type of gas producer has recently been invented by M. V. Smith, of the American Stoker Co., New York.* It is intended to produce from soft coal a gas entirely free from tarry matter. The novel feature of this process is the complete combustion of the volatile matter of the coal, forming carbon dioxide and steam, with the subsequent conversion of these products into carbon monoxide and hydrogen.

In construction the producer is cylindrical in shape similar to the ordinary steam blown producer for anthracite coal or coke. The coal is fed in at the bottom by means of an underfed stoker and immediately upon its entrance is met by a strong air blast which completely burns the tarry matter and distillates to carbon dioxide, gas and steam. These gaseous products become highly superheated and pass upward through a deep bed of hot coke in the upper part of the producer, where the carbon dioxide is converted into carbon monoxide by the reaction $CO_2+C=2CO$, and the steam is decomposed, forming water gas by the reaction $H_2O+C=CO+2H$. An additional quantity of steam may be blown into the coke if desired in order to reduce the temperature and to increase the quantity of water gas. The final product is a mixture of carbon monoxide and water gas diluted by the nitrogen carried in with the air. There have been no published results as yet of this producer, but there is apparently no reason why it should not furnish an excellent heating gas especially adapted for use in gas engines.

Crude Petroleum and Its Products as Fuel.—An article under this title, by Mr. H. Tweddle, in *Engineering and Mining Journal* of October 14, 21, and 28, 1899, treats exhaustively of the properties of the petroleum products that may be used as fuel, and of the best ways of burning them. Space will allow of a brief condensation only of this article. Of the several crude petroleum distillates, gasoline of more than 74°B. will never be used extensively for fuel under boilers, as it is of greater value when used directly in the cylinders of a gasoline engine. Benzene, 55 to 74°B., is the best of all liquid fuels, but its use is restricted owing to the care with which it has to be handled. Kerosene, 48 to 55°B., is an excellent fuel, but expensive. The heavier distillates, known as neutral or solar oils have no particular advantage as fuels over kerosene.

Crude petroleum may contain any portion of benzene and kerosene up to

* British Patent No. 2,506, of 1900. *Engineering*, March 30, 1900.

nearly 90%, varying with the locality where it is produced. Roughly, American crude petroleum contains 50 to 75% of kerosene and benzene; Russian, from 15 to 50%; Peruvian, from 15 to 50%. If distillation is stopped after the benzenes and kerosenes have been run off, there remains in the still an oil known by the various names of residuum, reduced oil, tar, fuel oil, etc.

If the distillation of this residuum is pushed still farther, neutral and lubricating oils distil over, or else, with certain forms of stills, decomposition sets in, and various products distil over, leaving a small amount of coke in the still.

The demand for mineral lubricating oils is so great in the United States that but little residuum would be placed on the market at a price which would render it available as a fuel oil. In Russia, however, where the crude oil contains a low percentage of kerosene, there is an enormous surplus of residuum, which cannot all be used for the manufacture of lubricating oils. It is used for fuel in all possible places, and is the fuel oil par excellence for marine and locomotive work where a perfectly safe oil is required. It is now distributed largely over the Russian Empire, and in 1890 some 600,000 tons were used for interior navigation in Russia alone, and the consumption has been constantly increasing.

In the United States few of the products of crude oil will be available for fuel; benzene will be largely used for enriching gas or for small motors, kerosene for illuminating purposes and the residuum for making lubricating oils. The petroleum fuel consumed in the United States is almost restricted to the use of crude oil, and this is not the fuel to suit the general consumer, especially if he is to use the oil for either railroad or marine purposes. Crude oil is a most excellent and easily handled fuel, but it must be used with caution, and is absolutely unfit for a locomotive or steamer, since, in case of accident, it may catch fire and spread with startling and deadly rapidity.

It is probable that in the future, petroleum fuel will be used more for marine purposes, on account of economy in space and weight. Next to this, mining and metallurgical purposes will be the largest consumer, owing, first, to the advantage of less weight to transport to regions difficult of access, and, secondly, to the ease of manipulation. California petroleum will probably be largely used for this purpose, as the production of crude petroleum there is being rapidly increased, and the oil is better suited for fuel than for refining purposes, owing to the small proportion of volatile constituents and large proportion of heavy hydrocarbons. It is just the contrary of the petroleum found in the Eastern States, which is especially adapted to the manufacture of illuminating oils, owing to the large proportion of volatile hydrocarbons present.

The heating value of petroleum is about 21,000 B.T.U. per lb. (including the latent heat in the steam formed by burning the hydrogen) or, about 45% more than that of pure carbon.

The simplest and best way of burning liquid fuel is to inject it in the form of spray by a jet of steam into the furnace, allowing the right amount of air to mix with it. The number of injectors or burners that have been devised for this purpose is legion. If we give the steam jet a fan-shaped opening, the greater part of the oil will be delivered at the sides and we shall have a wide and short flame. If, on the contrary, we desire a long, narrow flame, the steam jet is given a

concave opening, and most of the oil is then delivered on the center of the steam jet and is propelled forward to a considerable distance.

To obtain the greatest efficiency from fuel oil, it should be burned in a confined combustion chamber so as to obtain the highest possible temperature. To do this, the chamber must be of a refractory, non-conducting substance which soon becomes heated to incandescence, and all gases, together with the incoming air, pass through this focus of heat.

When the oil is injected with a steam jet better results are obtained than when air is used as the pulverizing medium; and the steam injected oil seems to have a much softer flame so that it is much easier on the furnace plates. For forges, or furnaces for heating iron or other metallurgical operations, air-injected oil is more suitable.

Gas, Gasoline and Oil Engines versus Steam Engines.—Much has been written in recent years concerning the prospect of supplanting the steam engine by engines in which the fuel is consumed within the cylinder, but thus far these engines have made but slow progress in this direction. About 600 h. p. is the largest gas engine that has yet been built, while 10,000 h. p. steam engines are not uncommon. The chief advantage possessed by the gas and oil engine over the steam engine is that it can produce a horse power with a smaller expenditure of fuel in the ratio of about 1 to 1·5. This is offset by the fact that the fuel usually is more expensive per given weight, and that the engine and gas producer combined not only cost more than the steam engine and boiler, but are more expensive to operate and maintain.

The internal combustion engine finds its field in special locations and conditions, as in automobile carriages, small yachts and torpedo boats, small powers (1 to 50 h. p.) where the use of a steam boiler entails more trouble than a gasoline motor and in places where water is scarce or bad. The field of the gas engine will be greatly increased if fuel gas can be distributed cheaply, as may be the case when by-product retort coke ovens have been generally introduced; under this condition the gas engine may become a serious competitor of the steam engines in sizes up to 500 h. p. It is doubtful, however, if a large electric lighting and power plant of 10,000 h. p. can ever be built to run with gas engines as cheaply as with steam engines, with the modern appurtenances of coal handling machinery, mechanical stoking, economizers and superheated steam.

One horse power hour from one pound of Pocahontas coal is already within the possibilities of steam engine practice, a figure very close to the best gas-engine records from anthracite coal gas. With the fuel consumption of steam engines reduced to this figure, the only hope for a gas engine to become a serious competitor is that gas from retort coke ovens may be sold at a price below its fuel value—the expense of running the coke oven being paid by the coke and by the valuable chemicals which are obtained from the tarry by-products and the ammoniacal liquor. Until this happens the steam engine will continue to be the machine for utilizing the greater part of the coal that is used for power purposes.

COPPER.

ALTHOUGH the production of copper in the United States in 1898 (535,900,232 lb.) was considered enormous it was exceeded in 1899, when it reached 581,319,091 lb. The Montana mines, which in 1898 showed a production below that of the previous year, showed some improvement, just exceeding the production in 1897. Arizona again showed increased production, each of the large producers increasing their output. Lake Superior, however, fell off mainly through the decline in the product of the Tamarack mine. The Calumet & Hecla with a production of 98,002,137 lb. surpassed any previous year's record. The average price of Lake copper during 1899 was 17·61c. per lb. as against 12·03c. in the previous year. The production of copper sulphate increased in 1899 to 67,903,370 lb., against 55,119,361 lb. in 1898; the price also increased, averaging 5·20c. per lb., against 3·86c. in the previous year. The home consumption slightly declined. The total production of electrolytic copper by American refiners was 386,410,356 lb. in 1899, against 314,107,776 lb. in 1898. The stock of copper on hand at the end of 1899 was 88,722,559 lb., against 54,361,470 lb. at the beginning of the year, showing considerable reduction. These figures include copper in course of treatment at electrolytic works, but not the mineral carried at Buffalo by the Calumet & Hecla Co., nor the stock at railway depots and in second hands. The statistics of imports reported in the subjoined table are compiled from direct returns from the importers, and the statistics of the Treasury Department.

COPPER PRODUCTION IN THE UNITED STATES. (LB. OF FINE COPPER.)

States.	1896.		1897.		1898.		1899.	
	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.
Arizona	73,745,321	32,922	81,019,922	36,170	110,823,864	49,475	125,377,758	55,972
California	1,971,545	880	14,129,920	6,308	21,543,229	9,618	23,915,486	10,677
Colorado	9,539,245	4,259	9,437,663	4,213	10,870,969	4,853	10,614,252	4,739
Michigan	144,058,524	64,312	145,839,749	65,107	156,669,098	69,951	155,845,786	69,574
Montana	228,958,164	102,213	237,158,540	105,874	216,979,334	96,866	237,953,951	106,229
Utah	3,550,050	1,585	3,854,821	1,721	5,385,246	2,404	9,310,344	4,156
Eastern and Southern States	3,750,124	1,674	3,727,939	1,664	4,478,218	1,999	3,804,617	1,698
All others	2,050,000	915	2,018,929	901	2,134,999	953	4,989,000	2,227
Copper in sulphate (*b*)	12,183,210	5,439	4,182,812	1,867	7,015,375	3,132	9,307,897	4,245
Total domestic production	479,806,183	214,149	501,370,295	223,825	535,900,232	239,241	581,319,091	259,517
Stock January 1	86,961,280	38,822	83,600,000	37,321	48,882,143	21,821	54,361,470	24,269
Imports bars, ingots, old, and ores	*a*27,404,087	12,234	*a*26,938,254	12,026	43,479,288	19,410	94,568,050	42,218
Total supply	594,171,550	265,255	611,908,549	273,172	628,261,663	280,474	730,248,611	326,004
Deduct exports	281,905,217	125,851	288,626,240	128,851	299,765,054	*c*133,824	249,923,941	*c* 111,573
Deduct consumption	228,666,333	102,083	274,400,166	122,500	274,135,139	122,382	391,602,111	174,832
Stock December 31	83,600,000	37,321	48,882,143	21,821	54,361,470	24,269	88,722,559	39,608

(*a*) This includes copper imported in low-grade Spanish and other pyrites chiefly for sulphur, and the copper imported from Canada in copper-nickel matte, in which the nickel is the metal of chief value: also the copper in certain gold and silver ores. These items did not until 1898 appear in the United States statistics of imports. (*b*) Including only the copper in sulphate obtained as a by-product in 1897 and 1898; in 1896 all the sulphate produced was reckoned, which makes the total production of copper somewhat larger than it ought to be on account of certain duplications. (*c*) Not including Mexican copper en route for Europe.

One of the most important things that has happened in the copper market was the formation early in the year of the Amalgamated Copper Co., which is closely identified with the Standard Oil Co. The public flotation was a pronounced success, although the preliminary statement of the company did not disclose what it was proposed to acquire. It is understood, however, that the new company has control of the Anaconda, Boston & Montana, Butte & Boston, the undeveloped Washoe group, formerly owned by Marcus Daly and J. B. Haggin, as well as large interests on Lake Superior. The Amalgamated Copper Co. is not understood to have invested in Arizona mines or those in other countries.

Arizona.—The following table shows the output of individual mines in this territory during the past and previous years:

COPPER PRODUCTION IN ARIZONA.

(Pounds of fine copper.)

Mines.	1894.	1895.	1896.	1897.	1898.	1899.
Arizona Cop. Co..	9,985,812	11,806,910	13,042,000	18,727,911	18,169,096	19,072,709
Copper Queen...	12,688,372	15,741,781	23,298,150	23,999,873	33,749,390	36,901,684
Commercial.....	90,805	*Nil.*	46,040	*Nil.*	*Nil.*	*Nil.*
Detroit.....	5,777,744	3,790,128	7,016,848	8,405,138	11,428,992	13,906,258
Old Dominion....	4,839,396	*Nil.*	5,650,000	2,000,000	1,800,000	6,300,000
United Verde....	10,904,453	16,491,402	22,327,950	31,355,025	42,328,926	43,995,982
United Globe....	*Nil.*	*Nil.*	2,020,565	1,241,975	2,847,460	4,451,180
Other mines.....	214,536	997,232	344,268	290,000	*e* 500,000	*e* 750,000
Totals.....	44,581,108	48,329,403	73,745,321	81,019,922	110,823,864	125,327,758

(By James Douglas.)—Arizona has proved to be the only one of the three important districts of copper mining which has made any notable increase of production during the past year. The figures for 1898 were 110,823,864 lb. The output for 1899 reached 125,327,758 lb. The stress of industrial work prevailing during the past year has prevented the completion of certain improvements at some of the large mines, but the work under way will again tell upon the coming year's production and raise it somewhat above that of the year just closed.

The Verde mines are said to be enlarging their plant; the improvements on the Copper Queen plant, which will slightly increase the Bisbee production, have been made; the Arizona Copper Co. has relaid the Arizona & New Mexico Railway with heavier rails so as to meet the increased requirements of the Arizona Copper Co., and its neighbor, the Detroit Copper Co. The plants of both these companies are being enlarged to a moderate degree, though not to an extent which will produce any large increase in product output. The older companies at Globe are still impeded in their progress by the lack of basic ores with which to make a desirable furnace mixture. In Globe, as everywhere in Arizona except at Jerome and Bisbee, the ores are distinctly acid, and therefore unsuitable for smelting unless rich enough to be diluted with large quantities of barren basic flux. Of late the United Globe mines have inaugurated a new departure, shipping its richer silicious ores to Bisbee, where they are used for converter lining and where basic ores are more abundant than in Gila County. On the solution of the problem of how to handle economically the

very large bodies of very lean aluminous and silicious ores which exist in the southern counties of the Territory depends in great measure her future position as a copper producer. In Graham, Gila, Pinal and Pima counties, and in fact throughout the whole of southeast and southwest Arizona, there are large areas of copper stained rock resulting from the decay of sulphide ores in a feldspathic gangue. At the Arizona Copper Co. these ores are first concentrated, whereby are obtained a rich jig product and tailings somewhat poorer than the original ore. These tailings are carried to the leaching tanks where the copper is extracted by sulphuric acid made by the ordinary chamber process from the company's own sulphide ores roasted in kilns. From the leach-liquors the copper is in part crystallized out and sold as bluestone, and in part precipitated by metallic iron. In the neighborhood of Globe there are very large quantities of lean silicious ore of, it is fair to assume, 3 to 4% copper contents, with, of course, limited quantities of ore of higher grade. A small plant has been erected within the last twelve months to treat the ores of the Black Warrior Group. The solvent used is sulphuric acid imported from California. The precipitant is metallic iron. The structural peculiarity of the plant is that the leaching vats are mounted on trunnions and can be readily tilted and their contents thrown into waste vats. The plant is, however, as yet small and its production insignificant. The scarcity of water, and in many localities the scarcity of heavy sulphide ores, suitable for the generation of sulphuric acid, are obstacles to any very rapid extension of the leaching methods now in vogue. It is, however, in this direction that progress will probably be made through the adoption of some regenerative process. A method such as the ammonia method, where the solvent will not be wasted in attacking worthless ingredients, would seem to be peculiarly applicable to the wants of Arizona. These lean surface carbonates all represent sulphides in depth. Experience heretofore tends to show that the average copper contents of the sulphide ores is low, but, being capable of concentration, a comparatively lean average may be economically valuable. The iron of such concentrates supplies a flux to a limited extent for handling the silicious and aluminous carbonates, and, when roasted, an acid solvent. There is no field where metallurgical skill could be better applied than in handling these vast quantities of very low grade ore with which large areas of the rocks of this Territory are permeated.

The new enterprises which the high price of copper has tempted Eastern capital to invest in, have as yet increased the Territory's production to a very insignificant extent. Most of them, it is fair to assume, will never do otherwise. Those that have some substantial basis for existence will before they become profitable, confirm the experience of all those who have had to do with copper mines, that an outlay of money, worry and experience that finds no place in the promoter's prospectus will have to be made before returns, in cash, balance expenditure.

California.—The Mountain Copper Co., Ltd., at Keswick continues to be the only large producer in the State, though owing to the high price of the metal a number of old mines were reopened. In the vicinity of Keswick important development work is going on. Captain De Lamar purchased the

Bully Choop group of mines and has been actively developing it with, it is said, favorable results. The King of the West mine at Acton, Los Angeles County, has shipped some ore to the Selby works at Port Costa. The vein is said to be 20 ft. wide with from 4 to 5 ft. of pay ore on the hanging wall. Work was resumed at the Union Copper mines at Copperopolis. In February, 1900, the mine had been unwatered to the 500 ft. level. Much copper was recovered from the water by precipitation with iron scrap. The Preston Peak Copper Co. was incorporated to work a mine 28 miles east of Crescent City, Siskiyou County. The vein is stated to be about 15 ft. wide and the ore to assay 10% copper and a small amount of gold. There are three productive plants in California, the Selby works at Port Costa with a daily capacity of 100 tons of ore, the Mountain Copper Co., Ltd., which treats between 500 and 600 tons daily and the furnace at Ivanpah in San Bernardino County which reduces some 40 tons of ore per day. This latter plant is the sole one in the State which produces metallic copper. The Mountain Copper Co., Ltd., ships its matte to Elizabeth, N. J., while the Selby Co. utilizes a portion of its product in manufacturing bluestone and exports the balance.

Colorado.—The production as reported by the Hon. H. A. Lee, Commissioner of Mines, was 10,614,252 lb. against 10,870,869 lb. in the previous year. The Calumet mine near Salida was the greatest producer. The Ibex mine at Leadville, however, produced a considerable amount of copper. The total production for Leadville was 3,340,064 lb.

Idaho.—The Seven Devils district in Washington County, which has promised so much and performed so little, remains in *statu quo,* although a considerable amount of good development work has been done on the Blue Jacket, which has shipped some rich ore, and on the mines of the Boston and Seven Devils' Mining Co. The Northwest Copper Co., has passed into the hands of a receiver. Successful mining in this district is said to await the completion of the Idaho Northern Railroad from Weiser. A branch from the surveyed line of this road has been surveyed from Price's Valley, a distance of some 16 miles to the mines.

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1894.	1895.	1896.	1897.	1898.	1899.
Arcadian	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*e* 2,000,000
Atlantic	4,437,609	4,832,497	4,894,328	5,109,663	4,377,899	4,675,882
Baltic	*Nil.*	*Nil.*	*Nil.*	*Nil.*	42,766	603,570
Calumet & Hecla	61,849,722	77,439,907	90,356,733	88,378,986	94,103,000	98,002,187
Central	584,950	370,381	469,248	611,172	291,339	*Nil.*
Franklin	3,602,608	3,086,933	2,746,076	2,906,384	3,568,978	1,230,000
Kearsarge	1,998,710	1,946,163	1,337,226	(*a*)	(*a*)	(*a*)
Mass	41,805	18,372	38,684	24,061	*Nil.*	61,000
National	36,390	50,128	53,762	28,959	18,441	*Nil.*
Osceola Consol'd	6,879,000	6,270,373	6,251,304	9,500,000	11,800,000	10,950,000
Quincy	15,484,014	16,304,721	16,863,477	16,924,618	16,354,061	14,301,182
Ridge	52,481	64,363	*Nil.*	12,590	.....	(*b*)
Tamarack	15,375,000	14,840,000	16,495,000	20,000,000	22,500,000	17,750,000
Tamarack Junior	2,350,000	2,605,000	2,330,000	(*a*)	(*a*)	(*a*)
Wolverine	1,665,255	1,817,806	2,195,791	2,316,296	4,588,114	4,789,015
All other mines	176,011	94,121	27,000	25,000	*e* 25,000	1,483,000
Totals	114,526,555	129,740,765	144,058,524	145,839,749	156,669,098	155,845,786

(*a*) Consolidated with the Osceola. (*b*) Consolidated with Mass. (*e*) Estimated.

Michigan.—The production at the Osceola, Quincy and Tamarack mines fell off during the year, while the Calumet & Hecla, Wolverine and the Atlantic showed an increase. Classified reports of the Atlantic, Osceola, Quincy and Tamarack Mining Cos. are given in the subjoined table:

Name of Company.	Tons Stamped.	Fine Copp'r Produced.	Yield. %	Total Receipts.	Selli'g Price Per Lb.	Mining per Ton Milled.	Cost of Ton of Ore Stamped.	
							Trans. to Mill. 7-9 Mls	Surface Expenses.
Atlantic.....	380,781	4,675,862	0·614	$802,304·57	17·16c.	75·4c.	6·5c.	28·2c.
Osceola.....	546,326	10,500,000	1·04	1,791,471·01	(b)	(b)	(b)	(b)
Quincy.....	559,164	14,301,189	1·27	496,758·31	17·13c.	.....	1·518c.	d7·1c. (c)
Tamarack...	631,090	17,000,000	1·33	2,952,098·91	(b)	(b)	(b)	(b)

Name of Company.	Cost per Ton of Ore Stamped. (Continued.)			Total per Lb.	Total per Ton Milled.	Net Profit.		Dividends.
	Stamping, per Ton.	Improvements.	Smelting & Brokerage.			Per Lb.	Per Ton, Milled.	
Atlantic.....	28·35c.	20·62c.	a 10·34c.	13·19c.	$1·711	4·08c.	39c.	$90,000
Osceola.....	26·30c.	(b)	(b)	11·05c.	2·300	(b)	96c.	558,450
Quincy	22·16c.	72·08c.	e 27·04c.	10·96c.	2·808	6·49c.	$1·66	950,000
Tamarack....	22·79c.	65·3c.	(b)	13·00c.	3·500	4·35c.	1·17	600,000

(a) Includes office expenses. (b) Data not available in report. (c) All running expenses at mine, and surface expenses. (d) Taxes in Michigan. (e) Smelting, transportation, brokerage, office and other expenses.

The foregoing table summarizes figures from the reports of Lake Superior companies, which have been issued to the present time. It will be noticed that while the Atlantic and Quincy give comparatively full information, the Osceola and Tamarack mines do not give the detailed information that model reports should contain. Of the mines in the preceding table the Tamarack has crushed the richest ore, being 1·33% fine copper in contents against 1·27% fine copper at the Quincy mine; yet while the Tamarack crushed a greater quantity of ore than the Quincy the cost of production was 13c. per lb. against 10·96c. at the Quincy. In both mines large expenditures were made on capital and construction account. The gross sums were almost equal in each case, but the cost per ton of ore treated at the Quincy was, on account of the smaller tonnage stamped, somewhat above that at the Tamarack. It is somewhat remarkable that the total amount of expenditure at the Tamarack divided by the total amount of fine copper produced, gives an absolutely even cost of 13c. per lb.

The report of the Centennial Mining Co., for 11 months ending April 30, 1899, shows a balance on hand of $288,683, including $27,000 in copper unsold. No. 1 shaft on the Osceola was sunk to the ninth level. No. 2 shaft on the same lode was sunk below the tenth level. No drifting or stoping was done during the year on the Calumet conglomerate; 5,795 ft. were drifted and 755·5 ft. of shafts were sunk on the Osceola lode; 3,800 fathoms were stoped in this ground mainly in the upper levels. The reserves opened out during this period on this lode are said to be large. The directors still have faith in the Calumet conglomerate and hope to explore it thoroughly in future. Work on the Kearsarge lode has been suspended, as it proved to be unsatisfactory.

The report of the Wolverine Mining Co. for the year ending June 30, 1899, showed a production of 4,700,373 lb. of copper at an expense of $362,454. A profit of $313,405 was made, from which dividends amounting to $150,000

were declared, $104,500 paid for property, and a balance of $58,905 placed to reserve. The total reserve at the end of the year stood at $227,690. Considerable development work was done during the year, which was satisfactory, except on the west lode; 217,243 tons were hoisted, from which 32,444 tons of waste or material too low grade to treat were discarded; 184,799 tons was stamped, which yielded 5,360,740 lb. of mineral, going 87% refined copper, an average of 21·64 lb. per ton hoisted and 25·43 lb. per ton stamped. The cost per ton of rock hoisted was $1·33, and $1·56 per ton of rock stamped. The cost per pound of copper produced was 7·71c., divided as follows: Underground expenses, 3·93c.; rock house, 0·33c.; stamp mill, 1·44c.; construction, 0·23c.; smelting, freight and marketing copper, 1·34c. The average price received for copper was 14·35c. per lb.

The Mohawk Copper Co. has 800 acres of ground which formerly belonged to the Fulton Mining Co. It is developing the Kearsarge lode. Three shafts have been commenced and a full equipment of compressors, hoists and other machinery was supplied. The developments on the conglomerate have been encouraging, a new mineral which was identified by Prof. Koenig of the Michigan School of Mines was found, and has been called Mohawkite. It contains some arsenic as well as a high percentage of nickel. This occurrence of nickel in conjunction with copper is interesting, and it is thought may assume some future commercial importance. One of the new companies formed during the year was the Rhode Island Copper Co. with 800 acres adjoining the Franklin, Jr. It is developing the same lode as that in the Franklin mine. The capital is $25,000,000; $200,000 was placed in the treasury for working capital. The Phœnix Consolidated Co. has 2,240 acres in Keeweenah County. The Calumet conglomerate and ash bed veins are said to run through the property. The Washington Copper Mining Co. began operations on what was considered to be the ash bed lode, but it has since been proven to be a separate vein. The capital of the company was increased from 40,000 to 100,000 shares of a par value of $25 each. The Arcadian Copper Co., which is controlled by the Amalgamated Copper Co., did considerable development work during the year. Sinking took place at six shafts and many thousand feet of drifts were run. A mill containing three steam stamps, nine double-decked slime tables and 108 jigs was erected at Grosse Point, 5·5 miles from the mine.

For dividends declared by Lake Superior mines the reader is referred to the table under "Mining Stock Exchange" in the present volume.

Montana.—The Boston & Montana company had an extremely successful year's business and disbursed $5,400,000 in dividends. Its production with that of the Butte & Boston aggregated 79,000,000 lb. of refined copper, an increase of 17,000,000 lb. above the production of the previous year. The company, however, is involved in expensive and what promises to be prolonged litigation with F. A. Heinze.

The sales of copper, silver, gold and bluestone were $11,257,280·18. The total expenses, including electrolysis, handling and commissions of sale of product were $5,202,430·95. The net income after deducting $75,004·48 for interest on debentures and appropriation for sinking fund, was $5,979,844·75. After

paying dividends of $5,400,000, $579,844·75 was put to reserve, making the balance of assets, December 31, 1899, $5,096,091·64. The company suffered to the extent of $25,140·68, which was on deposit in the Globe National Bank previous to its failure.

COPPER PRODUCTION IN MONTANA.

(Pounds of fine copper.)

Mines.	1893.	1894.	1895.	1896.	1897.	(d) 1898.	1899.
Anaconda	75,256,657	95,578,000	99,775,294	125,350,698	131,471,127	107,214,059	107,914,357
Boston & Montana	31,800,000	57,987,633	60,746,000	60,250,000	60,000,000	62,000,000	79,000,000
Butte & Boston	20,457,928			4,500,000		7,000,000	
Butte Reduction Works	2,985,485	2,282,000	3,390,000	4,225,647	7,883,795	9,685,068	10,049,629
Colorado Sm. & Mg. Co	6,703,488	5,158,730	7,750,000	9,090,680	8,911,578	7,657,938	9,572,155
Hecla Cons. Mining Co	77,565	362,897	239,631	312,445	215,431	121,089	155,719
Parrot	7,791,167	7,469,908	7,257,000	8,045,648	14,824,487	12,444,828	10,625,696
Montana Ore Purch'g Co	8,400,000	12,293,686	14,861,000	15,049,066	13,047,648	13,084,000	15,260,679
Others	827,810	2,011,901	750,000	4,242,583	804,474	(b)	5,775,716
Totals	154,300,100	183,094,755	194,768,925	a228,958,164	c237,158,540	c216,979,334	237,953,951

(a) In addition to 818,581 lb. of Canadian copper smelted by the Montana Ore Purchasing Co. and deducted in the above table, there was also deducted 1,000,000 lb. more of foreign copper estimated as having been included in the aggregate of the above returns, leaving the net amount 228,958,164 lb. (b) Included in reports of smelters itemized above. (c) Totals reported by E. B. Braden. (d) The individual reports include some copper derived outside of Montana, wherefore their sum exceeds the total as given.

The Anaconda Mining Co. produced a little more copper in 1899 than in the previous year. It was claimed that the ore had so fallen in value that it was impossible to maintain production at anything like the past year's figures, which were much lower than the production of the two preceding years, yet the production of 1898 of 107,214,059 lb. was more than equalled by the 1899 one of 107,914,357 lb. After the control of the property had passed into the hands of the Amalgamated Copper Co. the latter decided not to publish their accounts. This action was much criticised. It is stated by men familiar with Butte that the falling off in the percentage of copper in the ores treated is not due to decreased contents of the ore from the main Anaconda mine but to the low tenor of copper in the ores from the old syndicate mines which are worked in conjunction with the Anaconda. The Anaconda company, like the Boston & Montana, is much involved in litigation with F. A. Heinze. At the reduction works at Anaconda substantial changes have been made. Extremely large reverberatory furnaces with interior dimensions of 42×14 ft. have been placed in operation and are stated to have a capacity of over 150 tons per day. Wilfley concentrators have been introduced and the blast furnaces added to. These improvements tend to increase production, this having risen 2,500,000 lb. in each of the last two months of the year. The Montana Ore Purchasing Co. showed a slightly increased production, although a number of its best mines were closed or partially closed owing to existing litigation.

Nevada.—Considerable activity was shown in copper mining in this State, the Glasgow & Western Exploration Co. was actively worked and the Lone Mountain group of mines, near Elko, was taken up by a Boston syndicate, who have undertaken development work.

Some prospecting was done in the Pine Valley district, some 25 miles below Elko. The Vulcan Copper Mining and Smelting Co. have done considerable development at the mines at Siegelton near Luning, Nev. A smelter has been

erected. It is claimed that there is now about 75,000 tons of ore averaging 8% copper and $5 a ton in gold and silver in sight in their mines.

New Mexico.—The Santa Fé Copper Co., operating mines on the San Pedro grant, was reorganized with a capital of $2,500,000. According to engineers who have examined the mines the cost of mining will not exceed $2 a ton, hauling to the mill, breaking and concentrating will not exceed $1 a ton; other expenses should amount to $0·45. The ore is said to contain gold and silver in the proportion of three cents to the pound of fine copper. The old shareholders were permitted to reinvest on the payment of $5 and the relinquishment of five old shares for each new one. It had been sold for the benefit of bondholders to the extent of $300,000 and was bought in by Leonard Lewisohn.

Tennessee.—The Tennessee Copper Co. was formed to take over the Pittsburg, Tennessee, London and Burra Burra mines at Ducktown. It was formed under the auspices of Messrs. Lewisohn, and Burrage and Rogers, who are closely identified with the Amalgamated Copper Co. The company has a capital of $5,000,000 in 200,000 shares of $25 each. It is understood that $1,000,000 has been placed in the treasury for working capital. It is expected that production will begin early in this year and will be maintained at the rate of 1,000,000 lb. of copper per month.

Utah.—The leading producer in Utah has been the Utah Consolidated Gold Mines, Ltd. The total production, however, was not large, amounting to 7,812,875 lb., which is an increase above the production of the previous year. The company has erected a new plant, from which copper was produced on May 17, 1899. Wethey calcining furnaces are used from which the roasted ore is automatically discharged into bins and again delivered without labor to an electric tramway. The roasted ore is treated in reverberatory furnaces which discharge the matte directly into converters. The plant has a capacity of 250 tons per day. The pig copper is claimed to run $60 gold and 40 oz. silver per ton. The Centennial Eureka mine was purchased after an examination by J. W. Neill, George W. Small, R. A. Parker and A. F. Holden by capitalists connected with the United States Oil Co. While the property is worked independently of the mines belonging to the United States Oil Co. at Bingham their ore will probably be treated in conjunction with the sulphide ores of Bingham as they are silicious and require fluxing material. The St. George Copper Mining Co. in Washington County, has worked the old Ibex mine. It produced 151,742 lb. of copper, 5,853 lb. of matte and 264,828 lb. of shipping ore averaging 30% copper, during the year. The Horn Silver Mining Co. produced 1,612 tons of first class copper ore during 1899 as well as its usual amount of lead-silver ores. Its production was 717,120 lb. of copper. The first class copper ore netted the company $45·45 per ton at the mine.

Vermont.—The Ely copper mines which had been largely productive under the administration of the Messrs. Dawson, father and son, were purchased by George Westinghouse during the year. It is many years since the property had been worked, but it is anticipated that results under careful management, which is now assured, will be remunerative to the owners. The Eureka mine, which is in the same vicinity, formerly dressed its ore to 10% copper, before

shipping to the smelters at Copperfield. The Union mine was worked to a depth of about 500 ft. Ore from this property in 1879, during which time 9,039 tons was shipped to the smelters, averaged 9·15% copper. In these properties the ore exists in chimneys in a gneissoid and mica-schist formation.

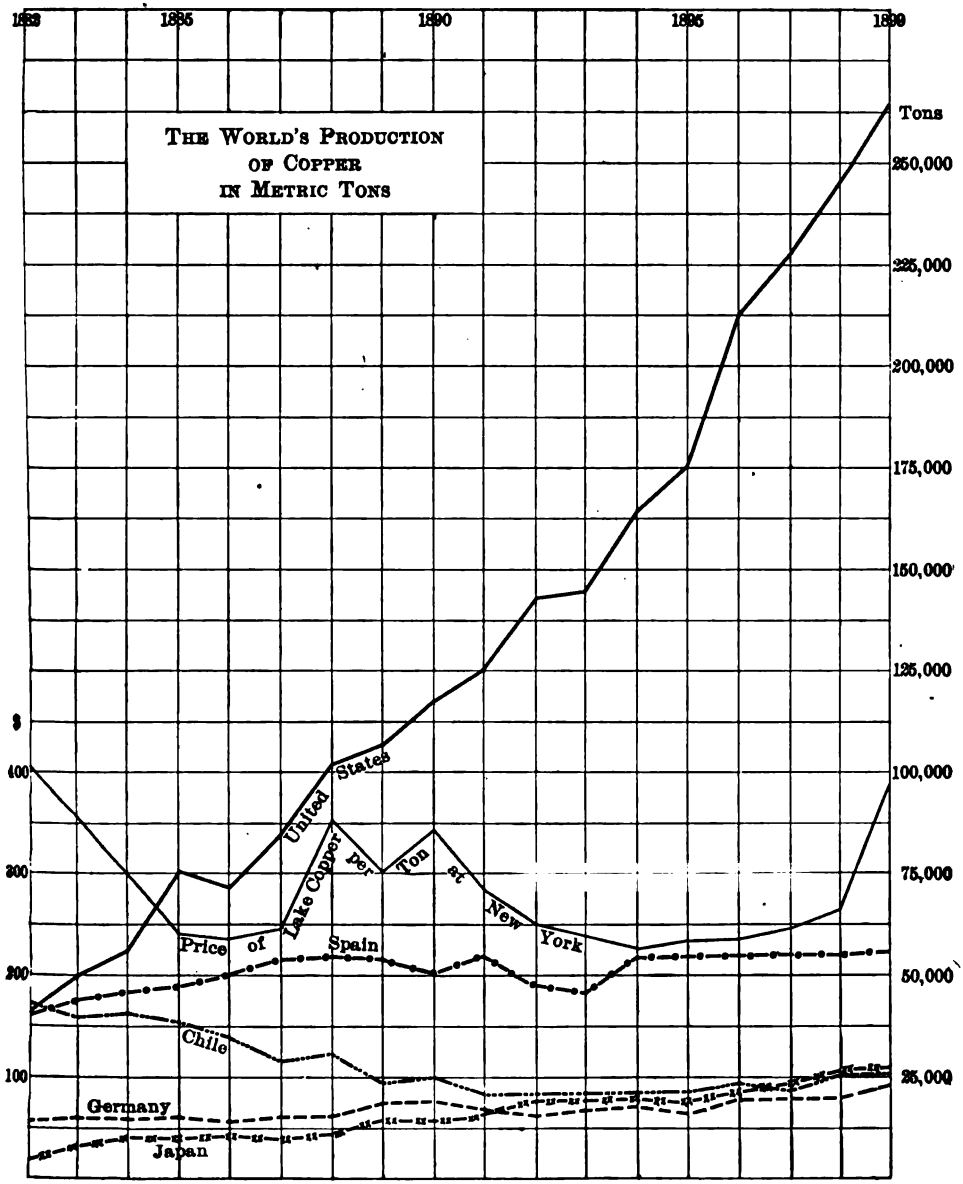
Washington.—According to N. H. Norton, formerly inspector of coal mines in the State of Washington, there is a belt of gold and copper mines in King County, Washington, near Snoqualmie River, an elevation of 5,000 ft., which has not been developed though it has been known to the Indians for many years. The owner of the Dutch Miller mine has purchased a smelter with the intention of placing it on Wapatus Lake.

Wyoming.—The Battle Lake copper mines shipped a small quantity of ore of fair grade. Only a small amount of work has been done, the shaft being at present only 80 ft. deep.

Copper Mining in Foreign Countries in 1899.

Australia.—Copper mining in New South Wales is described later on under this caption. In Victoria the production has been very small, the only mine working being the Bethanga, which is worked mainly for gold. Some development work was done on the Thompson River copper mine. From 15 to 17 ft. of low grade copper ore is reported here. In West Australia the Anaconda and Malcolm mines near Mount Malcolm, 75 miles north of Menzies, shipped some 215 tons of ore to Wallaroo. It returned 28·5% copper. The shipment netted about $18,000. There is said to be a large quantity of ore in sight and the ore-body is stated to be 50 ft. wide. A furnace was erected on the property early in the present year. Charcoal has to be used as fuel as coke would be too expensive, owing to the distance of freighting necessary. Another West Australian mine is owned by the Mount Pleasant syndicate, at Broad Arrow. Several shipments were made with satisfactory results, it is claimed. There has been considerable activity in copper mining in South Australia, a number of the abandoned old-time mines having been reopened. The leading producer of the Colony has been the Wallaroo; 1,820 men are employed at this and the Moonta, an amalgamated property. The shaft has reached the depth of 1,620 ft. and on the 1,570 ft. level the lode is 12 ft. wide, exposing a large amount of ore in sight. The mine is wet and three large pumping engines in as many shafts are required to keep it drained.

Among the many copper flotations in London during the year, the Paramatta Copper Co., Ltd., appeared. These mines were originally discovered shortly after the Wallaroo and Moonta mines were found. It is said considerable profits were realized from them, but work was abandoned in 1879, as the ore became of too low grade in depth to yield a profit. The mine at the time of flotation was full of water and though H. R. Handcock stated that the mine could be worked with a profit with a proper concentrating plant under modern conditions, the fact that the vendors demanded £130,000 for the property caused much comment. The Balhannah copper mine, on which work was suspended for a number of years, and which was completely under water, was reopened. The lode is said to be 9 ft. wide and to contain a fair amount of gold.



Of the world's total copper production in 1899, the output of the United States amounted to over 55%, while the five countries appearing in the above table contributed 83%. In 1882 the output of the United States was only 22% of the world's total production. About two-thirds of the copper produced in the United States in 1899 was electrolytically refined, the actual amount being 386,410,356 lb., against 314,107,776 lb. in 1898 and 250,000,000 lb. (estimated) in 1897.

THE WORLD'S COPPER PRODUCTION, 1895–99. (a)

Countries.	1895.		1896.		1897.		1898.		1899.	
	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.	Tons of 2240 Lb.	Metric Tons.
Algeria	35	35	*Nil.*	*Nil.*	*Nil.*	*Nil.*	50	51	*Nil.*	*Nil.*
Argentina	150	152	100	102	200	203	125	127	65	66
Australasia	10,000	10,160	11,000	11,176	17,000	17,272	18,000	18,288	20,750	21,082
Austria-Hungary	1,310	1,331	1,285	1,306	1,655	1,681	1,540	1,565	1,505	1,529
Bolivia	2,250	2,286	2,000	2 032	2,200	2,235	2,050	2,083	2,500	2,540
Canada	3,924	3,987	4,190	4,225	5,905	5,999	8,040	8,169	6,730	6,838
Cape of Good Hope										
Cape Company	5,350	5,436	5,470	5.558	5,290	5,375	4,660	4,735	4,140	4,206
Namaqua	1,730	1,758	1.980	2,012	2,150	2,215	2,400	2,438	2,350	2,388
Chile	22,075	22,428	23,500	23,876	21,900	22,250	24,850	25,248	25,000	25,400
Germany:										
Mansfeld	14,836	15,079	18,238	18,536	17.960	18,247	18,045	18,334	20,785	21,118
Other mines	1,695	1.722	1,800	1,829	2,185	2,220	2,040	2,073	2,675	2,718
Italy	2,500	2,540	3,400	3,454	3,480	3,536	3,039	3,088	3,000	3,048
Japan	18,430	18,725	21,000	21,336	23,000	23,368	25,175	25,578	27,560	28,001
Mexico:										
Boleo	10,450	10,617	9,940	10,099	10,170	10,334	9,436	9,587	10,338	10,503
Other mines	1,170	1,189	1,210	1.239	1,200	1,219	6,232	6,332	8,788	8,924
Newfoundland	1,800	1,829	1,800	1,829	1,800	1,829	2,100	2.134	2,700	2,743
Norway	2,685	2,728	2,500	2.540	3,450	3,505	3,615	3,673	3,610	3,668
Russia	5,280	5,364	5,100	5,181	6,025	6,121	6,000	6,096	6,000	6,096
Peru	450	457	740	751	1,000	1,016	3,040	3,089	5,165	5,248
Spain-Portugal:										
Rio Tinto	32,985	33,513	34,501	35,055	33,900	34,442	33,705	34,244	34,370	3,9240
Tharsis	12,380	12,638	11,040	11,217	12,000	12,192	12,000	12,192	11,500	11,684
Mason & Barry	4,100	4,166	3,381	3,435	4,300	4,369	3,600	3,658	3,600	3,658
Sevilla	1,050	1,089	1,025	1,041	810	823	800	813	1,200	1,219
Other mines	4,300	4,369	3,400	3,455	3,050	3,099	3,120	3,170	3,550	3,607
Sweden	515	523	500	508	545	554	480	488	520	528
United Kingdom	580	589	580	589	555	564	640	650	550	559
United States	172,524	175,294	208,760	212,112	223,825	227,420	239,241	243,069	259,517	263,685
Totals	334,554	339,994	378,440	384,493	405,350	412,088	434,023	440,972	468,463	475,976

(a) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by THE MINERAL INDUSTRY direct from official sources.

Development work is going on rapidly at the properties of the Chillagoe Railway & Mines Co., in Queensland, though it will be a considerable time before it will figure as a producer.

During March a loan of £2,000,000 covered by debentures had been secured, and since then £60,000 has been derived from the proceeds of an option on 50,000 shares. Work can be divided into three heads: railway, mines and smelting works. Of the 103 miles of railway necessary he was confident that 50 miles would be completed during 1899. Mine development is confined to the Mungana, Ruddigore and Ti-Tree groups. The capitalization was reduced from £1,800,000 at which it was incorporated to £900,000 by making the shares at the nominal value of £1 each instead of £2; 600,000 shares only were issued, the balance being reserved for future issue. The Mount Chalmers Copper Mine, Ltd., was formed in London with a capital of £300,000 to work properties at Mount Chalmers. The lode is said to have been proven for a length of 600 ft. with a large width. The average value was about 6%; £50,000 only was provided for working capital. The Mount Garnet Co. is building furnaces to treat silver-lead and copper ores in the Chillagoe district.

Brazil.—A French company with a capital of 2,500,000 fr. of which 1,500,000 fr. was issued, has been formed to work the copper mines at Camaquam. Reports of the examining engineers state that the veins are rich and are favorably situated for economical work. The Camaquam River furnishes ample water power.

British Columbia.—Trail Creek continues to be the leading copper producer of

this province. The production increased last year from 5,232,011 lb. in 1898 to 7,783,000 lb. in 1899. The value of the ore, however, including gold and silver, fell to $17·81 per ton. Of the total of 180,300 tons shipped 92,500 tons with a gross value of $1,250,000 came from the Le Roi mines. The general average of the ore was low, including copper, gold and silver, $13·51 per ton. Although the War Eagle mine shipped only 63,500 tons its gross value was only slightly below that of the Le Roi, as the average per ton was $18. The Center Star shipped 16,700 tons with a gross value of $221,475, an average of $13·25 per ton. The Iron Mask shipped 5,378 tons at a gross value of $70,268·81, an average per ton of $13·25. The British Columbia Copper Co., Ltd., was incorporated under the laws of the State of West Virginia and registered in British Columbia under the company's act of 1897. The capital is in 200,000 shares of a par value of $5 each. A smelter is being erected with an initial capacity of 300 tons per day. It is stated that the ores are self-fluxing. Considerable amount of development work has taken place on Vancouver Island. The outcrops are as a rule high grade magnetite with traces only of phosphorus and silica. This is a capping of chalcopyrite which contains low values in gold. The outcrops are extensive, but prospecting is difficult, as the island is covered with a very heavy growth of timber and underbrush.

The Van Anda Copper & Gold Mining Co. has developed a property on Texada Island. The ore is bornite and chalcopyrite and contains an average of $9 a ton in gold. A 50-ton water jacket furnace was erected and has been in constant operation since July 13, 1899. A custom smelting plant was erected on the north fork of Kettle River in the boundary district, operated by water power. It is estimated that it will have a capacity dependent on the character of the ore from 150 to 250 tons per day for each of the two furnaces. It is anticipated that a matte converting plant will be added shortly as well as an electrolytic refinery. Promising developments occurred at the Pot Hook mine at Kamloops. On the deepest level 12 ft. of bornite ore carrying about $3 per ton in gold and silver was encountered. A concentrating plant is to be erected during the year. The Dominion Copper Co., Ltd., was formed with a capital of $5,000,000 in $1 shares; $2,000,000 of these were taken by the vendors and the remaining $3,000,000 was held part in reserve and part for the purpose of raising immediate working capital. The mines are located near Greenwood, and the ore is said to average from 3 to 5% copper and from $4 to $5 gold per ton. The expense of mining is estimated not to exceed $2 per ton, and Crow's Nest coal can be delivered at the mines at $8 per ton.

EXPORTS OF COPPER FROM CHILE AND BOLIVIA.

	1895.		1896.		1897.		1898.		1899.	
	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.	Quintals.	Fine.
Bar copper.....	443,532	443,532	459,380	459,380	418,665	418,665	448,585	448,585	400,955	400,955
Copper regulus...	20,547	8,251	67,945	34,506	86,538	42,661	81,141	41,930	72,006	32,949
Copper ores.....	192,027	35,440	133,712	24,139	94,550	20,786	324,919	64,023	646,028	118,193
Total.....	.....	487,223	.....	518,025	.....	482,112	.....	554,538	.....	552,097

NOTE.—The quintal is 101½ lb. avoirdupois.

Chile.—The copper production in Chile in the last five years has not kept pace with the increase in consumption of this metal, although the output rose from 21,000 to 25,000 tons during this period. At present there are 3,474 mines under development or producing and it is expected that this number will be increased by 1,000 in the near future. A greater part of the new concessions has been received by Chileans and for this reason a large increase in the production is not to be looked for immediately. The lack of capital and of skilled labor offer great difficulties which have to be overcome before these copper deposits can be opened up. The Government is always willing to lend every assistance possible. A report to the President of the Chilean Republic gives a very glowing account of the copper resources of that country. The mines Vallemar and Higuera are especially commented upon. The veins are said to extend for several kilometers in length and to be from 6 to 20 m. wide. They have been proved to a maximum depth of about 150 m. The average grade of the ore is said to be from 10 to 15% copper with a fair amount of gold. The country is arid and devoid of transportation facilities. A railway is recommended to tap this region. Otto Nordenskjöld of the Geological Institute of the University of Upsala, describes in its bulletin the copper deposits of Coquimbo and Atacama. Some 200,000 tons in the Atacama district are annually raised, with an average contents of 5% copper. Concentration increases the value 25%. The Copiapo Mining Co., Ltd., had a successful year's business. The profit is announced at £51,761, being an increase of £31,000 over the profits of the previous year, and being the highest year's earning of the company save in 1888; 12,500 new shares at £2 each were issued at a moderate premium to provide part of the purchase money of the Ojancos mine. The chairman stated at the annual meeting that should copper fall to £45 per ton, 10% dividends could be paid to shareholders. It is believed that the company, which has heretofore shipped its ore, will erect a smelting establishment in Chile. If so, 12,500 extra shares will be issued for construction purposes.

China.—Mines at Sankiatschang in the Province of Huanan, are worked in a crude manner by Chinamen. From 20 to 30 miners are employed and the output of concentrates is about 50 lb. daily per man. All ore below 20 to 30% copper is thrown on the dump. Roasting is done in open kilns holding 4,000 lb. of ore and requires 2,000 lb. of wood per charge. The operation lasts 10 days and is repeated several times. The ore is then smelted in a small blast furnace. The output at present is only 40,000 to 50,000 lb. copper, while formerly it was about 270,000 lb. per annum.

Ecuador.—Copper is found in the Province of Loja at Catacocha, but bad roads and remoteness preclude development. Recent discoveries have been made in the Province of Azuay, 35 miles from the coast at an altitude of about 5,000 ft. The ore of the Virginia mine is said to run 19 to 21% copper, 41 to 48 oz. silver per ton, and $1·10 to $1·50 gold. The conditions as regards fuel, water and labor are favorable and development work is being carried on.

Germany.—The Mansfeldsche Kupferschieferbauende Gewerkschaft during the first six months of 1899, after paying off interest on debts, earned a dividend of 4,037,542 marks. There was sold during the first six months of 1898 10,306

metric tons of refined copper at an average price of 1,050·70 marks per ton; 51,258 kg. of fine silver at an average price of 77·16 marks per kg.; in 1899, however, there were produced and sold during the first six months 9,739 metric tons of refined copper at an average price of 1,313·80 marks per ton and 54,517·176 kg. of fine silver at an average price of 81·50 marks per kilo. The great movement in the copper market raised the average to about 263 marks per ton above the price for the preceding year. The expenses of mining and smelting also increased. It is to be observed that the above figures are to be accepted with this explanation: for 1899 are for the first six months ending June 30th, while the figures for the preceding year include the first 10 days in July.

India.—According to Harold Harris the Singhbhoom district in Bengal is one of considerable extent. There are numerous old workings, many of which reach water level. In 1892 the Rajdolia Copper Co. pumped two shafts. Hand dressed ore was shipped, assaying 13·5% copper while the crude ore ran from 4 to 7%. Transportation was expensive, and as capital for the erection of reduction works was lacking operations were suspended. A railway, however, has been constructed within one mile of the property and it is anticipated that work will be renewed.

Japan.—Owing to recent copper demands as well as to increased consumption, the production of copper in Japan has been stimulated, due mainly to the development of electric railway systems. The largest producer of copper in the country, the Besshi, was flooded on September 20, 1899, with a large loss of life. The production of this property in 1898 was about 4,200 metric tons. It is thought that during the present year operations will be resumed.

Java.—The Nederlandische Indische Mijnbouro Maatschappij of Batavia, Java, has erected a copper matting furnace, using the pyritic process.

Mexico.—The main producer in Mexico still continues to be the Boleo Mining Co., though the Nacosari mines have become an important producer. Considerable ore from Durango and other States was smelted by the Guggenheim Bros., at Aguas Calientes. Considerable development work has taken place on the Zone concession of Inguaran in the State of Michoacan. The erection of a smelter on the Marquez River is contemplated. At this point abundant water power is available. A railway is being constructed from the mines to the Grand Pacific line which will give communication to both the City of Mexico and the ocean. In Chihuahua, 12 miles west of Santa Isabel, an 8 ft. vein containing fair grade copper ore and some gold and silver, has been developed. The Barranca del Cobre mine was purchased by Lewisohn Bros. This property was at one time owned by an English company which worked it for gold, but unsuccessfully. The vein varies from 2 to 18 ft. wide, and outcrops for 3,000 ft. in length. The ground is precipitous and the mine can be worked entirely by tunnel. It is stated that the ore will average 12% copper. The Panuco Copper Co., owned by an English company, is said to have been floated on erroneous statements and it is anticipated that the company will go into liquidation.

New Caledonia.—(By E. A. Weinberg.)—The most important copper properties are those known as the "Pilou" and the "A. O." mines, which were re-

cently acquired by the "Caledonia Copper Co., Ltd.," floated on the London market under the auspices of the London and Globe Finance Corporation, with a capital of £750,000. The Pilou mine consists of three claims, which cover an area of 3,000 acres and is situated at the foot of the Arama range of mountains on the western side of the Diahot Valley and almost opposite the harbor of Pam. From 1887 to 1890 some 2,300 tons of ore, assaying 20% copper, were extracted and shipped to Australia, in addition to which nearly 23,000 tons of ore, assaying between 6 and 12% were mined, of which a small quantity was locally smelted, while the greater portion of it was left in stock at the mines and wharves, but of which since some 5,000 tons were shipped to Australia. The country rock is mica schist, while the lode itself is in black slate. The mine is being worked at present from a three-compartment shaft and is equipped with a good winding plant. The shaft is down 380 ft., and from it short cross-cuts have been driven into the lode, from the intersection of which four levels have been extended. The ore occurs in the lode in five large shoots of lenticular shape. The average width of the lode may be placed at 4 ft. It may safely be assumed the one-quarter of the ore in sight will yield by simple hand picking an ore which will assay about 15% copper, while the balance may run from 5 to 9% copper, requiring dressing to be prepared for smelting. Chalcopyrite is the copper bearing mineral, but this is invariably associated with zinc blende and a little galena; for nearly every unit of copper there is one unit of zinc present in the ore. The silver contents vary between 2·5 and 4 oz. per ton, with a trace of gold.

The "A. O." mine, situated on the western slope of the Arama Range, covers an area of 1,100 acres. A tunnel has been driven through the mountain to establish connection with Pilou by a tramline. The country rock is the same as at the Pilou mine. The "A. O." outcrops are of an extensive character but very little development work has been done beyond cross-cutting the lode in the oxidized portion of it. Some 900 tons of ore have been stoped from the surface workings, of which about 450 tons have been dressed up by hand picking and jigging to about 14·5% copper, while the balance will assay from 6 to 8%. A new three-compartment shaft is now being sunk. The mine is also connected with the sea by a narrow gauge tramway about 2·5 miles long, but as there is not always sufficient water for the convenient loading of ores into barges, a further 2·5 miles of tramline is now being laid to deeper water at Dilah Point, at the mouth of the Diahot River, where ore hoppers will be erected.

During the time of my visit 155 convicts were working at Pilou, and at "A. O." 50, *at a cost of 0·5 fr. per day.* This is certainly very cheap labor, but it will be borne in mind that on the whole it is a very inferior one. The convict labor practically belongs to John Higginson, who received from the Administration as an indemnity in a case of compensation a cession of 300 convicts for 20 years, at 15 centimes per man per day. I understand, however, that this contract was to expire in less than two years from the time of my visit, and that the French Government will not enter into further contracts for the supply of convict labor. The probability is that the New Caledonian mine owners will endeavor to substitute the present convict labor by Tonkin

boys, of whom already a large number is being employed at some of the nickel mines and who are generally reported to give satisfaction. These men receive 12s. per month, besides daily rations consisting of 0·5 kg. rice, 0·5 kg. bread, 5 g. tea, 50 g. sugar, 500 g. meat (if salt meat 250 g.). The employer has to pay two passages from Amoy and return.

Even allowing for the development of other outcrops in the Diahot district, I do not consider it likely that the possible production for the next few years will exceed 3,000 tons of copper annually.

Newfoundland.—According to James T. Howley, F.G.S., Director of the Geological Survey, the output of the Tilt Cove copper mines belonging to the Cape Copper Co., Ltd., for the 12 months ending December 31, 1899, was 62,138 tons containing 3·5% copper by wet assay and 3,699 tons from the West mine of an average grade of 11%, making the total production 65,837 tons. The shipments from June to December totaled 86,079 tons, 80,206 tons of which from the East mine and 5,873 tons from the West. Messrs. Reid shipped a small quantity of high grade copper ore from New Bay, Notre Dame Bay. The copper mine at York Harbor is said to be developing well, although as yet no ore has been shipped. The Newfoundland Copper Co., Ltd., according to the report of the chairman at the general meeting of the shareholders, has but very little copper in its property.

Nova Scotia.—The Cape Breton Copper Co., Ltd., had entered into negotiations to purchase a considerable area of copper-bearing ground near Eagle Head, Gabarus Bay. The properties are accessible to seaboard and the ore can be transported to the smelters of the company at Sydney harbor. Active work at the Coxheath copper mines commenced on March 1st. The main shaft was unwatered and retimbered to the 120 ft. level. Considerable work was done on the 136 and 200 ft. levels, and a new lode was located about 1,000 ft. north of the Coxheath vein. It has been traced on its strike over 2,000 ft. The ore is more silicious than the Coxheath ore, but contains from $3 to $5 in gold.

Peru.—About 3,000 tons of copper ore a month is raised at Cerro de Pasco, though only about 1,000 tons is delivered at the Oroya railway terminus, owing to lack of transportation facilities. The occurrence of copper is new, as a shipment was not made until 1898. The result was so satisfactory that operations since have been confined to the production of this metal. The cost of shipping from Oroya to Swansea is about $22·50 per ton. The high expenses between the mine and Aroya will be reduced considerably when the railway is extended to Cerro de Pasco. A vein of copper ore was found on Isla de Santa, north of Chimbote. The ore averaged 25% copper and 1 oz. gold per ton. Several veins are said to exist on the main land opposite this island.

Russia.—The improved price of copper has stimulated the industry in the Ural Mountains. About five years ago the mine and smelter owners relinquished their property to the Government. The peasants, however, mined and reduced by means of crude appliances, at Kukmor and other villages. It seems probable that the industry will now be rehabilitated.

Spain.—There was considerable comment in Spain over the proposed duties

on the copper-iron ores of Huelva and Sevilla. According to *The Revista Minera* the duties and cost would almost entirely consume the profits. It gives the following figures: admitting that the ore averages 2% copper by wet assay and 3 to 4d. are obtained per unit of sulphur, and with best selected copper at £80 per ton, the price f. o. b. would be 21 pesetas. From this 5 pesetas must be deducted for mining, 7·5 pesetas for transport to the ships, and 0·25 pesetas for port charges. A tax amounting to 15 pesetas per hectare is levied on the superficial area. Charged back to the ton of ore mined this would be 0·05 pesetas per ton of ore. The tax upon the crude product amounts to 0·40 pesetas, the tax upon transportation 0·27 pesetas, export duty 2·0 pesetas, embarking duties 1·5 pesetas, a total of 16·97 pesetas, leaving a margin of only 4·03 pesetas for profit. It would not be possible to mine 2% ore were the price of copper not extremely high.

The interim report of the Rio Tinto Co., Ltd., shows a satisfactory business for the first half of 1899. The demand for pyrites has been on an increasing scale and the deliveries in the United Kingdom, the continent of Europe and the United States have been greater than at any time in the past. Up to the period covered by the report, the refining work at Cwmavon had not produced as much as in the same period of the preceding year, though the copper was satisfactory in quality. A dividend for the six months ending June 30th, of 2s. 3d. per share less income tax was declared on the 5% preference shares, and an interim dividend of 35s. per share on the ordinary shares, payable on November 1, 1899. This was at the rate of 70% per year on these latter shares.

The Tharsis Sulphur & Copper Co. extracted 195,205 tons in 1898 against 231,384 tons in the preceding year from its Tharsis mines. At the Calanas mines 308,740 tons were extracted, an increase of 63,749 tons above the production from this source in 1897, although 314,474 cubic meters of overburden had to be removed. The Calanas ore is found especially adapted for export, while the 82,145 tons produced at the Lagunazo mine were reserved for local treatment. The total quantity raised during the year was 586,090 tons exclusive of ore too low grade for treatment against 565,949 tons in 1897. Although the rain fall was less in 1898 than in the preceding year, the production of copper, which depends largely upon this was not interfered with greatly, owing to the excellent reservoirs the company had constructed; 378,085 tons of pyrites were shipped against 310,702 tons in 1897, an increase of 67,383 tons; 8,823 tons of copper precipitate were shipped against 8,906 tons in the preceding year. The net profits were £349,964, from which dividends amounting to £343,750 or 27·5% were declared, leaving a balance, with that carried from the preceding year of £20,379.

South Africa.—The report of the Cape Copper Co., Ltd., shows a very successful year's operation, in fact the best in the history of the company. The profits reached £331,985. This is due not only to the increased price of copper but to the improved results from the smelting works at Britain Ferry, the two accounting for £142,301. At the mines in Namaqualand for the year ending August 31, 1899, 26,598 tons were raised from Ookiep and 1,400 tons from Spectakel, while in the succeeding year there was produced 24,262 tons from the

first and 932 tons from the second mine, showing a diminution from the two mines of 2,336 tons and 468 tons respectively. The assays, however, were practically identical in the two years, being 18·5% on the ores of the first named mines and 32% on the ores of the latter. The cost per unit of copper was 1s. 7·02d. per unit against 1s. 6·53d. a unit in the previous year. The railway shows an increased profit, as does the Tilt Cove establishment, the latter increasing its profits £27,387 above the previous year. At one time these mines were debited with an outlay of £96,213.

Copper has been found at Kameels Poort, some 45 miles N. E. of Pretoria, in a line of mineralized country extending from Mapoch's country across the Olifant's and Moos Rivers. There are also extensive workings of an unknown age in the vicinity of the Palabora in the Murchison range on the Transvaal. Apparently many thousand tons of ore have been mined. The matrix consists essentially of a highly crystalline calcarious to silicious rock containing magnetic iron ore associated with sulphides, oxides, silicates and carbonates of copper. It is thought that the Kaffirs worked these ores to obtain copper for ornaments.

Tasmania.—During the half year ending September 30, 1899, the Mount Lyell Mining & Railway Co. treated 137,414 tons of ore, 128,388 tons of which came from the open cuts, and 9,026 tons from underground. In addition 13,017 dry tons of purchased ore were treated, making the total amount of ore which passed through the furnace 150,431 dry tons. The ore averaged 3·38% copper, 3·03 oz. silver and 0·093 oz. gold per ton of 2,240 lb. The converter plant treated 9,192 tons of converter matte and produced 4,592 tons of blister copper, of an average assay value of copper 98·80%, silver 84·53 oz. and gold 2·971 oz. The total cost of producing this copper was $5·6135 per ton of ore of 2,000 lb. divided as follows: Mining, $0·5598; removal of overburden, $0·4584; smelting, $4·0959; converting, $0·4994. The net profits for the six months amounted to £182,596 3s. 5d. Dividends during this period were paid to the amount of £75,625. The smelting costs showed a slight increase above that of the preceding half year, but it is anticipated that these will be materially reduced when the company's coking plant at Port Kembla, New South Wales, is in operation. The North Mount Lyell Mining Co. has developed exceedingly well. Its smelting plant was almost completed at the end of the year and the railway likewise is nearly finished. The body of ore is said to have been proven to 330 ft. in depth and 90 ft. wide. The shaft on the South Mount Lyell was sunk to the depth of 447 ft., where a large body of ore of an average value of £4 5s. 6d. per ton was met with. This is believed to be a continuation of the Mount Lyell lode and has been developed to a depth relative to 650 ft. within the latter property. The value of the ore ranges from 3 to 5% copper.

Copper Market in 1899.

The improvement in the regular copper trade as well as the large orders given out by electrical companies and the very large ship-building commissions of various governments caused a most active and prosperous market during the year. The cheapness of money during the early part of the year also aided the speculative movement which brought prices to such a figure that they could

not be maintained. Though the inevitable reaction followed and prices were depressed to a point below the present quotations the constantly increasing demand restored prices to a figure at which it is anticipated it will be well held for a considerable length of time. Notwithstanding the great prosperity in the trade, the shortage of stocks and increased activity in mining, the production, which was forced at many existing mines, was but slightly above 10% greater than the previous year. In fact, it did not keep pace with the consumption. While many of the mines which have been organized into companies during the year will never become important producers, yet there are many legitimate enterprises whose product in the future may have some material bearing on the course of the market. However, the length of time necessary to develop a copper mine is so great that no pronounced effect may be felt for several years. This is true not only of this country but of others where copper is being produced. As long as consumption keeps abreast of or before production there is no reason to fear a sharp decline, and as the large existing producers have been worked practically to their full capacity it seems probable that high prices even if not so high as those maintained at present will rule for several years to come.

New York.—The year 1898 closed with a firm tendency and at practically the highest prices of that period, but it was not until early in 1899 that consumption became manifestly larger than production and stocks already small were drawn upon to such a degree that an upward movement in values took place. Whatever the future may bring forth, it may be stated confidently that consumption is likely to remain satisfactory, and that the prosperity of the last few years in this industry will continue for some time to come. Promoting has been stimulated greatly and for a time new propositions came out almost daily. It is unnecessary to say that a great number of them were worthless or nearly so, but that others possess great merits and will continue as regular producers even if a low level of values should be established permanently.

Exports to Europe have been irregular, but in the main satisfactory; during the last few months the somewhat curious movement has been experienced of Chile bars stored in English warehouses being shipped to the United States refineries for treatment by electrolysis, while the refined copper is to be re-exported to Europe. This demonstrates very clearly that our home refineries are worked to considerably better advantage than those in Europe.

Last December Lake copper closed at 13·25c. and electrolytic at 13@13·125c., but quite early in January a very active trade prevailed, and with simply enormous transactions during the entire month, prices advanced almost from day to day, until, by the end of the month, 16c. for Lake copper and 15·5 for electrolytic was firmly established; and the month of February did not remain far behind this phenomenal record, bringing the price up to 18c. for Lake and 17@17·25c. for electrolytic.

March then brought a slight reaction, and prices for the month fell off about 0·5c. But this retrogressive movement did not continue long, and April brought prices up to the highest levels established in January, especially values for

Lake copper, 18·5c. being freely paid, and in some instances sales were even made at 19c., while electrolytic copper ruled at 17·25@17·5c.

Toward the end of May a slight easing off again took place and continued throughout the month of June, causing a decline of 0·125@0·75c., but in July manufacturers found themselves compelled again to enter the market and purchase heavily. Besides, the Europeans, who had for some time refrained from buying to any extent in this market, were at last forced to replenish their depleted stocks.

Slowly, but steadily, values again advanced to 18·125c. for Lake and 17·125@17·25c. for electrolytic copper, which values were firmly maintained during the summer months, August and September showing hardly any change.

With the beginning of October matters changed. The outbreak of the war between England and the Transvaal unbalanced trade. Money rates advanced and the bank reserves became smaller and smaller, all of which affected trade in general unfavorably.

It will have been noticed that between Lake copper and electrolytic rather a wide margin had been established, while formerly the difference between the two descriptions rarely exceeded 0·25c. During this year, however, the margin widened to 1c. and even 1·25c., or a little more. The reasons for this are somewhat difficult to explain, except that at a certain time earlier in the year Lake was in specially large demand, while orders for electrolytic copper, which is now being produced in enormous quantities, were more eagerly competed for. Producers of Lake were evidently for some time under the impression that this difference could be maintained, and, waiting for the trade to approach them, they accumulated some stocks, while electrolytic copper was delivered to consumption as fast as produced.

The first pressure, therefore, came from the producers of Lake copper, and when it was found that the largest producer of this description was ready to sell a big quantity of copper, consumers became extremely shy, and negotiations were pending for some weeks before a large sale was consummated, which proved then to be a cut from 18·5c. down to 17c. for delivery up to the end of March, 1900, with the condition that the selling company guarantees its own price against a decline. This large reduction startled the trade and some of

AVERAGE MONTHLY PRICES OF LAKE COPPER IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April	May.	June.	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1895	10·00	10·00	9·75	9·75	10·25	10·63	11·25	12·00	12·25	12·00	11·00	10·50	10·76
1896	9·87	10·64	11·03	10·98	11·15	11·67	11·40	10·98	10·66	10·66	11·23	11·28	10·88
1897	11·75	11·92	11·80	11·48	11·03	11·11	11·11	11·16	11·30	11·13	10·85	10·78	11·29
1898	10·99	11·28	11·96	12·14	12·00	11·89	11·63	11·89	12·31	12·41	12·86	12·93	12·03
1899	14·75	18·00	17·54	18·43	18·25	17·93	18·33	18·50	18·46	17·76	16·98	16·40	17·61

the smaller Lake producers evidently were under the impression that they might be able to realize somewhat better prices, but in this they failed, and it was already in November that sales were made slightly below that figure. Though electrolytic copper did not decline in the same proportion, sales were made at 16·5c.

In the beginning of December matters became worse. Money rates advanced more and more, and at about the middle of the month financial centers experienced a panic more severe than at any time since 1893. Inflated prices of all industrials suffered tremendously and copper stocks shared the same fate. All this naturally reflected on the prices of the different metals, and some sales were made at 16·25c. for Lake and even at 16c., while of electrolytic copper sales were made at 15·5@15·875c.

London.—Statistics published above show the world's supply of copper to have been 424,126 tons on January 1st, an increase of over 26,000 tons on the previous year, but the stocks were rapidly diminishing and at the end of the first fortnight there was already a decrease in the visible supply of 1,318 tons.

The turnover in the London market was enormous, some 59,000 tons for the month of January. The speculative element was very strong, business exceedingly lively, and options were freely dealt in. The price of standard copper at the opening was £57 17s. 6d. cash and £58 5s. three months, but large purchases drove up the market steadily and in spite of some temporary reactions, brought about by realizations, a gain of £5 was recorded by the 20th. A prominent feature toward the close of the month was the covering of old bear sales, falling due in February, which produced a scarcity of cash copper and established a temporary backwardation. The closing figures were £69 16s. 3d. cash and £69 15s. three months—a remarkable advance on the opening figures. India and the East began by sending good inquiries and orders for yellow metal and copper sheets, but the high prices had the effect of practically stopping these.

February brought no abatement in the excitement. The high prices drew out some large selling orders which caused heavy fluctuations but did not stem the upward current, which received a new impetus from further excited purchases. When the high figure of £74 10s. cash had been reached, however, the market became top-heavy and collapsed about £4, but quickly recovered to £73, every decline at this time offering new temptation to buyers. The ultimate effect of such heavy fluctuations, however, was unsettling and business became less active, the month closing at £71 15s. cash and £71 11s. 3d. forward.

About this time efforts were being made to increase the production of copper, and makers of mining and engineering plants were reported to be extremely busy, but it was long before the effect of these efforts could be felt. Trade generally became quieter and manufacturers were withholding orders on account of the high prices. America was an eager buyer and even repurchased good quantities previously sold to Europe.

In the beginning of March the total visible supply of copper was the lowest on record, 24,326 tons, having decreased further by 1,500 tons during the latter half of February. The fluctuations in prices continued and were remarkably sharp compared with the extent of the turnover, bear covering on the one hand and large profit taking on the other, making the market very sensitive; £72 for three months and £72 8s. 6d. cash were the opening figures, but they were not maintained, and the downward tendency was assisted by an unexpected increase in the stocks of 4,171 tons for the month. When £66 7s. 6d. had been touched there was a sudden rush of buying orders, chiefly for American account,

and as much as £3 was gained in a single day, the close being firm at £70 for cash and three months. Business with consumers was restricted, the high prices discouraging new orders, while the demand for manufactured articles was practically *nil.*

Notwithstanding a further increase in the visible supply of 2,614 tons for the preceding fortnight, April opened with still higher prices, the large purchases by American operators being renewed with great energy. The bears also began to cover and prices rose from £70 10s. to £72 5s. at the first session. After a temporary reaction, caused by the publication of the statistics, a new demand set in and speculation became very excited. Some heavy buying of near dates caused a backwardation on forward copper which at one time reached 30s. Bear covering continued on a large scale, as did also the strong buying on behalf of the American clique, who were forcing up the market. The intentions of the latter party were somewhat criticised, and certain people, who doubtless had in mind the French combination of 10 years earlier, predicted a heavy fall. But, unlike the French party of 1889, this American movement had enormous funds behind it, and, what was more important, very good trade conditions. Toward the close of the month the record figure of £78 for standard copper was reached, in an exceedingly lively market. Rumors of an American combination of producers were again rife and caused considerable excitement in the copper share market, though they lacked confirmation.

May, however, brought the expected confirmation of these rumors, and the formation of the Amalgamated Copper Co., of New Jersey, with a capital of $75,000,000, was announced. It was said that American holders then controlled the whole available stock of standard copper. Business became restricted and prices fell off from £77 10s. to £75 10s. Three months copper was neglected and sold as low as £2 under cash metal, but this backwardation was soon reduced to 5s. About this time American and English prices were almost on a level, making possible some export business in American electrolytic copper, but there was little new speculative business and the turnover was small. The prices ruling at the close were about £76 10s. cash and £76 5s. forward. With the lull, consumers began to cover their requirements, but they were by no means reconciled to the high level of prices and bought only from hand to mouth. The statistical position helped to keep down the market, the visible supply being increased by 3,627 tons for the month. A decline in the copper share market also contributed to depress prices.

At the opening of June the commencement of trouble in the Transvaal, political agitation in Paris and a large increase in supplies during the latter half of May, had a depressing effect on the market. Little support came from America until the close of the month, which firmed the market back to £76 7s. 6d. cash with forward copper about 1s. 3d. less. Confidence of both consumers and speculators was renewed in July. Producers were sold out for some months ahead and began to hold out for higher values. A large demand arose in Germany and considerable demand was experienced in England and France. The tight money market prevented speculative business, however. This caused sales of cash warrants and a contango of 12s. 6d. on three months copper. The

market closed at £76 7s. 6d. cash and £76 17s. 6d. three months. The increase of shipments from Chile and the arrival of two large cargoes of copper caused prices to decline in the beginning of August, but the market, supported strongly by American operators, recovered from £75 11s. 3d. to which it had fallen back, to £76 5s. Large sales were made on the Continent even as far forward as December, and this coupled with the dearth in the supply of electrolytic copper straightened the market at the close of the month to £76 18s. 9d. cash and £77 3s. 9d. three months. Political troubles in the Transvaal were a serious factor in September, the depression in shares reacting to a certain degree upon the copper market. Great scarcity of electrolytic copper was felt, as the activity in the wire market was almost phenomenal. Chile bars were shipped in large quantities from England to the United States, the electrolytically refined product being returned. The closing prices at the end of the month, after experiencing a sudden drop of 20s. due to close money bringing out a plentiful supply of cash copper, were £75 11s. 3d. cash and £76 2s. 6d. forward. The shipments of Chile bars to America had its effect in October, as supplies were reduced 3,199 tons in the preceding fortnight, but as it was finally recognized that shipments to America simply meant a transfer to America of copper which sooner or later would find its way back, quotations actually fell 20s. The decline, however, was aided by weakness in the American market. The Lake companies suddenly reduced their prices, causing realizations on a great deal of weakly held copper. Quotations fell to £72 17s. 6d. cash, but by dealers supporting the market they rose to £74 12s. 6d. three months. Renewed selling caused a weaker market again and the month finally closed at £72 15s. cash and £73 at three months. Aided by weakness in the American market and the alarming situation in the Transvaal, prices receded to £72 cash, but strong support raised quotations up to £74 15s. The demand, however, was confined chiefly to cash warrants which were held firmly, with a result that a premium of nearly 20s. for this position was momentarily established. Statistics, however, at the middle of the month showed a decrease of 1,276 tons and a fresh element of vigor was introduced into the market, £75 2s. 6d. being reached with a large turnover. Inactivity followed with extreme scarcity of cash copper owing to the control which had been obtained by the Americans. Consumers purchased for immediate requirements only and the rise in the bank rate at the end of the month caused a precipitate decline to £73 12s. 6d. cash and £73 forward. In December a prominent metal merchant was forced to assign, and his large commitments in tin and copper caused a decline in values. The British reverses in South Africa were also a great factor, as was the continued stringency in the money market and the high bank rate. In the latter part of the month the German bank rate was raised to 7% and the French rate to 3·5%. There seemed to be a strong prospect that the Bank of England rate would rise to 7%, so that there was a rush to realize, causing a fall to £67 for cash. A recovery, however, was experienced at the end of the year when £70 was again established with a reassuring outlook.

Notes on the Metallurgy of Copper.

Copper Smelting in Mexico.—Otto H. Hahn* describes the practice at the Aguas Calientes smelters. The blast furnaces are of the top-feeder kind, and rest on iron standards. They have wrought-iron water-jackets with eight tuyere openings in each side, 24 in. above the furnace bottom, and bronze tap-jackets. The sides have no bosh, but slope toward the top in an unbroken line; the front and back are straight. The furnaces have no crucibles, but discharge their contents into huge, circular-shaped settlers of about 8 ft. diameter, from which the slag overflows into a stream of water, is granulated and washed down-hill, while the matte is tapped into large vessels or so-called ladles, resting in tilting frames, which are mounted on railroad trucks. The matte is hauled in its ladle over a track to a hydraulic elevator, which lifts it to the converter platform. The converter being in proper position, the matte is poured by means of a gutter attachment, down through the nose into the converter, and blown. It takes three fillings and skimmings before the required quantity of converted copper, running 98%, is obtained. Dip samples are taken from the metal in the converter while it is being emptied into moulds for ascertaining the assay value in copper, silver, and gold. The converter lining is made of a Pachuca gouge-ore, so-called "refractious"; an artificial mixture of pulverized quartz and clay, which had to be used in a pinch, does not stand so well as the former, and has given much trouble by scaling or flaking off. The converter slag is re-melted in the copper furnaces along with the charges. Its approximate composition is seen in the following analysis: SiO_2, 26·6%; Fe, 41·9%; CaO, 0·2%; S, 1·1%; Cu, 4·5%; Ag, 14·3 oz. per ton.

The principal source of copper ore for the copper furnaces are the mines of the firm at Tepezala. As the supply was insufficient, and as even with the aid of purchased ores not enough matte was produced to keep, at least, one converter going, matte had to be imported from Copper Queen Consolidated Co., in Arizona, and the Silver City Reduction Works, in New Mexico, U. S., to fill the deficiency. The heaviest shipper of custom copper ore has been the Panuco mine, near Candela, Coahuila; smaller quantities came from Avino, Durango; Mazapil, Zacatecas; Parral, Chihuahua; and other places, even from San Jose de Gracia, Sinaloa. The Michoacan Railway and Mining Co. of Angangueo shipped the largest part of the silver ore smelted in the copper furnaces.

The following analyses show the composition of the principal ores:

	SiO_2. %	Fe. %	CaO. %	S. %	Cu. %	Ag. Oz.	Au. Oz.
Tepezala	39·5	11·8	13·1	5·3	5·5	8·0	.....
Panuco, first-class	8·9	20·9	.....	27·3	35·7	5·0	0·16
Panuco, second-class	15·6	32·3	.....	35·3	10·8	1·6	0·09
Avino	64·4	3·7	.....	4·7	14·1	30·3	0·019
Parral	52·5	7·6	1·1	0·8	11·0	197·3	.....

The Angangueo ores ran as follows:

**Transactions of the Institution of Mining and Metallurgy*, 1900.

	SiO_2. %	Fe. %	CaO. %	Zn. %	S. %	Pb. %	Ag. Oz.
Carrillo, gabarro	31·7	23·8	2·0	4·7	25·7	1·3	40·0
Carrillo, tierras	26·6	24·1	1·3	5·0	24·3	0·8	26·9
Purisima, gabarro	25·0	25·3	0·8	4·0	27·3	1·4	26·8
Purisima tierras	17·5	26·2	0·6	6·0	27·3	1·2	23·3

The Angangueo roasted ore: SiO_2, 28·8%; Fe, 29·0%; CaO, 0·8%; S, 6·8%; Pb, 1·6%; Ag, 28·3 oz. per ton.

The composition of mattes purchased was as follows:

	SiO_2. %	Fe. %	S. %	Cu. %	Ag, per oz. Tons.	Au, per oz. Tons.
Descubridora, Durango	0·3	14·0	22·0	60·7	36·4	0·96
Copper Queen	1·0	28·5	23.0	41·5	5·4	0·11
Silver City	0·4	17·1	21·6	45·9	83·4	4·30

The copper furnaces labored under great disadvantages as the dust chamber and exhaust chimney were of inadequate dimensions, and in consequence the smoke issuing from the top of the furnace was even worse than that from the lead furnaces on account of the sulphur dioxide evolved. Copper furnaces are run with a hot top when sulphides are treated, owing to the blowing in of a large excess of air, to effect a partial roasting before the tuyeres, hence the sulphur dioxide. To check the smoke nuisance somewhat, the blast had to be cut off, which prevented the furnaces from doing their full duty, and it was not until after a new flue chimney of proper dimensions had been built that a satisfactory tonnage could be attained. Under the old conditions, the average tonnage per furnace was 110 tons of charge, comprising 75·5 tons of ore, and 34·5 tons of flux and secondaries. After the erection of a new chimney and flue, the average tonnage rose to 152·5 tons of material, composed of 120·7 tons of ore, and 31·8 tons of flux and secondaries. According to P. Johnson, the late superintendent, there were smelted in 1898:

Tepezala ore, 24,387 tons; purchased copper ore, 16,985 tons; crude Angangueo ore, 15,369 tons; roasted Angangueo ore, 13,345 tons; other silver ores, 5,928 tons; extraneous matte, 4,261 tons; total, 80,275 tons. First dust, 3,579 tons; furnace barrings, 1,753 tons; domestic matte, 16,835 tons; lime rock, 8,505 tons; total charge, 110,965 tons. Converter slag, 15,309 tons; lead blast furnace slag, 484 tons; total, 126,758 tons, with 10,653 tons of coke. The average composition of the slag, resulting from this work has been: SiO_2, 37·1%; FeO, 32·1= (25% Fe.); CaO, 17·1%; Cu, 0·3%; Ag, 0·6 oz. per ton; and that of the matte: Cu, 32·30%; Pb, 4·24%; Ag, 103·93 oz. per ton.

In the converter, 115,119 tons of matte were treated, yielding 4,623 tons of converter copper, which assayed 97·2% copper, 371·8 oz. silver, and 1·0976 oz. of gold per ton of 2,000 lb.

These figures do not quite agree with the books of the general office, which show that: 78,386,407 kg. ore smelted, yielded 4,840,532 kg. of copper, with 62,932·54 kg. silver, and 156·9479 kg. gold. The cost of smelting one ton of ore in the copper furnace has been $7·50, including $2·10 taxes. The cost of bessemerizing one ton of matte has been $17·80. Johnson admits of only 2% loss

in the converter, notwithstanding the presence of such volatile metals as lead and zinc in the same matte, but states there is a larger loss in the blast furnace, which is in part attributed to the constant hot top, and in part to the existence of chloride of silver in some of the ores.

The blast for the cupolas is furnished exclusively by Roots blowers Nos. 6 and 7, which are erected near or behind the furnaces; that for the converters, by a 250 h. p. horizontal Corliss blowing engine, with 24×48 in. steam cylinders, and 48×48 in. air cylinders, made by the E. P. Allis Co., of Milwaukee. The pressure of the air is between 18 and 20 lb. per sq. in. Under the same roof with this engine there is one compound Worthington duplex pump for pumping tanks, one Worthington duplex pump for a hydraulic coke elevator and fire service, and one Wilson-Snyder duplex pump for operating converters and hydraulic elevators.

Copper is used as collector of precious metals on a limited scale only by the Aguas Calientes and the Mazapil works, while the largest producer of copper in the Republic, the Compañia de Boleo smelts only copper ores; as does the Descubridora of Hornillas, Durango; the Magistral, Canton Victoria, Chihauhua; and the San Carlos Copper Co., of San Jose, Tamaulipas. With the exception of the Descubridora ore, these copper ores are oxidized, and range in grade from 3 to 6%. As the treatment of oxidized ores in the blast furnace involves considerable loss of copper in the slag, the Boleo Company, of Santa Rosalia, Lower California, uses gypsum to at least partially sulphurize the copper in the ore, pyrites not being available. The gypsum requires to be dehydrated before use, as it would otherwise derange the furnace by incrusting the top. The ores, being fine, are mixed with the dehydrated gypsum and coke-dust, and bricked in a machine, such as is used in Europe for briquetting brown coal breeze. It has a capacity of 3 to 4 tons of briquettes 3·5×1·75 in. in size, per hour, and costs 5,000 fr., f. o. b., Antwerp. The manufacturer is N. Fouquemberg, Wasmes, Belgium. The smelting of this material proceeds very smoothly, and gives great satisfaction.

It is to be expected that with the development of many new deposits in all parts of the Republic, copper will come into use for the extraction of silver and gold, especially as the lead ore deposits are strained to a rate of production which cannot be maintained indefinitely.

The David Selecteur.—The Société des Cuivres has resorted to a bottom tuyere converter—the invention of M. Paul David, its manager. The converter is bulbous shaped, vertical, and the tuyeres which perforate the bottom are arranged in a circle and pointed upward at such an angle that the blast impels the charge to rotate, and thus wash the lining uniformly in every part. The tuyeres can be punched, it is claimed, without standing under the converter, and the lining is acted on more evenly, and its mechanical destruction is less liable to occur than when the tuyeres are on one side of the shell, and the blast impinges on a corresponding zone on the other side. As a consequence, the lining is said to last longer. It is only dissolved, not wasted, and carried off in unaltered particles in the slag, as always occurs to some extent in hand-packed lining. The slag is also more basic. The most original feature of the *selecteur* is, however, a pocket

on one side about equally distant from the bottom and the mouth of the converter, into which the first copper formed, which carries all the gold and most of the impurities, can be collected, and from which it can be tapped—whence the name *selecteur*. M. Burthe in his description of the *selecteur* gives what he claims to be actual figures of the cost of converting a 35% matte into copper at Aiguilles Per metric ton the cost was as follows: General labor, including foreman, $0·90; lining, including labor, clay and quartz, $0·834; blowing, including labor, fuel, and sundries, $1·494; repairs and sundries, $1·110; a total of $4·338. He also gives the cost of refining by the old style converters as $8·106 at the same place. Common labor cost only 70c. per day; skilled laborers at lining were paid 80c., and those at blowing, $1. The foreman received $1.*

Progress in the Electrolytic Refining of Copper during 1899.

By Titus Ulke.

Electrolytic copper refining in the United States during 1899 was characterized by the remarkably large output of fine copper and precious metals, as shown by the subjoined table, and the reduced cost of handling the material, both to and from the reverberatories and the tanks. Several of the labor-saving devices used by the progressive iron and steel metallurgist were adapted to the needs of the copper refiner. The form of the anodes and their manner of suspension in the tanks were modified in several refineries, producing an increased efficiency; methods of analysis were simplified and other improvements made to which attention will be called in the following pages.

Charging and Casting Machines.—Two new modifications have been successfully developed at the works of the Raritan Copper Co., Perth Amboy, N. J. One consisted in the use of a Wellman-Seaver charging machine similar to those used in steel works for the open-hearth furnace for charging the reverberatory furnaces which are ranged in a battery and served from the front side by the machine running on rails. The other is a casting arrangement for making anodes designed under the direction of Manager McCoy, which is simply described as follows: The molten copper is tapped from the furnace into ladles suspended from jib cranes and then carried to position and poured into the copper moulds attached to an endless conveyor. In the rear of the battery furnaces the Wellman-Seaver electric traveling crane runs on ways above the jib cranes.

Peirce's semi-mechanical and Walker's improved casting apparatus which were described in The Mineral Industry, Vol. VII., are operating satisfactorily at the Baltimore Copper Co.'s works. At the Guggenheim refinery, Perth Amboy, N. J., the old 75-ton anode furnace is now equipped with Walker's casting arrangement, by which two to three anodes are cast per minute, and a new furnace, now in process of erection, is to be supplied with similar apparatus.

Improved Cathode Rack.—The tankroom of the Raritan Copper works contains several Wellman-Seaver cranes, each carrying an arrangement of hooks for lifting and handling a tank-load of cathodes at one time. A similar device

* James Douglas, Jr., "The Treatment of Copper Mattes in the Bessemer Converter," a paper read before the Institution of Mining and Metallurgy, Oct. 25, 1899.

ELECTROLYTIC COPPER REFINERIES IN THE UNITED STATES

(In operation November 20, 1899.)

No.	Name of Company and Location of Works.	Kind of Material Chiefly Treated.	Daily Output of Electrolytic Copper in Tons (2,000 lb.)	Number and Capacity of Generators. Kw. = Kilowatts.	No. of Tanks in Refinery.	Arrangement of Electrodes.	Daily Output of Gold and Silver from Slimes.
1..	Anaconda Mining Co., Anaconda, Mont.	Anaconda anodes.	100 (Capac. 135.)	6 W'hse. @ 270 Kw 2 W'hse. @ 300 Kw 1 W'hse. @ 220 Kw	1,430	Multiple.	8,000 oz. Ag. 35 oz. Au.
2..	Baltimore Smelting and Rolling Co., Baltimore, Md.	Anaconda anodes.	80 (Capac. 100.)	8 Edison @ 80 Kw 3 W'hse. @ 80 Kw	540	L. p. series. S. p. multi.	6,400 oz. Ag. 28 oz. Au.
3..	Raritan Copper Works (Lewisohn Bros.), Perth Amboy, N. J.	United Verde, B. & M., and Mt. Lyell copper.	80 (Cap. d'ble.)	3 Westinghouse @ 600 Kw.	800	Multiple.	4,800 oz. Ag. 142 oz. Au.
4..	Boston & Montana Cons. Copper and Silver Mining Co., Great Falls, Mont.	Boston and Montana anodes.	60	2 Westinghouse @ 600 Kw.	600	Multiple.	3,960 oz. Ag. 9 oz. Au.
5..	Nichols Chemical Co., Laurel Hill, N. Y.	Copper from pyrites, cinders, Mt. Lyell and Mountain Copper Co.'s anodes.	60	4 General Electric @ 200 Kw.	400	Series.	6,000 oz. Ag. 34 oz. Au.
6..	Guggenheim Smelting Co., Perth Amboy, N. J.	Mexican and Colorado copper anodes.	50	3 General Electric @ 180 Kw.	390	Multiple.	20,000 oz. Ag. 175 oz. Au.
7..	New England Electrolytic Copper Co., Central Falls, R. I.	Hanauer copper, B. & M. and Omaha and Grant anodes.	40	7 Gen. Elec. and Edison, aggregating 750 Kw.	380	Multiple.	3,800 oz. Ag. 20 oz. Au.
8..	Balbach Smelting and Refining Co., Newark, N. J.	Orford anodes and miscellaneous.	30	8 Excelsior @ 75 Kw.	432	Multiple.	2,500 oz. Ag. 13 oz. Au.
9..	Bridgeport Copper Co., Bridgeport, Conn.	Parrot anodes.	30	3 General Electric @ 200 Kw.	432	Multiple.	1,800 oz. Ag. 6 oz. Au.
10..	Irvington Smelting and Refining Co., Irvington, N. J.	Miscellaneous.	9	2 Hochhausen @ 80 Kw.	90	Multiple.	720 oz. Ag. 2 oz. Au.
11..	Chicago Copper Refining Co., Blue Island, Ill.	Copper from by-products & misc. pig-cop'r.	5	2 Edison @ 64 Kw.	200	Multiple.	100 oz. Ag. 2·5 oz. Au.

Estimated total daily production of electrolytic copper, 544 tons, or about 198,600 tons annually.
Estimated total gold and silver obtained in refining copper annually: 170,273 oz. gold; 21,199,200 oz. silver.

was made a few years ago by the Morgan Engineering Co. for the Boston and Montana refinery at Great Falls, Mont.

Tilting versus Stationary Furnaces.—Tilting furnaces for melting-down reverberatory or converter copper in making anodes are not considered by several refiners as satisfactory as the large stationary furnaces now being built, and in one case, at least, have been replaced by the latter.

Improved Form of Anode and Method of Suspension.—Anodes of the shape and dimensions shown in Fig. 1 are now cast regularly by one of our largest refining companies. These anodes are preferred to the ordinary multiple anodes, as they are more apt to hang vertically in the tanks and thus reduce the danger of short-circuiting when set closely. In addition they have the advantage of leaving less scrap to be re-treated. They are cast in iron moulds in preference to copper moulds, as the former warps to a lesser extent.

Arrangement of Electrodes and Tank Details.—Where great care is taken to have the electrodes even and suspended parallel in the tanks, as at the Guggenheim works, the distance between them can safely be reduced from the

general space of 1·5 or 2 in. to about 0·75 in. In this way the number of plates and the consequent output per tank are correspondingly increased.

Fig. 2, in which the distance between the plates is purposely exaggerated, shows a convenient arrangement of the plates, conductors and insulators, and a good way of distributing the current.

Each tank contains 27 anodes and 28 cathodes arranged in multiple circuit, and all the tanks are coupled in series. In one of the most recently erected refineries it has been found advantageous to set the tanks about 2 ft. lower than usual, thus securing more space above the tanks for handling the electrodes. At the same works insulators are used consisting of wooden blocks, grooved to fit over the tops of the conductors; these are convenient and effective.

To prevent loss of current between tanks, the solution either passes from a tank through a lead spout into a porcelain receiver terminating in a lead pipe leading into the following tank, or it flows under and inside of a lead partition

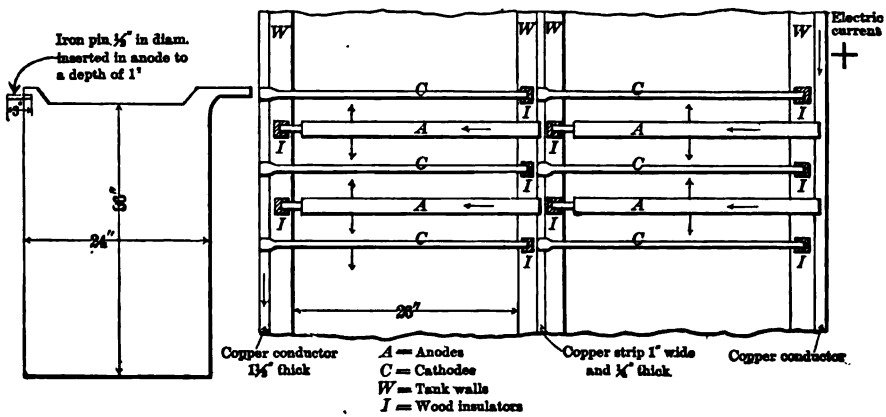


FIG. 1.—IMPROVED SHAPE OF ANODE.

FIG. 2.—TOP VIEW OF A PORTION OF TWO ADJOINING TANKS, SHOWING THE ARRANGEMENT OF ELECTRODES AND DISTRIBUTION OF THE CURRENT.

extending to within 6 in. of the tank bottom and is passed through a rubber tube, serving as an insulator, to the following tank.

Operations in the Tank Room.—It is now usual to empty and recharge the tanks with cathode sheets every 14 days; that is, each cathode sheet remains in a tank this length of time and receives a deposit of about 150 lb. of copper before being replaced by a new cathode sheet, whereas the anodes are changed every 28 days. Two double rows of commercial tanks may thus be emptied and recharged every day, the slimes being taken out every two weeks. The weight of anode scrap obtained is generally about 15% of the total anode weight charged. Instead of cleaning the anodes in special washing tanks placed at the ends of the tank rows, it is now preferred to wash the anodes in the commercial tanks after removing the cathodes and pumping or siphoning off the electrolyte.

The composition and temperature of the electrolyte is of great importance and in a typical Eastern refinery the temperature of the solution is maintained as near 200°F. as possible. The acidity is kept at about 9·25 to 9·50% free

sulphuric acid and about 13% bluestone. If the bluestone content fall below 11%, inferior copper is apt to be deposited.

Treatment of Saturated or Impure Solutions.—When refinery solutions become so saturated with copper that there is danger of crystallization, it is well to take out part of the copper or lead anodes, and, after concentrating the balance of the solution by boiling, to crystallize out the bulk of the copper as blue vitriol. At one of our largest refineries approximately 1% of the total copper produced is thus taken out as bluestone.

The impure mother liquor remaining after two or three crops of bluestone crystals have been secured, is usually run into tanks in which scrap iron has been placed so as to obtain cement copper. The latter is quite impure and is preferably roasted in a reverberatory furnace to drive off as much of the arsenic and other impurities as possible; it is then dissolved by sulphuric acid and steam, and the resultant copper solution is returned to the vitriol crystallizing vats. In a typical 50-ton refinery about 5 tons of blue vitriol are made daily, but only about 2·5 tons are shipped, it being usually necessary to redissolve and recrystallize the balance to meet the exacting demands of the trade.

Present Commercial Methods of Copper Assaying and Analysis.—These were described by the writer in the *Engineering and Mining Journal,* December 16, 1899, as far as they relate to refinery solutions, bullion, anodes, cathodes and slags.

With regard to the rapid method of determining arsenic, F. D. Aller claims to have originated (1) the method of dissolving one portion of the charge by the other portion, using the second portion to reduce the first to the proper state for the arsenic distillation, and (2) the colorimetric determination of arsenious acid. Aller also claims originality in the use of $ZnCl_2$ with Cu_2Cl_2 in the described method of distilling antimony, although the practical application of the method is chiefly due to A. E. Knorr.

The following method of depositing silver electrolytically and separating it from copper is recommended by F. W. Kuster and H. von Steinwehr.* Silver is precipitated perfectly and in a weighable form, when the solution (volume about 150 c.c.), containing about 0·5 g. of silver, is mixed with 1 or 2 c.c. of nitric acid, sp. gr. 1·4, and 5 c.c. of alcohol, the electromotive force being kept constant at 1·35 to 1·38 volts for from 6 to 8 hours. The results obtained check within 0·05%.

Electrolysis of Doré Bars.—As the methods of reducing the silver mud obtained in copper refineries to doré bullion, and the electrolytic parting of the latter have been described in previous volumes of THE MINERAL INDUSTRY, it will suffice to add that the Moebius continuous process of silver refining has been abandoned at Perth Amboy, N. J., because of the difficulty, it is stated, of removing the deposited metal from the silver belts and on account of the frequent breakage of the latter and the difficulty of their repair.

The continuous or band process is now used in the United States only by the Globe Smelting and Refining Co., at Denver, Colo., and in the old Moebius plant of the Guggenheim Smelting and Refining Co., at Perth Amboy, N. J. The latter's new silver parting plant contains 120 tanks and a general electric

* *Zeitschrift Elektrochem,* 4, pp. 451-455, 1898.

generator of 62 kw. capacity. With 133 volts and 240 ampères nearly 70,000 oz. of silver are deposited daily, corresponding to two million ounces per month, and a monthly output of 3,000,000 oz. is contemplated for the future.

The doré anodes are about 5 in. wide and 1 ft. long and weigh from 100 to 115 oz. The amount of anode scrap produced is not more than 2 to 3% of the total weight of anodes charged. Each block of six tanks is about 13 ft. long, 25 in. wide (inside measurement) and 23 in. deep, and produces on an average 4,000 oz. of silver daily. To prevent the copper contained in the doré bullion, amounting at times to 3 to 5%, accumulating in the electrolyte to an injurious extent, 100 gal. are drawn off daily and replaced by freshly made copper free silver solution, the silver in the rejected solution being precipitated on copper plates in a special tank.

The gold mud is boiled in an annex with glass hood containing eight porcelain pots, each with a capacity of at least 300 oz. of gold mud. About 800 oz. of gold from 990 to 993 fine are secured per 24 hours.

The silver solution decanted from the gold is sent to the special tank containing copper anodes and the silver precipitated out of the solution by the copper.

Sources of Material for Refining.—Large quantities of copper ore in the shape of cupriferous iron pyrites are received from Spain annually and after being utilized in the manufacture of sulphuric acid, are concentrated by smelting into pig copper and electrolytically refined. In this way the small amounts of copper and silver contained in the Spanish ore are profitably extracted.

American copper refineries receive and treat also considerable quantities of gold and silver bearing copper from Mexico, Australia, Tasmania and Japan. Cheap transportation and the low cost of electrolytic refining in the United States—less than half a cent per pound—have made this possible. James Douglas cites an instance where matte was bought at full prices in Tennessee, transported by railroad to Norfolk, Va., trans-shipped thence to Tampico in Mexico, carried halfway across the Republic, saturated with gold and silver, concentrated into black copper, and then brought back by rail and water to a port in New Jersey for electrolytic treatment, and yet the operation was profitable.

Another curious and interesting case, noted during 1899, was that of Chili bars stored in English warehouses which were shipped to the United States refineries for treatment by electrolysis, and the refined copper therefrom re-exported to Europe.

Large exports of copper ore, matte and blister from this country to Europe have practically ceased, and instead, considerable quantities of ore, matte and bullion are coming here to be treated.

The above facts demonstrate that our refineries can successfully ccmpete with those abroad, occasionally even in their own markets.

Copper refining costs less here than in Europe, because our refineries are larger, better equipped and are worked to better advantage.

Hofmann's Method for the Manufacture of Blue Vitriol.

By Ottokar Hofmann.

If roasted copper matte is treated with diluted sulphuric acid the solution obtained will have, in addition to cupric sulphate, salts of iron, arsenic, antimony, bismuth, cobalt, nickel, etc. A solution of this character cannot be used for the manufacture of blue vitriol because these salts, to a certain extent will crystallize with the sulphate of copper and make it too impure for the market. The usual method employed by blue vitriol manufacturers is the cumbersome, slow and expensive Oker process, in which the copper matte is repeatedly roasted and smelted until it is brought to the condition of metallic copper. This product is granulated and alternately subjected to the action of air and warm, dilute sulphuric acid; the copper is oxidized by the air and dissolved by the acid and yields resultantly a copper sulphate solution which is crystallized into a reasonably pure product, and mud residues which contain silver, gold (if present) and impurities.

To overcome the disadvantages of the Oker process I have invented* a method by which the above-named impurities can be removed from a cupric sulphate solution in one operation.

This process has been introduced in the works of the Consolidated Kansas City Smelting and Refining Co., at Argentine, Kan., and has there been in successful operation for over two years; the average daily production amounting to about 30 tons of blue vitriol. A description of the process is as follows:

Crushing.—The copper matte is first pulverized to a fineness of at least 50 mesh to the linear inch; the finer this crushing is done, the better it will be for the subsequent operations. Matte is one of the most difficult materials to pulverize. It is rather tough, less brittle than ore, and causes great wear of the working parts of all ore crushing machinery with the single exception of the Krupp ball mill. In my long experience I have found no other pulverizing machine which has shown so little wear in proportion to its work and has required so little repair. The superiority of this machine is due more to the admirable material of which the wearing parts are made rather than to its construction. The Krupp ball mill which I use has a working capacity of 18 tons of matte per 24 hours, pulverized through a 50-mesh screen, and is in constant use day and night. The matte is charged in a coarse condition, 2 to 3-in. lumps. If it were finer size, 1 in. for instance, the capacity of the mill would be greatly increased. The perforated plates on which the pulverizing is done lasted three years, and the charge of 1,400 lb. of balls showed a wear of only 35 lb. in six months, which corresponds to 5% loss in a year. This record is excellent and it is strange that such an admirable pulverizer is not in more general use for dry crushing.

Roasting.—The pulverized matte of 34 to 40% copper content is roasted, and for the benefit of subsequent operations, the roasting is conducted with

* United States Patent No. 634,751, Oct. 10, 1899; English Patent No. 13,320, June 27, 1899; French Patent No. 290,336, Oct. 18, 1899; and Spanish Patent No. 24,429, Sept. 29, 1899. Application has been made for a Mexican patent.

great care, and with attention equally divided between the oxidation of the copper and of the iron. The copper is to be converted into oxide and sulphate by roasting, and the formation of cupreous oxide should be avoided, as one-half only of its copper will dissolve in diluted sulphuric acid while the other half will precipitate as metallic copper and will remain in the residues. The iron should be transformed into an insoluble oxide, otherwise too much iron will go in solution, and the precipitate which results in the process of refining the solution will become voluminous and will retard the filtration. It is not advisable to conduct the roasting so that as much copper as possible will be converted into sulphate, because the roasted material will then contain too much soluble iron. In order to obtain the desired result, the roasting must be done in a furnace with excellent draft, and under the conditions of excess air in the furnace.

At the beginning of the operation the temperature should not be raised above that produced by the combustion of the sulphur of the matte. During this period the charge assumes a rather bright red appearance, an effect due more to light than to heat, and if excess of air is admitted to the furnace but few lumps will form even with very leady copper matte. Gradually, as the oxidation advances, the surface of the charge becomes darker, especially near the air doors, and when the entire surface of the charge begins to darken, the fire is slightly increased to prevent cooling, as from this time on, the supply of heat furnished by the oxidation decreases rapidly. If this condition is overlooked and the charge cools too much, it will be difficult to raise the temperature again to the proper degree.

The roasting is continued at a moderate temperature until no more heat is evolved by the oxidation of the material, after which the temperature must be raised to a cherry red. A skilled roaster can readily determine this point by stirring the charge; if the particles thrown to the surface brighten, the roasting has not advanced far enough, but if the entire charge maintains a dead and uniform red color, it shows that this part of the roasting has been completed and that it is time for an increase of temperature. This can now be done without danger of lumping the charge, because by this time the greater part of the sulphides has been oxidized. The increase of temperature is necessary for two reasons: first, to hasten the oxidation of the remaining sulphides, which would require a very long time at a low temperature, and second, to convert the iron as much as possible into the red oxide, in which state it is not soluble in diluted sulphuric acid. Care must be taken, however, to avoid too high a temperature, as the cupric sulphate formed, which is very desirable to have in the roasted matte, would then be decomposed. At a comparatively early stage of the roasting nearly all the copper is in a state in which it can be extracted by diluted sulphuric acid; about 75% of it is present as oxide and 25% as sulphate. The roasting, however, cannot be considered complete at this stage, because quite a large percentage of the iron is soluble in acid as well and would make the resultant cupric sulphate solution unnecessarily impure. The task for the roaster is to convert as much iron as possible into the insoluble oxide, without the decompo-

sition of the cupric sulphate present. Cupric sulphate resists considerable heat and it is possible to conduct the roasting in this way; but this increase of temperature requires judicious care, because if the heat is too high the cupric sulphate will be reduced to cupreous oxide, in which condition but half of the copper is soluble in diluted sulphuric acid. Should cupreous oxide be formed the amount of extractable copper will be greatly reduced. If, for instance, the extractable copper before raising the temperature was 97%, after an excessive heat it will be reduced to 92 or, even, 90%. When the roasting is done in a common reverberatory furnace a mistake of this kind can be corrected by keeping the charge longer in the furnace and thus oxidize the cupreous to cupric oxide. In a mechanical roasting furnace, however, this cannot be done, but with experience and care the decomposition of the cupric sulphate can be avoided.

I use a double-story Pearce roasting furnace and obtain very satisfactory and uniform results, both with regard to copper and iron. This satisfactory working was not attained until an extra fire-place was built between the two last fire-places on the lower hearth. The space between these two fires in the original construction was too great and the matte which arrived at the second fire-place had lost the larger part of its sulphur and did not have sufficient left to furnish the heat necessary for the maintenance of the required temperature. It was cooled below a red heat and the last fire, though kept very strong, was not sufficient to complete the roasting. The charge became hot in the immediate neighborhood of the last fire, but as this is close to the discharge, the matte was not exposed long enough to this heat for the desired effect. After the addition of the extra fire-place it was easy to regulate the roasting operation and to obtain good uniform results. The present method of roasting with the altered furnace is as follows: A moderate fire is kept on the upper hearth close to the drop slot through which the matte falls to the hearth below. The other fire-place is used only at the start of the roasting, and when the charge is ignited the fire is discontinued so as to prevent it from becoming overheated. The remaining fire on the upper hearth is so regulated that the charge is dark red when it is dropped to the lower hearth. By far the greater part of the sulphur has to be oxidized on the upper hearth, otherwise too much work will be left for the hearth below, and it will be difficult to regulate the finishing part of the roasting properly. For this reason it is necessary to have an excess of air on the upper hearth, and all the doors with the exception of the one next to the feed should be kept open.

A free access of air is of such great importance in the roasting process, whether oxidizing or chloridizing, that particular attention should be paid to this requirement in the constructing of a roasting plant, and care should be taken to provide the furnaces with sufficient draft. In fact an excess of draft should exist, so that the operator may regulate it by the use of the damper. In most plants dampers are useless on account of a lack of draft even when they are opened wide.

It is not possible to avoid the formation of some lumps, especially in roasting leady matte, but if the roasting is conducted properly, these will be

small, soft, porous, and of well-roasted material. Roasted matte is always of coarser grain than the raw pulp, and for this reason, as well as on account of the lumps, it is necessary to pulverize the roasted material before treatment with sulphuric acid. The pulverized material should pass through a screen not coarser than 50 mesh to the linear inch.

Solution.—It is not practicable to dissolve the matte by filtration in tanks. The best method for the solution of the cupric oxide of the matte is accomplished in agitating tanks, and, unless the matte is in motion when the water or diluted sulphuric acid is applied, it will harden to such an extent that it will be necessary to use picks for its removal. This hardening greatly prevents free filtration and impairs the extraction. In addition to this objectionable feature there are others quite as bad, as the uneven concentration of the resulting solution, etc. If the dissolving is done in agitating tanks the process is under perfect control, and the extraction is accomplished in much less time and in a more thorough manner.

The agitating tanks are filled about two-thirds full with water, the agitator set in motion and sulphuric acid added until the liquid is about 3% acid strength. The matte is then charged gradually, and at the same time a stream of acid is allowed to flow in so as to maintain the acid strength during a portion of the charging. In this way the dissolving is accomplished with an acid strength of 3% or less, and still yields a strong solution of cupric sulphate. It is preferable to work with weak acid, because much less iron and other impurities will be dissolved than with stronger acid. When the solution has attained a certain strength the flow of sulphuric acid is stopped, and matte only is charged until the solution is neutral. Toward the end of the operation it is advisable to charge the matte at intervals, and to make frequent acid tests to avoid an excess of matte. The addition of sulphuric acid to the water, and the matte to the diluted acid produces heat, which aids the solution of the cupric oxide. This heat is not sufficient and the temperature is further raised by a jet of steam. It is well to interrupt the charging of matte while neutralizing as soon as the solution shows 1% free acid. If the agitation is then continued this 1% of acid will be diminished, but if in half an hour after the last acid test the percentage of free acid remains the same, more matte is added, until the solution is neutral; in this way a mistake of adding a large excess of matte is avoided. When the solution is neutral, the pulp (solution and residues) is drawn through an outlet near the bottom into a large cast-iron pressure tank, and from thence by means of compressed air it is forced into a large filter press to separate the residues from the liquid. The cylindrical side of the pressure tank is lined with sheet lead, which is protected against mechanical wear by a wooden lining. The spherical bottom is lined with a heavy hard-lead casting.

It is not advisable to make the cupric sulphate solution too concentrated, as it will partly crystallize when cooled moderately. These crystals will incrust the filter cloth of the press, blockade the pipes, incrust the sides and bottom of the receiving tanks, and interfere with the manipulation. A concentration when warm of about 24 to 26° B. gives good results.

The plates and frames of the press are made of wood and are 4×4 ft. in size. The crude solution from the filter press contains, in addition to cupric sulphate, compounds of iron, arsenic, antimony, bismuth, cobalt, nickel, etc., and is conveyed to the receiving tanks for further treatment. No trace of silver can be found in this solution, as whatever silver is converted into sulphate during the roasting is precipitated in the agitating tank by the ferrous sulphate present in the solution. The residues are well washed in the press, to remove

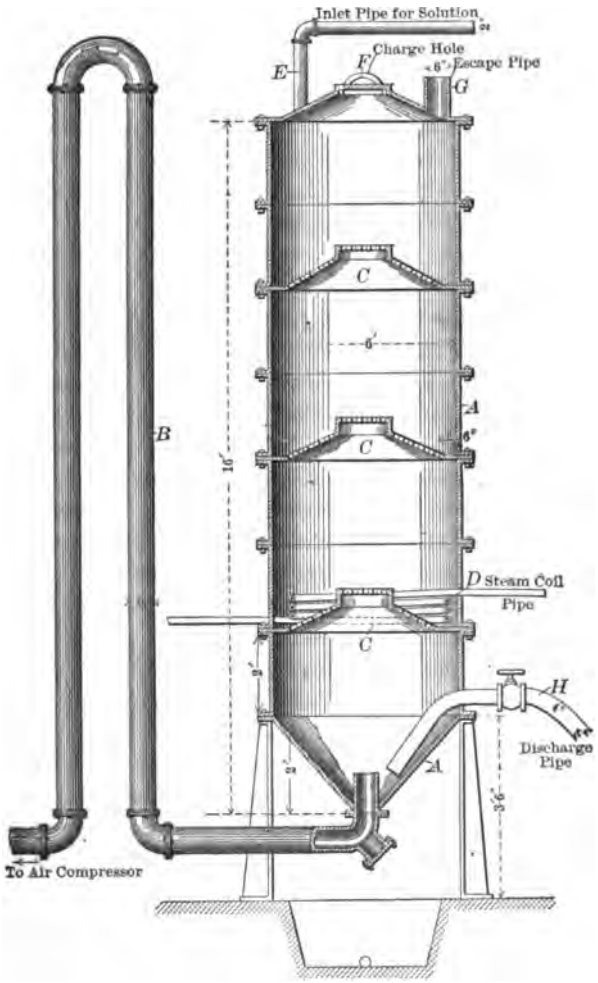


FIG. 1.—HOFMANN'S TOWER FOR REFINING CUPRIC SULPHATE SOLUTIONS.

all copper solution. The wash-water solution, which contains about 1% copper, is collected in special tanks and used instead of water for the preparation of a new charge in the agitating tanks.

The residues contain all the silver, gold and lead of the original matte; they are of a dark red color and consist mainly of iron oxide. This material is very desirable for the blast furnace, and is sent to that department for the ex-

traction of the precious metals and the lead. A large portion of lead is converted into sulphate by the roasting and therefore does not act as a consumer of acid. As the roasted matte contains 20 to 25% of its copper as sulphate the total consumption of acid is much less than the equivalent amount of acid in the blue vitriol produced.

Refining of the Solution.—The crude, neutral cupric sulphate solution from the receiving tanks is forced by means of a special cast-iron and lead-lined pressure tank into high tanks or towers. These towers are about 16 ft. high, 6 ft. in diameter and are of cast-iron, lined with lead (20 lb. per sq. ft.), and constructed as shown in Fig. 1. Through the funnel-shaped bottom, *A* enters the lead pipe, *B*, which is connected with an air compressor. The pipe rises above the top of the tower and then descends to the air compressor. This is done to insure the exclusion of the copper solution from the air pump. The ascending part only of the pipe need be of lead, the balance may be of iron. Between every other section of the tower is inserted a perforated cone, *C*, made of hard lead and provided in the center with a manhole and a lid well fastened to the cone. The perforations toward the periphery are larger than those nearer the center. Just above the lowest cone a lead steam coil, *D*, is inserted. *E* is the inlet pipe for the solution; *F* is the charge hole, and *G* the escape pipe for air and vapors. The pipe, *H*, serves as a discharge pipe after the solution is refined.

When the tower is filled, steam is allowed to enter the coil and heat the solution, and at the same time air is forced in through pipe *B*. The ascending air strikes the perforated cones, is divided in numerous streams and imparts a violent boiling motion to the liquor. The cones help to retain the air in the solution. Part of the iron is precipitated as a basic salt, by the action of the air, but I never succeeded in the precipitation of more than one-half, although the treatment was extended many hours. When the solution is hot (75 to 80° C.) the roasted matte is added through the charge hole, *F*. The violent boiling motion of the solution keeps the matte in suspension, and after three to four hours the solution will be entirely free from iron, arsenic, antimony, bismuth, etc. To observe and to regulate the progress of the operation the solution is tested from time to time for iron by taking samples through a small cock inserted in the side of the tower. It is not necessary to test the solution for other impurities, because the iron predominates, and by the time all of it has been precipitated, no trace of the others will be found.

The chemical reaction of this process may be expressed by the following equation:

$$2FeSO_4+O+2CuO=Fe_2O_3+2CuSO_4.$$

This shows us that the cupric oxide, which together with air, is used to precipitate the iron, combines with the sulphuric acid of the ferrous sulphate and goes in solution as cupric sulphate; a decided advantage, as the precipitant is converted into cupric sulphate, and thus enriches the solution in copper.

A tower charge consists approximately of 3,000 gal. of solution. When the reaction is completed, after three to four hours, the refined liquor, the precipitate, and the residues of the added matte is discharged through the pipe.

H, into an agitating tank at a lower level. From thence it is drawn to a pressure tank and forced through a filter press. The clear and refined solution from this press is conveyed to the evaporating tanks and concentrated by evaporation. The residues which consist principally of the precipitated impurities contain some undecomposed matte and some basic sulphate of copper; for this reason they are subjected to a short treatment with very dilute cold acid. The unwashed residues are removed from the press and charged into an agitating tank which contains a cold acid solution of 2·5 to 3% strength. This acid readily dissolves the basic sulphate of copper, while it reacts but slightly on the iron and not at all on the other impurities. The liquor of this charge passes to the crude solution and the residues to the matte residues. The refined solution is concentrated to the desired degree and then conveyed to special tanks for crystallization.

The solution, now freed from impurities, yields very pure crystals of blue vitriol of a brilliant color. Unlike the blue vitriol which is found in the market, the crystals obtained by this process do not change their physical condition by exposure to light and air, even in the course of years. They retain their deep blue color, remain hard and solid, and are affected by the direct sunlight only. The permanency of these crystals is caused by their production from a neutral solution.

This method of refining the cupric sulphate solution yields a superior product of blue vitriol by the direct treatment of roasted matte with sulphuric acid, an advantage hitherto not possible, as no practical method existed for the removal of the injurious impurities from the solution. In the Oker process the copper of the matte must first be converted into metallic copper by repeated roasting and smelting, before it is subjected to the alternate action of sulphuric acid and air. The process is slow and costly and requires an extensive plant even for a moderate production. The crystallization is done from an acid solution, and as a consequence the crystals easily deteriorate. All the sulphuric acid contained in the blue vitriol has to be furnished, while by a direct treatment of the matte about 25% of the required acid is formed in roasting.

Not only for the manufacture of blue vitriol does my method of refining cupric sulphate solution offer advantages, but in the electrolytic copper process is found a larger field for its utility. Instead of making copper anodes, the cupric sulphate solution is prepared direct from the matte, then refined and subjected to the electrolytic treatment. In this way all the lead of the matte remains in the residues, which form an excellent smelting material. When anodes are made, a large part of the lead is lost, as it is desirable to have them as high as 93 to 95% copper, and these can only be produced from a leady matte by a partial furnace refining of the copper, in which most of the lead is lost. The silver and gold contained in the anodes will settle as slimes to the bottom of the tanks together with the impurities, arsenic, antimony, lead, etc., and small pieces of copper which drop from time to time from the dissolving anodes. The removal and treatment of these valuable slimes are connected with considerable trouble and require great care. If we contrast

these conditions with my process, in which the solution is made direct from the matte and refined before subjecting it to the electrolytic treatment, we see that the precious metals, with the lead and impurities originally contained in the solution, will remain in the residues, from which the metallic contents can be easily extracted by smelting.

As the electrolyte is purer than in the ordinary electrolytic process a much stronger current can be used without impairing the excellent quality of the copper produced, and the copper content may be as high as 8%. In proportion to the amount of copper deposited, sulphuric acid is set free, and for this reason the resistance of the electrolyte is not noticeably increased when the latter becomes impoverished in copper.

When the copper of the electrolyte is reduced to about 1 to 1·5%, the current is interrupted, and the acid withdrawn for use in the preparation of fresh copper solution. By the continuous introduction of the solution into the first tank and its subsequent flow through the succeeding ones, as is the usual practice in electrolytic work for the gravity circulation of the electrolyte, and by the use of a sufficient number of tanks, the process can be so regulated that the acid will flow continuously from the last tank, and thus avoid the delay caused by discharging and filling.

Another important feature of this method is the fact that the precious metals are bound up in the process for a very short time only. As soon as the copper is extracted from the matte the residues can be smelted at once and the precious metals quickly obtained, while in the case of the ordinary treatment of the anodes, the gold and silver are bound up for a long time and consequently a much larger capital is required, the interest on which is lost.

With regard to the cost, it is undoubtedly cheaper to produce a cupric sulphate solution by treating roasted matte with sulphuric acid (especially as the acid is always returned by the process) than to transform the matte into suitable anodes. This is still more pronounced when the matte contains much lead. The cost of the electrolytic treatment may be the same in both cases, although I believe it will be somewhat less in the treatment of refined solutions.

I have experimented to a moderate degree with very good results, and a plant on a working scale is in course of erection.

Extraction of Copper from Burnt Pyrites by the Henderson Process.

By Joel G. Clemmer.

The cinder from pyrites used in making sulphuric acid contains too low a percentage of copper to admit of profitable smelting direct to "coarse metal," so that the wet method has to be resorted to. Still less could the raw pyrites of this class be utilized for direct smelting for copper, the tenor of the raw ore seldom exceeding 4% Cu. On the other hand the wet method is not applicable to pyrites carrying a high copper content. It is by the combination of processes which recover three main products (or one main and two important by-products)—sulphuric acid, copper, and a high-grade iron ore free from phosphorus and very low in sulphur—that these lean ores can be profitably handled. The recovery of silver and gold (if any) and the manufacture of chemical by-products also help to round out the whole series of operations and reduce costs.

The recovery of copper from mine waters and weathering ore dumps, the sulphide having oxidized to the soluble sulphate which can be easily precipitated on scrap iron as metallic ("cement") copper, has been practiced for centuries. The existing practice of roasting with salt to form copper chloride dates from about 1865, though the principles underlying it were laid down 25 years earlier, patents having been taken out by Hunt in 1840, and by Longmaid (with whose name the system is usually connected) in 1842. The latter indeed developed a very complete cycle of operations, having chlorine and salt cake as the principal products and copper as a by-product. Since then a number of modifications have been proposed, patented and tried, and some useful though minor improvements have been adopted. The process practiced at the present time is known commonly as the Henderson process.

During the '50's and early '60's many works were employing wet methods more or less similar to that of the present, but the date given above (1865) is that stated by Dr. Lunge as marking the beginning of the epoch of the universal adoption of the wet method of extracting copper from these lean ores, or rather, their cinders. It is not intended here to review all the successive stages of progress or of experiment in the history of the industry, but only to describe the practice at the works of the Pennsylvania Salt Manufacturing Co., at Natrona, Allegheny County, Pa., using Spanish pyrites containing 2% and less of copper.*

The burnt cinder is roasted with a proper proportion of common salt, with the view of producing copper chlorides, which are leached out with water, the residue, consisting chiefly of ferric oxide, being available as an iron ore. The copper is precipitated from the solution by means of metallic iron, while there are special methods for the recovery of the gold and silver, which differ at various works. Although the process is so simple in its outline there are many details which require careful attention in practice, and a proper design of the works and apparatus employed may be considered essential to success at

* Reference should be made to a paper by T. Ulke on the "Present Practice in Copper Concentration and Extraction," in The Mineral Industry, Vol. II.; and to *Sulphuric Acid and Alkali*, by G. Lunge, Second Edition, Vol. I., pp. 784-833.

the present time. In carrying out the process care must be taken especially to see that every charge of cinder and salt has been properly furnaced, sampled and tested before withdrawing from the furnace. Care must also be exercised that practically all the gas from the furnaced material is condensed in the towers, because all escaping gas not only contains hydrochloric acid necessary for lixiviation, but volatilized copper salts soluble in water.

With respect to the construction of the towers and furnaces, the reader is referred to Lunge's *Sulphuric Acid and Alkali.* The side elevation and section of a muffle furnace given herewith is one used by the Pennsylvania Salt Manu-

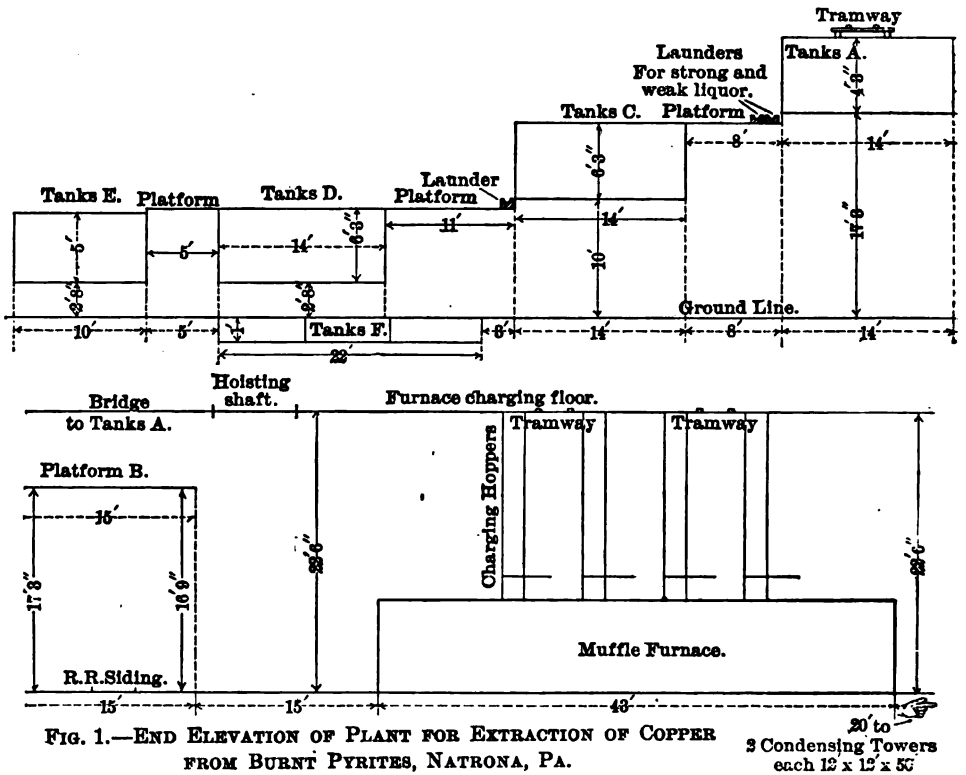


FIG. 1.—END ELEVATION OF PLANT FOR EXTRACTION OF COPPER FROM BURNT PYRITES, NATRONA, PA.

This drawing is divided into two sections. The tanks *E* and *D* rest directly on the foundations. The tanks *C* and *A* and the platform *B* are supported on trestle work.

facturing Co. It serves admirably the purpose of calcining burnt pyrites. The furnace is provided with four charging hoppers, or round sheet-iron pipes 20 in. in diameter, which terminate in cast-iron boxes directly over the holes in the top of the furnace. The boxes are provided with dampers to drop the material into the furnace, and to prevent the escape of gas at those points. The other ends of the charging pipes are open and terminate in and are flush with the charging floor above the furnaces.

The method of operating the plant is as follows: The cinders are ground dry in pan mills to about 20 mesh size, and mixed during the operation with about

10% of salt testing high in NaCl. The mixture is then lifted by bucket elevators to overhead hopper bins, from which it is drawn into tramcars, weighed and dumped into the furnace hoppers, or pipes, set in the floor over the furnaces. Should the unburnt sulphur in the cinders not be equal to the copper contents, sufficient "green ore" (pyrites fines) must be added in the mill to bring it up to, but not to exceed, 1·5 times the copper content. Then 9,600 lb. of mixture is taken as a furnace charge, and is heated to a very dull red heat, say about 800° F., and well stirred. When properly worked such a charge will be finished in about eight hours, making three charges each 24 hours per furnace. The charge when finished is drawn from the furnace to the floor to cool, and then transported in tramcars to the lixiviating tanks *A*, shown in the accompanying engravings.

For the success of the process much depends on the proper construction of

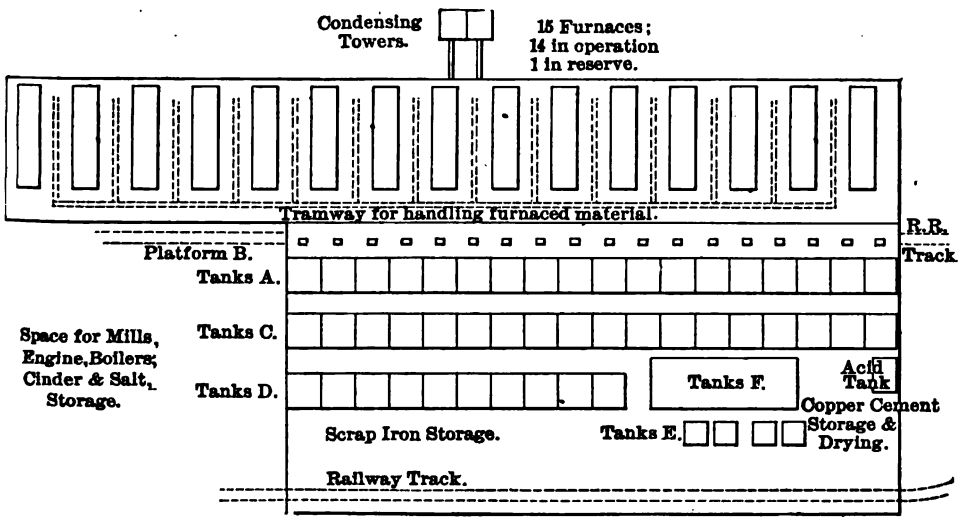


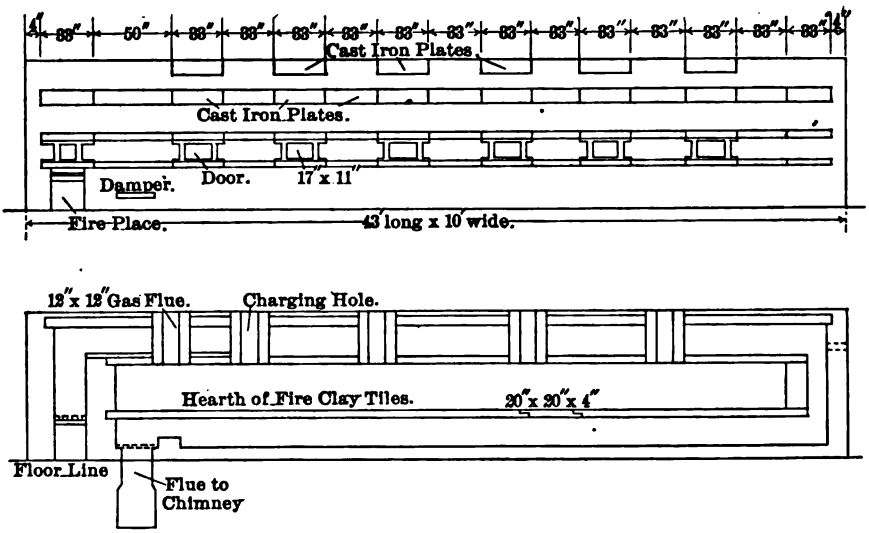
FIG. 2.—GENERAL PLAN OF PLANT FOR EXTRACTION OF COPPER FROM BURNT PYRITES, NATRONA, PA.

Scale, ¼ in. = 20 ft.

the leaching department. Means must be provided to recover any leakage or accidental overflow of the liquors from the tanks, and if this is not done much value may be lost in the ground. To prevent this result the earth in the area the tanks are to occupy must be dug up and filled with broken stone and cemented to the depth of 2 ft. or more, according to the nature of the ground. The surface of the concrete should have a sufficient slope toward all sides, so that any liquor escaping from the tanks will flow to a common sump, or cemented and tarred catch pits, of convenient dimensions. Or two or more such catch pits may be provided if desired. It is customary to top the concrete with a layer of common hard-burnt red brick set upon edge or flat, the joints and upper surface of the brick being filled and covered with hard tar. Such a floor may also serve as foundation for the supporting timbers of the tanks, though it is best to set them upon brick foundations built on the concrete

(before the floor bricks are laid) with hard red brick laid in cement and tarred. In the latter case the supporting timbers will be free at all times from escaping liquor from other tanks than those directly overhead. They will consequently last longer, and will be found more economical in the long run.

In constructing the tanks the corrosive nature of the liquors they are to contain must not be lost sight of, or the repair account will be unnecessarily high. Even in the best constructed plants this account is very large, especially if the foreman or manager in charge is of a slovenly nature. Every supporting timber, plank and pin should be painted on all sides with hot, soft tar before being put in place. This is absorbed by the wood and protects it against the destructive action of the acid liquors. The tanks should be constructed of 3-in. plank, with an inside shell of the same thickness, and a space of 3 in. between the tank and the shell. All should be put together with wooden pins



FIGS. 3-4.—SECTION AND SIDE ELEVATION OF MUFFLE FURNACES, NATRONA, PA.
Scale. 1 in. = 10 ft.

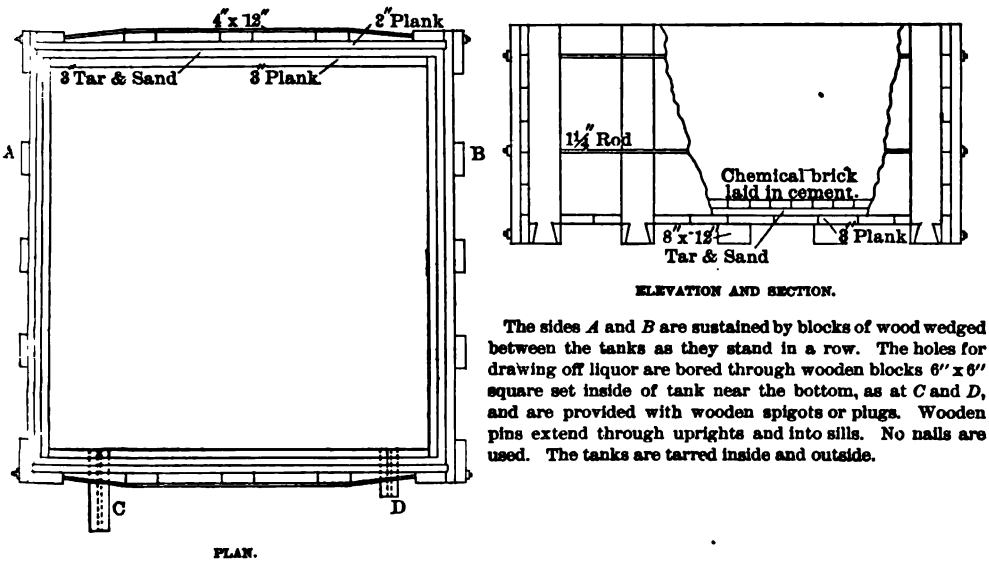
and bound together as shown in the accompanying engraving. The space between the tanks and the shells should be filled with a mixture of hard tar and sand. The bottoms should be covered, to the depth of 3 in., with the same mixture fused to that between the shell and the tank. This bottom covering should then be protected against wear of the shovels by a layer of chemical brick laid in cement.

The holes for drawing off the liquor are bored through wooden blocks 6×6 in. square, set inside of the tank near the bottom, and are provided with wooden spigots or plugs extending to the weak and strong liquor launders. All the launders are made preferably of very sappy yellow pine, dug out; with the ends, or joints halved together and calked with oakum and red lead.

The proper dimensions of the lixiviating, settling, and precipitating tanks are: 12×12×4 ft.; 12×12×6 ft.; and 12×12×6 ft., respectively.

In preparing the mixture of hard tar and sand, composed of about equal parts, care must be exercised not only to produce a homogeneous mass, but also that all the moisture and air be expelled. The usual method of procedure is to heat the hard tar in a large kettle, and stir it until all the moisture and air has been expelled, then adding the sand hot and stirring the mixture until the desired result is obtained. The sand must of course be first screened, and have all combustible matter burned out before mixing with the tar.

The lixivating tanks are provided with wooden slats laid on the bottom and covered with old hay or small pieces of refuse coke. The slats are usually made of hemlock lumber 2×2 in. square, beveled on the tops and laid close together. The calcined mixture is first lixiviated by weak liquor from a previous operation, and then by water and weak muriatic acid from the condensing towers con-



The sides *A* and *B* are sustained by blocks of wood wedged between the tanks as they stand in a row. The holes for drawing off liquor are bored through wooden blocks 6" x 6" square set inside of tank near the bottom, as at *C* and *D*, and are provided with wooden spigots or plugs. Wooden pins extend through uprights and into sills. No nails are used. The tanks are tarred inside and outside.

FIGS. 5–6.—PLAN, ELEVATION AND SECTION OF LEACHING TANKS, NATRONA, PA.

nected with the furnaces. After the lixivation is completed the residue (purple ore or "blue billy") is shoveled out on the platform *B*. Under this platform should be a railroad siding for the purpose of conveniently loading the purple ore on gondola cars, for shipment, through the openings provided in the platform, as shown in the plan. The platform being constructed higher on one side than on the other, also serves to drain the lixiviation water from the purple ore.

All the copper liquors are run into the settling and storage tanks *C;* the weak liquors being pumped or blown back with steam injectors to the lixiviation tanks when required. Any liquors of 18° B. and upward are left in the settling tanks until all the lead sulphate, etc., has settled out. The lead sulphate usually contains some gold, indeed an average of $100 per month is

not an unusual recovery in a plant of the capacity described, and it is therefore essential for the recovery of this as well as for other reasons that everything that will settle out of the liquors be given time to settle in these tanks. A part of these tanks *C* may also be utilized for precipitating the silver by the iodine, *i.e.*, the Claudet process;* but since in practice this process leaves an average of 5 oz. silver in a ton of the precipitated copper, and since electrolytic copper works, as well as blue vitriol works handling or using silver and gold bearing copper will pay for at least 95% of the silver contents, and full market value of the copper and gold contents of the copper cement, this part of the copper extraction process does not pay, especially since the cost per annum of precipitating the silver by the iodine method in a plant of this size is upward of $12,000.

After the lead sulphate, etc., has settled out of the strong liquors they are run into tanks *D* and the copper, silver and gold contents are precipitated by means of clean, thin scrap iron from rolling mills. These tanks *D* should have wooden slats so placed as to form an open false bottom, about 2 ft. above the real bottom, for the support of the scrap iron. Live steam being let in the liquor during the process of precipitation, the heat therefrom serves to accelerate the process, and also serves to keep the liquor in motion, and wash off the copper from the iron as fast as precipitated. The copper then naturally finds its way between the slats to the bottom of the tank, and when it is desirable to remove it the remaining scrap iron can readily be removed from the false bottom practically free from cement copper. The cement copper on the bottom of the tank after washing through perforated cast-iron plates set in a frame over tanks *E* assays about 90% Cu, 35 oz. Ag, and 0·15 oz. Au per ton. The remaining chloride solutions are then run into the sewer, first through a series of tanks *F* in the ground, which are filled with scrap iron, to recover any copper, etc., that may be left in the waste chloride solutions and wash waters.

The cost of treating 2,000 lb. of Spanish pyrites cinders, as above described, at Natrona, Pa., in a works having a daily capacity of say 200 tons of mixture (cinders and salt), will be about as follows: Two samplers at $2·50 each, $5; six mill men at $1·75, $10·50; one mechanic, $2; two engineers at $2, $4; three firemen at $1·75, $5·25; four weighmen and furnace chargers at $1·75, $7; 28 furnacemen at $1·75, $49; one hoistman, $2; 27 furnaced material handlers, coal and cinder wheelers, etc., at $1·50, $40·50; two leachers at $1·75, $3·50; four copper precipitators at $1·50, $6; unloading cinders and salt, $20; loading purple ore for shipment, $15; 21 tons salt, at $3, $63; pyrites fines, $7; 20 tons coal at $1, $20; 5½ tons sheet-iron scrap at $7, $38·50; repairs, depreciation, management, etc., $40—total, $338·25; or $1·87 per 2,000 lb. of pyrites cinders and $1·69 per 2,000 lb. of mixture.

* In the Claudet process the silver is precipitated from the chloride solution by the addition of zinc iodide, the quantity required being larger than called for by the silver if lead be present in the ore, since a part of the lead is thrown down as lead iodide. The precipitate, consisting chiefly of silver iodide (and lead iodide), is reduced by metallic zinc in the presence of hydrochloric acid, metallic silver and zinc iodide being produced. The silver is obtained as a spongy mass mixed with pieces of zinc and metallic lead reduced from lead iodide, together with any gold that may have been contained in the ore. These reactions are expressed by the equations: (1) $2\,AgCl+ZnI_2=2\,AgI+ZnCl_2$; and (2) $2\,AgI+Zn=ZnI_2+2\,Ag$. The zinc iodide is used over again.

Copper Mining in New South Wales.

By Louis Janin, Jr.

The increased price of copper has greatly stimulated the working of copper mines in the several British Colonies, but sufficient time has not as yet elapsed to show a great increment of production, as most of the newly formed companies are still developing their mines, whether new discoveries or formerly abandoned ones. In point of past production South Australia easily leads the other Colonies, being credited to date with nearly $100,000,000 worth of copper since the discovery of the famous Burra-Burra mine. This mine, though provided with a small working capital, produced some $3,640,000 in the first five years of its existence, and has paid dividends to the amount of $4,000,000.

Next to this Colony in the discovery of profitable copper mines was New South Wales; though this has not as yet rivalled the former in extent of production the conditions surrounding its mines justify special prominence.

Copper mining had a virtual renaissance when the Western division of the New South Wales Government Railway was completed to Cobar in 1891, and this was followed by an enterprising syndicate of coal miners and coke producers obtaining a lease on the property of the Great Cobar Copper Mining Co., Ltd. These tributers made a great success, and the attention of investors was attracted to other mines which had been shut down for years. Still, the progress which might have occurred was greatly interfered with by the law which prevented mining on private property for metals other than gold. In 1896 the Mining Law Amendment Act allowed entry on private lands for many other minerals, but in the interest of the pastoral sovereigns of the country, copper, which is yet destined to be the most valuable metal product of the Colony, was exempted. This state of affairs is likely to be changed soon, as a bill, including copper among the metals which it is permitted to search for on private property, is before the Legislative Assembly at the present writing.

The copper mines of New South Wales may be divided in order of their present relative importance into three geographical divisions, which also have their geological differences. I. The Cobar, or Western division, embracing the Cobar, Mount Hope, Nymagee, Overflow, Giralambone, Gilgunnia, Yellow Mountain and Melrose districts. In this division eruptive rocks are uncommon, the deposition of ore being in Silurian slates as a rule, though it is possible that some schistose rocks, now called slates, are altered eruptives. II. The Central division, with Mount Conoblas as the principal center of volcanic activity, embraces Copper Hill, Cow Flat, Burley-Jackey, Blayney, Brown's Creek, Apsley, Burraga, Summer Hill, Peelwood, Belmore, Goodrich, Sunny Corner, Oberon, Milburn Creek, Cadia, Cargo, Carangara, and others more or less prominent. The ore occurrence in this belt is always accompanied by eruptive rocks, which have never been classified, but which are termed porphyry-andesites and diorites. Owing to the profusion of mines in a comparatively small area and the proximity to fair coking coal, as well as railway facilities, this district will doubtless be the most important one of the three. III. The Coast division. To the north and to the south of the coal areas copper is of frequent occurrence.

In this division there can be no similarity traced between the scattered mines. Unlike the other districts there will be little opportunity to centralize metallurgical operations. The mines in this section which have been the most productive are the Cornish mine at Barraba and the Lake George mine at Captain's Flat, with other occurrences more or less promising.

The Cobar District.

For thousands of square miles from the Darling River on the north to the Murray on the south and from Dubbo on the east through New South Wales, South Australia and West Australia to the Indian Ocean, the topography of the country is monotonously uniform. There are slightly undulating plains with a succession of ridges, and here and there a hill which is entitled a mountain. In proximity to these so-called mountains, which never extend beyond a few miles on their strike, and which may mark some geological disturbance, the majority of the lodes occur. The veins generally conform to the strike of the chain and it is to the resistance to erosion of the silicious matter of the veins that many of these "mountains" owe their prominence. While it is probably an exaggeration to say that every "mountain" has its vein, still every profitable or promising mine in the wide belt of country is near these occurrences, if not actually on them. These veins invariably contain some metallic substance, either precious or base, but are often too low grade to be profitably worked. It is a saying among miners familiar with this district, that the best shoot of ore is on the south side of the hill, and the writer's experience confirms this. It is in such country as this that the Cobar and other districts in the same group lie, and that the famous lead-silver mines of Broken Hill and, some two thousand miles to the west, the gold bonanzas of Kalgoorlie occur.

The Great Cobar mine here has been extremely profitable and in replacing antiquated methods by modern practice, even to a limited extent, has opened the eyes of many of the old school copper miners. In 1870, when the mine was discovered, it was remote; 459 miles from Sydney by the present railway, which then extended to Bathurst only. In shipping the high-grade ores the miner had the choice of 300 miles of rough road to Bathurst or over 100 miles to Louth on the Darling River, with its precarious water stages, and for reduction the choice of the works at Waratah, New South Wales, or at Adelaide, South Australia, both belonging to the English & Australian Copper Co. Over 3,000 tons were shipped to the latter place, before reverberatory furnaces were erected at the mine and the product became ingot copper. Cobar is not a well wooded country and as smelting proceeded with voracious Swansea reverberatory furnaces, the country became skinned of box, gum and pine for miles around, and as the ores became of lower grade, notwithstanding careful sorting, and more rebellious, according to the lights of the old Welsh practice, expenses, instead of being reduced to meet the changed conditions, increased greatly. This, combined with the fall in the price of copper after the collapse of the Secretan syndicate, caused a suspension of work which bid fair to be permanent.

In 1893 some intelligent coal miners who had unprofitable coke ovens and coal mines at Rix's Creek, searching for an outlet for their product, entered

upon copper smelting, at first unsuccessfully, in the eastern part of the Colony, and later at Cobar. Here the maps of the mine at the date of its closing down showed solid bodies of sulphide ore from 34 to 40 ft. wide, extending over a considerable length of ground. Oxidized ores of too low grade to work previously had been left in the upper levels and 200,000 tons of cullings were exposed on the dumps. The mine made no water and was in condition for inspection and work. The railway was completed, and the Government administering the railways granted low freights on their coke. Believing that the price of copper was as low as it was likely to be, some old lead water-jacket furnaces were purchased, the crucibles filled, and smelting commenced on a suitable mixture of ores. Everything succeeded; gold, which had not been previously known to exist in payable quantities, was found to occur to the extent of $80 to $100 per ton of copper. This added greatly to the profits, so that the furnaces have been constantly added to until now there are five in blast at Cobar and one treating refinery slags at Lithgow, in juxtaposition to coal and coke.

The deepest level of the Cobar mine is 540 ft., though a bore hole has been put down below this in sulphide ore. From the surface to 228 ft. the ore was oxidized; from this to 324 ft. it was mixed oxides and sulphides; while below this it became massive pyrrhotite and chalcopyrite with some magnetite, low in silica and averaging about 4% Cu. It is not clearly proven that the sulphides decrease in grade in depth, though this may be the case, as in most mines. The vein system is wide, there being practically three veins; the West, Middle and Eastern, on which latter two practically no work has yet been done. There are the shoots of ore which appear to be making together in depth, that at Beckers' shaft is 120 ft. long at the 420-ft. level, but is elongating at the 540-ft. level. It averages 20 ft. wide. The main shoot is at the Barton shaft and is a lens some 320 ft. long, with its median horizontal axis 70 ft. long. To the south of this there are two shoots, separated by a horse of country, and these on the 228-ft. level are 68 ft. long, the western body 10 ft. wide, while the eastern is 25 ft. Between the shoot at Beckers' shaft and that at Barton's there is blank ground for 260 ft. in length. Ore has replaced much of this country at the 540-ft. level, so it would seem probable that in depth there will be a nearly continuous shoot here as well as at the southern end, where there is now 130 ft. of barren ground between the shoots.

The upper levels contain all forms of oxidized ore, while secondary ores as well as sulphides occur in the intermediary zone. The following analysis shows the composition of the rich carbonate ore: SiO_2, 10·64%; Cu, 38·23%; Bi, 0·21%; Fe, 0·78%; CO_2, 17·47%; H_2O, 3·79%; oxygen, etc., 10·24%; total, 100%.* The following analysis of the sulphide ores at present mined is interesting, although the sample is above the average grade in copper: H_2O at 100°C., 0·20%; Fe, 42·65%; Cu, 10·15%; Zn,Pb, traces; SiO_2, 16·09%; Al_2O_3, 0·44%; MgO, 0·34%; S, 20·74%; SO_3, 0·23%; oxygen by difference, 9·16%.† The oxygen is to be charged to the magnetite present. Notwith-

* W. A. Dixon, F. C. S., *Annual Rep. Dept. of Mines*, New South Wales, 1878.
† An analysis by J. C. H. Mingaye, F. C. S., Analyst to the Dept. of Mines, New South Wales.

standing the low sulphur contents the ore heap roasts well, considering the slight attention paid to the operation.

Heap roasting is carried on in the usual manner, the V system being partially adopted. The ore takes a rather shorter time than usual to roast completely, six to eight weeks being normally consumed. Some of the piles sinter when the heat is too fierce; as a consequence blasting of the ore is necessary. Matte is sometimes produced, but offers no difficulty save in breaking up, and the cost of the operation, including trucking, is about 25c. per ton, which seems low, as the furnaces are at a considerable distance from the roast piles and at a higher elevation. The plant is uneconomically arranged. The water jackets, five in number, are of wrought iron, 96×36 in. at the tuyeres, are boshed and are 54 in. in height. Above the jackets is common brick, and as the furnaces are run cool with the charge kept low these last well. Little flue dust is made, as there is little fine ore charged, the furnace mixture being roughly composed of one-third heap-roasted ore, one-third raw sulphides and one-third oxidized ore, with certain additions of foul slag. No flux is used and 10% of coke is sufficient. These furnaces treat about 100 tons of charge, excluding coke, daily, sometimes exceeding this and sometimes running slightly below. The slag runs well, is fairly clean, and its composition is normally SiO_2, 27 to 29%; FeO, 56 to 58%; Cu, from 0·3 to 0·5%. The balance is practically Al_2O_3. It is not properly smelting but melting. The matte runs about 32% Cu, though by lessening the production of sulphide ore it is made to go as high as 45%. The former percentage suits future operations at Lithgow, however. The costs at Cobar, including smelting, mining, management, etc., no allowance for depreciation being made, may be put roughly at $5 a long ton for smelting to 32% matte, or $4·62 per ton of 2,000 lb. Mining is done by contract, some stopes with machine drills being let at 87·5c. per ton, including filling trucks and tramming to shaft. The total cost of making copper, as estimated by Mr. T. Longworth, does not exceed £32 per ton of ingots, or 7·2c. per lb. Cobar copper brings a higher price than Chile bars, but refining, transportation charges, commissions and allowances for losses must be deducted. The gold contents vary from 3·5 to 5 oz. per ton, and the silver from 12 to 20 oz. Some of this copper has recently been refined at Perth Amboy, N. J. An electrolytic refinery is now about completed at Lithgow.

Cobar, Chesney and Others.—This mine is similar to the others in the vicinity of the Great Cobar, such as the Cobar Gold Mines, Ltd., Berribungie, Mount Pleasant, Young Australian, Great Western and Occidental, all of which have produced gold, with the exception of the Berribungie, but none of which is yet on the roll of copper producers. The outcrop here is slate, often silicified to quartzite, as is the sandstone, and intercalated with quartz seams. Shoots of gold ore, some of considerable extent have been found at various points in this wide slate and quartz body. Exploration in depth has everywhere shown copper but nowhere massive sulphides except in the Berribungie, where the body as at present exposed is of limited size. The copper ores all carry gold, but even where the sulphides occur as at the Chesney they are too silicious to smelt without employment of flux. The lode in the Chesney has been proved as deep as 560 ft.,

but here also the ore is not massive, being merely intercalations of a good quality of chalcopyrite replacing the quartz seams found above. Concentration must be followed if these ores are to be treated alone, but as the ore averages in the upper levels 3·6 cwt. of gold, which is carried by the gangue as well as by the chalcopyrite, it will cause a loss of this metal. Fine crushing will be required and the loss of copper will also be high. Though mining will be cheap in such large bodies the eventual yield may not be over 3% Cu in the form of concentrates, which undergo a further loss in treatment.

Mount Hope District.—The New Mount Hope is the most important mine in this district, which is about 100 miles south of Cobar, in similar country rock. Its production up to 1899 amounted to 5,070 long tons of copper from about 25,000 tons of ore; on the tailings heap are about 8,000 tons of seconds which would pay for smelting if coke were cheap. New Mount Hope presents the same topographical appearance and resembles Cobar in its three parallel outcrops of ferruginous slates with quartz seams. The ore occurrence, however, differs, though the country rocks, slates and sandstones, frequently silicified to quartzite, are similar. Unlike Cobar, there is no well-defined boundary or limits for ore deposition. In the sandstone on the west wall of Cobar no ore is found, while the matrix of much of the Mount Hope ore is a fine-grained sandstone. The ore bodies are extremely irregular. The shoot is about 400 ft. long as shown in the workings—about 150 ft. of this was solid ore with a maximum width of 70 ft., though the main outcrop is only 24 ft. wide. Prospecting has not advanced further than 100 ft. to the north or south of the stopes, though owing to the irregular occurrence of the ore frequent cross cuts have been run.

The minerals being disseminated through the gangue the ore is necessarily concentrated. A rock breaker and Cornish rolls are employed for crushing and Cornish jigs for concentration; about 5 tons are concentrated into one. The furnaces are small, treating less than 5 tons a day of 24 hours. The furnace charge is made up as follows: Hutchwork, 1,500 lb.; raggings, 750 lb.; hand-picked sulphides, 375 lb.; foul slag, 375 to 750 lb.; ironstone flux, 375 lb. Charcoal is also added and occasionally ashes. The flux comes from Mount Allen, 10 miles distant. Here a large ironstone outcrop was mined for fluxes for a number of years before it was discovered to be auriferous. A company was then formed to work the property for gold. The tailings from amalgamation were purchased by the Mount Hope Co. at $1·75 a ton until a cyanide plant was erected. The Mount Hope Co. must now rely upon its own sulphides to supply the necessary flux. As there is limestone near by, should a railway be completed to Mount Hope, blast furnaces may eventually be employed, with replacement of some of the iron by limestone.

The Great Central mine, within a few miles of the New Mount Hope, produced previous to 1899, 2,170 long tons of refined copper. This deposit differs from the New Mount Hope, being associated with igneous rocks, brecciated porphyry being the gangue. At 240 ft. in depth sulphides are encountered, the ores in the upper horizon being carbonates and oxides. The amount of exploration work here is small, considering the great extent of the property and the numerous outcrops that are not only cupriferous but gold bearing.

The oxidized ore here is jigged in Handcock jigs, after the best ore is culled out to be smelted directly. The concentrates run 14% from this class of ore, while those from the sulphide ores average 12%. There is a large loss in concentration, owing to the scanty water supply and the necessity of using water charged with slimes over and over. Concentration as seen here and at New Mount Hope is in the most primitive form, though to improve on present methods would take considerable capital outlay for conservation of water and the use of slime dressing machinery.

The Giralambone mine was discovered in 1880, but has never attained great importance, owing mainly to the nature of the ore, which required concentration. This could be done only by submitting to heavy losses in treating the earthy carbonate ores of the upper levels and the melaconite ores of the transition zone. All the minerals here, whether oxidized or not, are finely disseminated through the gangue. Though the country rocks, slates and sandstones, the latter being frequently silicified to quartzites, are similar to those of Cobar, the ore occurrence is somewhat dissimilar, though resembling the occurrence of ore in the east and middle veins of the Great Cobar. Generally speaking, the ore occurs in segregations and replacements in the country rock, no wall limit as yet being found. The greatest deposition is naturally where the slates are less compact and compressed, thus affording passage for the cupriferous solutions. The main shaft has been sunk to a depth of 544 ft., though the mine has not been opened up thoroughly below the 300-ft. level. On the 200-ft. level the ore is stoped 25 ft. wide.

The successful treatment of Giralambone ore, with its high silica contents, its tendency to slime in concentration and in the absence of massive sulphides, has been considered one of the most difficult metallurgical problems in New South Wales. The mine is, moreover, in an arid region with little opportunity, owing to surface configuration, to conserve water at the time of the infrequent rains. The practice at the time of the writer's visit was to divide the ore into three classes: (1) Carbonates; (2) "oxides" from the zone of transition; (3) sulphides from the 240-ft. and 300-ft. levels. The carbonate ores are hand picked into three classes; firsts, average 13% Cu, go direct to the furnaces; seconds, go to the jigs; while the thirds, or waste, are discarded. The "oxides" and sulphides go directly to the concentrating works. The crushing and concentrating plant consists of a Gates breaker, Krom and Cornish rolls, a Handcock jig for the coarse product from the Cornish rolls and a May Bros.' jig for the Krom roll fines. The first concentrates are smelted, while the seconds or middlings are stacked to await future treatment, while the tailings run to waste. The concentrates from the carbonates and oxides average 10·9% Cu. The sulphide concentrates, however, assay only 6·5% Cu, although the original ore averaged 3·1%. The loss in concentrating this class of ore is said to be fully 50% of the copper contents. The general average of all seconds (middlings) and tailings is said to be 5·8%. The accounts of the company for the half year ending June 30, 1898, show the enormous loss that occurs in dressing. During this period 2,579 tons of carbonates of an average of 4·6% Cu, 3,263 tons of sulphides, averaging 3·1% Cu, and 953

tons of "oxides" were concentrated. The total copper contents of this ore was 283 tons, 12 cwt. of workable product. There was produced 464 tons of carbonate concentrates going 10·9%, 706 tons of sulphides going 6·5% and 273 tons of oxides averaging 15·8%; a total quantity of 1,443 tons containing 139 tons, 10 cwt. of copper, a recovery, not including the seconds, of 47·5%. This product has to undergo a second loss, as the fusion slags run as high as 12% Cu.

The saving on the various classes of ore was as follows: Carbonate ore, 42·2%; sulphide ore, 44·9%; "oxide" ore, 67·5%. These concentrates are smelted in reverberatory furnaces 20×15 ft., made of local refractory material. The charge consists of 18 to 20 cwt. of carbonate and oxides, 10 cwt. of sulphide concentrates and 12 cwt. of ironstone flux. The proportion between SiO_2 and FeO in such a charge would be 1 to 1·05, excluding the iron which entered the high-grade regulus. Such a slag would work well in the furnace were it not for the high alumina. The slag is of high specific gravity and the rapidly reduced copper is held in it in fine shots. The books of the company show that for the half year 1,443 tons of concentrates with 107 tons of 10·1% concentrates held over from the preceding year were smelted, together with 1,153 tons of screenings. The total copper contents was 295 tons, 11 cwt. The product was 206 tons, 14 cwt. 2 qr. 1 lb., a saving of 68·1%. The saving in concentration, as previously mentioned, was only 47·5%, so that the net result was the phenomenally low figure of 32·34%, the lowest saving, perhaps, in the modern history of copper ore treatment.

Of the mines, which have published reports, the Nymagee ranks second in the amount of dividends. Worked by practically the same company as the Great Cobar, up to 1896, when it was purchased by the syndicate working the Great Cobar mine, it had paid $470,000 in dividends on a nominal capitalization of $400,000. The last dividend was declared in 1889. The production up to 1899 was 17,008 long tons of refined copper, containing no gold and a trifling amount of silver, but notable for its purity. This came from 164,692 tons of ore mined, giving an average of 10·3%. The high average was obtained only by careful assorting in the mine and on the surface. Since the Cobar syndicate took over the property the general average of ore smelted in the blast furnace has been 4·28% in 1897, when 11,910 tons were treated and 4·53% in 1898, when 17,150 tons were smelted in the water jackets.

The occurrence of ore here is similar to that of Cobar and the country rocks are similar. The mine has been worked to a greater depth, however, though the shoot seems to be pinching. One distinguishing difference from the Great Cobar is in the existence of extensive bodies of lead carbonates in the upper levels. There is a considerable quantity exposed and it is believed that exploration will show a quantity which will warrant the erection of a lead blast furnace.

The metallurgy practiced at Nymagee warrants no special mention. Partial pyritic smelting has been tried, with partial success, the matte being refined afterward in reverberatories. Coke here is expensive, as it has to be hauled by teams from Hermidale, 50 miles distant, by roads which are in poor condition, particularly in winter. It is economical to refine the matte to a high grade product; white metal is now shipped.

Other Mines.—At Bobadah, the Overflow mine shows considerable copper,

and in all probability when depth is attained the mine will be profitably worked for that metal. At the present time, though carrying a good average value (from $10 to $15 a ton in gold), a fair percentage of lead and considerable copper, it has earned nothing but assessments for its shareholders. At Anaconda, Big-Ben, Melrose, Yellow Mountain and other places in this district copper is found on or near the surface. As yet none of these prospects have been sufficiently developed to determine with certainty its future, but as they have had the same surface showing as those mines which have been profitably opened up it is more than possible that some of these may be successfully worked eventually, particularly if railway communication is brought about.

To Summarize.—In treating the Cobar and surrounding districts as a whole it will be seen that only two mines, the Great Cobar and Nymagee, have encountered the massive sulphide ores necessary to economically successful blast furnace work. The other mines, such as Mount Hope and Giralambone, have heavy losses of metal because their ores are silicious and mainly carbonates requiring concentration. Moreover, concentration itself is expensive and must be limited where the water supply is so deficient. So many promising mines have been opened, with others doubtless awaiting discovery, that any suggestion which may bring about the success of the district as a whole will doubtless be welcome.

Central District.

The Central district contains the oldest copper mines in the Colony, Copper Hill, near Molong, being the first worked, though Summer Hill, on the road from Bathurst to Rockley and Burraga, was the first producer, closely followed by Carangara. Many of the mines were worked by private enterprise or close companies, so a record of their production is unobtainable. There seems no doubt, however, that the Burraga has produced more than any mine in this district.

The Burraga mine, 42 miles from Bathurst, has been worked since its discovery in 1877. Three years later furnaces were erected by Mr. Lewis Lloyd, an experienced Welsh copper smelter, who worked the mines continuously, save in 1889 and 1891, until possession was taken by an English company formed early in 1899. Before this the property had often been bonded for sale and the owner had made a considerable portion of his profit from forfeited first payments, though $500,000, the price finally paid, was much in excess of any option that had been previously given. From 1880 to 1899 the production has been 5,654·6 tons of copper. A quantity of ore, estimated at 100,000 tons, is on the dumps. Samples from this rejected ore assayed 4% Cu, and a small amount of gold and silver. It is proposed to concentrate this, but as the gangue is silicious and hard and as water is not plentiful there is some doubt of success. It seems more probable that in the future the culled silicious ore will be smelted with the heap-roasted chalcopyrite. Careful blending will be necessary to produce a copper in which the gold and silver will pay for separation. In all probability a plant will be erected at Lithgow to treat the Burraga mattes with those of other mines, and with custom gold and silver ores, which latter of course will have to bear the expense of realizing on the gold and silver contents of Burraga matte. The

appliances have been of the most antiquated description. The reverberatories were small and the sheds dilapidated. The only concentrating appliance was a crude Cornish hand jig. Black powder was used in the mine, though the formation was one where a higher explosive could have been employed with economy. As the property had paid from the first, the new company with improved methods will probably be successful. As most of the development work here is confined to one shoot about 100 ft. long, which has been worked to a total depth of 500 ft. on the incline, the ore occurrence at Burraga is not clearly shown. The ore is chalcopyrite, occurring massively in a green acid rock, at times somewhat schistose from pressure. Locally it is termed diorite. Copper has been found on the strike of the lode and in the writer's opinion other shoots than the present, which has been so worked to resemble a pipe shoot, will be developed in time. The copper manufactured at Lithgow from Burraga matte is well known in the English copper market as "Lloyds" and commands a high price on account of its purity.

Summer Hill, the oldest copper producer in New South Wales, was unworked at the time of the writer's visit, though shortly afterward the old workings were cleaned out and a quantity of ore shipped to Lithgow. Assays of samples from the dump showed that the ore contained gold. A number of these ran from 1 dwt. to 1 oz. per ton. A shipment of picked ore from the old dumps ran 11·25% Cu by fire assay. Several shafts were sunk and a tunnel driven cutting the vein at a low level. The country rock is serpentine, in which formation gold is found in the immediate vicinity at Bunnamagoo. The outcrop is large and continuous. The shafts here have already reached the sulphide zone.

Cow Flat mine is about 5 miles from George's Plains on the railway and has been worked rather extensively in past years. The carbonate and oxide ores occurring in schists were found to extend only to a shallow depth when unaltered sulphides came in. These were of fair grade, but the size of lode in depth as reported was incommensurate with the outcrop. The levels were inaccessible in 1899. The six furnaces still to be seen were employed in treating ore from various mines in the vicinity, one of which was the Apsley.

The Belmore mine, Larry's Creek, Oberon, Wiseman's Creek, Essington and other mines are in this portion of the Central District, all within easy reach of the railway. All the lodes here are of fairly good size, though the ore, with the exception of the Belmore, carries a considerable quantity of blende and galena, making it rather refractory to smelt.

The Great Blayney mine is well situated at the junction of the Carcoar and Cowra branch with the Western Railway, the former running south and joining the main southern line at Harden and the latter going west to Cobar and Giralambone. The Western line also passes Lithgow. This mine was discovered in 1881 and was worked unsuccessfully by several companies. In 1897 it was unwatered, when exploration work showed the occurrence of large, though irregular, bodies of low-grade sulphide ore. At the 180-ft. level it was stated it was fully 30 ft. wide. The ore fortunately contains an excess of iron, which will enable the furnaces which have been erected on the mine to smelt more silicious ores from other mines in a large area surrounding it. No

reports have been made as to the success of its working, but considerable shipments of matte have been made, though of low grade in copper, containing only from 14 to 20%. While there is some doubt as to the success of this mine worked by itself, there can be none whatever as to the ultimate result, owing to its commanding position, through the fluxing character of its ores, so necessary in this part of New South Wales, and its location. About 6 miles from here is Brown's Creek, where a large ferruginous and calcarious deposit had been worked for gold for many years. An English company erected here a 60-stamp mill, utterly unsuited to either crush or treat the ore, as this is a soft, slimy mass which can be mined by picking. The extent of the ore body is considerable and is proven by most careful sampling to average 7 dwt. per long ton. The gold, however, is not in a condition to save by amalgamation and as a consequence operations were unsuccessful. Cyanide could not act upon it economically, as there is from 0·5 to 1% of copper throughout the mass with occurrences on one of the walls of much higher grade ore. The ore, owing to its high percentage of lime and iron and apparently low silica contents, was practically self-fluxing, as when charged into a crucible and heated in the assay furnace it smelted without additions of fluxes. In many places this body is over 300 ft. wide and has been explored for a length on the vein of about 1,000 ft. There are many hundred thousand tons of ore available above water level which would pay to smelt at Blayney with the ores from the Great Blayney mine, as a tramway could be cheaply constructed to the latter point. The percentage of copper is low, but the concentration of the 7 dwt. gold in the copper of the Great Blayney would render the operations profitable. There are other similar deposits in the district, as the appearance of copper is by no means unusual. Moreover, the mines at Milburn Creek would be enabled to ship ore to Blayney and have it more economically reduced than would be possible at works placed on the mine itself. The deposit at this latter place is said to be large and the grade to be quite high in copper, with a certain amount of precious metal contents.

Throughout this Central District there are many occurrences of copper which at the present time are too unimportant to enumerate, but which on the establishment of a central smelting plant would doubtless be developed and in many instances would show profitable ore bodies. The ores from all these mines could be treated at Blayney and the resultant matte shipped, together with the matte from the smelters of the Cobar district, to a point near tidewater where fuel is abundant and cheap. The railway department makes a lower rate upon matte than upon copper. For this reason it would probably be better to refine the matte at Newcastle than at Lithgow, though at this latter place there is also an abundance of cheap coal of fair quality.

The Coast District.

The most important mine in the Coast District is that belonging to the Lake George Mines, Ltd., at Captain's Flat, near Bungendore. Although this mine at the present time is not worked for copper, results obtained here are of great interest, as the ore is not only refractory, containing a high percentage of zinc, but the figures of cost give a comparison between ordinary

blast furnace smelting and pyritic or partially pyritic smelting. The vein here is wide and the ore bodies extensive, but they are of too low grade both in copper and the precious metals to be profitably worked. The lode is conformable to Silurian slates and as part of the lode shows a perfectly bedded structure the country rock has doubtless been gradually replaced by ore. In places no distinct walls are seen and the compact ore passes by insensible gradations into country rock impregnated with iron pyrites. The lode varies from 20 to 40 ft. in thickness and the copper has varied from under 1% to a maximum of about 2·5%. An average analysis (Report Mines Depart., 1897, p. 21) is as follows: Moisture at 100°C., 0·10%; combined H_2O, 0·65%; insoluble in acids, 23·70%; Fe, 17·75%; Zn, 14·50%; Pb, 8·4%; Cu, 1·11%; $PbSO_4$, 0·32%; Al_2O_3, 1·38%; CaO, 0·40%; MgO, 0·65%; SO_3, 0·25%; CO_2, 2·08%; P_2O_5, 0·57%; S, 27·97%; undetermined, 0·53%; the gold contents, 1 dwt. 12 gr. per ton; and the silver, 2 oz. 8 dwt. 15 gr. The high percentage of sulphur, lead and zinc is at once apparent.

When the property was taken over by the Lake George Mines, Ltd., three large water-jacket furnaces with a nominal capacity of 60 tons daily each were erected, with a smaller furnace to refine the low-grade matte produced in the first fusion in the smelting furnaces. A very small quantity of fuel was used, the blast being heated in stoves. Although initial difficulties were met with, this partial pyritic smelting was kept up for some time, until unsatisfactory results brought about a change of management and a change to direct concentration to a matte of about 30% Cu. Two classes of ore were mined separately, one known as bulk ore and the other as silver ore. The latter came from a richer seam on the foot wall, which in the early history of the company had yielded from 9 to 20 oz. of silver per ton, but which had decreased in value latterly. The main difference being between the two classes of ore at the last workings is in the quantity of silica and iron; the bulk ore being lower in silica and higher in iron than the silver ore. An analysis of the bulk ore showed SiO_2, 22·55%; Fe, 22·26; Zn, 13·26%; Pb, 9·70%; CaO, 1·60%; S, 29·50%; Cu, 1·25%; this ore also contained 1·15 oz. of silver and 82c. gold. The silver ore showed SiO_2, 26·48%; Fe, 15·19%; Zn, 14·28%; Pb, 8·05%; S, 27·50%; Cu, 0·94%; it contained $1·24 per ton in gold and 3·03 oz. of silver.

From the 1st of June to the 20th of October, 1898, three smelting furnaces were in blast, exclusive of the matte concentration furnace. These treated 24,173·93 tons of ore, including 28·98 tons of gossan ore, excluding the necessary shut-downs, which would probably work out about the same number of days or hours, no matter which process was used. These three furnaces worked 568 furnace days, with an average capacity of 42·2 tons daily in the period of pyritic smelting. When changed to direct concentration the three furnaces in 558 furnace days treated 33,758·45 tons of ore, an average of 60·5 tons per furnace daily, showing an increase of nearly 50% in capacity due to adoption of direct furnace concentration. The mattes in the first period ran 33·9% Cu, and in the second 29·30%, showing very little difference. The slags were about the same at both periods and may be taken as SiO_2, 34·43%; FeO, 35·73%; CaO, 10·86%; Zn, 12·16%; with copper contents from 0·3 to 0·4%. A matte analysis

showed Cu, 34·01%; Fe, 20·85%; Zn, 9·90%; Pb, 11·75%; S, 22·00%; Au, 1·63 oz. per ton; Ag, 60 oz. per ton. The cost, moreover, showed a decrease on the non-pyritic smelting, as will be explained by the following table; a period from January, 1898, to May 31st of the same year is also included, as the property at that time was under different management, though the system introduced then was followed from June to October 20, 1898:

	Raising Ore.	Development.	Surface Handling.	Smelting.	General Expenses.	Repairs and Plant.	Total.
January to May 31, 1898...	$1·556	$0·339	$0·105	$2·25	$0·315	$0·116	$4·684
June to October 20, 1898..	1·568	0·229	0·140	2·27	0·317	0·01	4·534
Nov., 1898, to May 31, 1899.	1·294	0·229	0·073	1·721	0·116	0·057	3·489

These figures show conclusively that the greatest reduction was in smelting expenses, though a certain amount is to be charged to the decreased salaries, coming under the head of general expenses. The smelting costs, November, 1898, to May 31, 1899, of $1·721 per ton of 2,000 lb. is probably the lowest cost yet published for matte smelting, and is all the more remarkable because coke averages about $10 a ton at Captain's Flat. At Mt. Lyell, where pyritic smelting is followed, under the charge of Mr. Robert Sticht, a most careful metallurgist, and where coke costs about the same as at Captain's Flat, the cost per ton of ore smelted to high-grade matte (thus including the cost of matte concentration) is given in the last half-yearly report of the Mt. Lyell Mining and Railway Co., Ltd., as $3·09 per ton of 2,000 lb. This would certainly indicate that under normal conditions the pyritic smelting process will not compete with ordinary matte concentration.

In this southern part of New South Wales there are several other properties, none of which has attained any prominence, but which in time may be developed to a productive point. In the northern portion of the Colony there are scattered occurrences of copper, none of which, as yet, assume any importance. The Cornish copper mine at Barraba has produced a small amount of copper and as it has been purchased recently for the Mining and Financial Trust of London by Mr. C. M. Rolker, it may be heard of soon. Generally speaking, however, the mines in this section of the Colony are too scattered to allow of being worked in unison as those of the two preceding districts may be in the future. Still, if bessemerizing and electrolytic refining works were erected on the seaboard, doubtless the production of copper and the development of copper mines would be stimulated in this section. For such a plant, Cockle Creek is probably one of the best sites, as not only is coal in the immediate vicinity, but the electrical generating plant, which was used in the expensive and disastrous Ashcroft experiments, is idle and available. New South Wales will probably show a marked increase in the amount of copper produced within a few years, especially if the laws permit and the people themselves do not place obstacles in the way of the introduction of capital. Short-sighted policy has prevailed heretofore, but doubtless experience has shown that the utmost encouragement must be given in order to develop this great natural resource of this Colony.

FELDSPAR.

THE conditions of general prosperity in manufacturing industries that characterized the year 1899 manifested itself in the increased production of feldspar above the output in 1898. The statistics are set forth in the following table, and the years 1898 and 1899 amounted respectively to 21,692 metric tons (21,350 long tons), valued at $107,147, and 27,399 metric tons (26,968 long tons), valued at $137,866, an increase of 5,618 long tons for 1899.

PRODUCTION OF FELDSPAR IN THE UNITED STATES.

State.	1897.		1898.		1899.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
Connecticut	5,871	$34,231	6,696	$37,944	11,104	$61,311
Maine and Pennsylvania	12,880	59,161	12,954	58,953	14,644	71,755
Maryland	1,500	7,875	None.	None.	None.	None.
Massachusetts	150	525	150	525	160	560
New York	1,500	9,600	1,550	9,725	1,060	4,240
Totals	21,901	$111,392	21,350	$107,147	26,968	$137,866

The chief supplies of feldspar in the United States were derived from Brandywine Summit, Pa., Branchville and Glastonbury, Conn., Georgetown and Topsham, Me., and Bedford, N. Y. Minor amounts were produced in Delaware, Virginia and North Carolina. At present the main production of feldspar for the market is limited to the States named above, but other Eastern States may enter in the field in the future. There are large quantities of feldspar in the granite areas of the Western States, but the freight rates are so high in comparison with the low price of the mineral, that it cannot be shipped with profit to Eastern potteries in competition with the present Eastern product.

For a detailed account of the occurrence, mining and uses of feldspar, reference should be made to THE MINERAL INDUSTRY, Vol. VII.

The main consumption of feldspar or "spar," as it is called in the trade, is in porcelain and china factories, where it is added to the mixture of kaolin and flint, and serves by its fluxing qualities to bind the entire mass together; yielding a partially vitrified material after it has been fired. It is also used as a constituent in making the glaze for chinaware and enamels, and in the manufacture of special kinds of glass. The purity of the product is very important, and for pottery uses it should be free from iron oxide, mica and quartz, yielding when fired a white opaque glass. The best quality is considered to come from the Maine quarries. The Connecticut supply while of good quality generally contains quartz. The price at point of production varies from $3 to $6 per long ton, depending on quality.

FLUORSPAR.

THE production of fluorspar in the United States in 1899 was 24,030 short tons, more than double the amount for 1898 and nearly six times as great as the production in 1897. This large increase was due to the greater demand of steel works, iron foundries, enameling works, glass factories and hydrofluoric acid makers. The price during 1899 varied with the quality and averaged $6·35 per ton of 2,000 lb. at the mine. The two chief producers of fluorspar in the United States in 1899 were the Kentucky Fluorspar Co. with mines in Crittenden County and works at Marion, Ky., and the Fluor Spar Co., whose works are in Hardin County and whose mines are in Hardin County, Ill., and in Livingston and Caldwell counties, Ky. The Rosiclaire Lead & Fluorspar Co., operating the older mines at Rosiclaire, Ill., also reported a large output and the Eagle Fluorspar Co., organized in 1898 to work the mines in Livingston County, Ky., produced a small amount.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton
1888....	6,000	$30,000	$5·00	1892.....	9,000	$54,000	$6·00	1896.....	6,000	$48,000	$8·00
1889....	9,500	45,885	4·82	1893.....	9,700	63,050	6·50	1897.....	4,379	36,264	7·65
1890....	8,250	55,328	6·70	1894.....	6,400	38,400	6·00	1898.....	12,145	86,985	7·16
1891....	6,390	38,000	6·00	1895	4,000	24,000	6·00	1899.....	24,030	152,655	6·35

PRODUCTION OF FLUORSPAR IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	France.	Germany.					Spain.	United Kingdom	United States.	Total.
		Anhalt.	Bavaria.	Prussia.	Saxony.	Schwarzburg.				
1896.....	1,940	5,600	5,218	8,672	805	1,218	3	400	5,443	29,629
1897.....	2,723	7,000	4,904	10,095	592	641	2	308	4,299	34,678
1898.....	3,077	6,415	4,440	11,863	775	294	5	507	11,018	38,394

From the official reports of the respective countries except the United States, for which the totals are based on direct returns of the producers, and for Anhalt, Saxe-Weimar and Schwarzburg-Sonderhausen, which are due to the courtesy of Herr von Scheel, director des Kaiserlichen Statistischen Amts.

In the American market flourspar is divided into six grades: American lump No. 1; American lump No 2; gravel; crushed; ground fine; ground extra fine. The foreign product appears in two grades only: lump and fine.

Fluorspar Mining in the United States.*

By E. E. Squier, Jr.

Fluorspar is mined in the United States only in Crittenden County, Ky., and Hardin County, Ill. It is found in veins in connection with limestone and yellow clay, together with a small amount of lead, calcspar and feldspar. The Kentucky fluorspar is a pure white massive variety, and is found in veins ranging from 6 to 30 ft. in width and varying greatly in length. The largest vein so far discovered in the State is owned by the Fluor Spar Co., of St. Louis, Mo., and is known as Mine No. 1. It is situated on the side of a hill and runs almost due north and south from 25 to 50 ft. below the ridge. It has been prospected a distance of over 1,200 ft. and the width varies from 15 to 30 ft. In this case the spar is found from the grass roots down.

The main shaft on this vein is 13 ft. square and has reached a depth of 96 ft. At this level two drifts, each 6 ft. high by 7 ft. wide, are being run north and south along the vein, cutting through an almost solid mass of pure white fluorspar. Heavy timbers are necessary to support the roofs. When a drift has been run far enough, a narrow track is laid, upon which small four-wheeled flat cars are run. The buckets are placed on these, and when loaded at the end of the drift are wheeled to the foot of the shaft. They are then hoisted to the top by a horse-power whim, so arranged that while one bucket is being brought up another is going down.

Upon reaching the top the bucket is put on another car, wheeled to the sorting and cleaning sheds, dumped, and returned for another load. At the sheds the spar is taken in hand by the sorters, hackers, screeners and barrelers. The large pure white lumps from 5 to 150 lb. in weight are selected first and turned over to the hackers, who chop off with hatchets the small amount of dirt adhering. It is then graded as No. 1 pure white lump fluorspar which is used for the manufacture of hydrofluoric acid and after grinding, for the manufacture of glass, enamels, etc. The slightly off or mixed colored lumps are also separated and cleaned. This grade is used by steel manufacturers and is known as lump fluorspar.

After taking out the large lumps the spar passes to the screeners, who separate the fine or gravel spar from the crushed or small lump fluorspar by the use of coarse screens. The crushed fluorspar is passed to the barrelers, who grade it into Nos. 1 and 2 qualities, according to color, and at the same time barrel it. When filled the barrels are headed up and loaded into wagons or placed in the warehouse. Barreled fluorspar is also used by acid makers, iron and steel manufacturers, blast furnaces, foundries and furnaces. Very little fluorspar of any grade is kept at the mines; it is hauled to the railroad and loaded on the cars or stored in the warehouse or on platforms as fast as it is taken out.

In Illinois, the fluorspar is found under the same general conditions as in Kentucky. It also is of the massive variety, but is of a mottled color and not as pure. The Fluor Spar Co. owns or controls extensive deposits at Karber's Ridge, Cypress Junction, and Shawneetown, Ill., which are worked on the same plan as in Kentucky.

* *The Engineering and Mining Journal*, May 6, 1899, p. 527.

FULLERS EARTH.

THE domestic production of fullers earth in 1899 was 12,356 metric tons (13,620 short tons), valued at $81,900. This was derived mainly from mines at Quincy, Fla., which were discovered about five years ago. There was a slight decline from the productions of 1897 and 1898, the amounts for these years being respectively 15,599 and 14,110 metric tons (17,195 and 15,553 short tons). The new mines of the California Fullers Earth Co., near Bakersfield, Cal., are being opened up, and the earth is reported by the California packing houses to be of good quality for bleaching purposes. For the past three years exploitation has been rife in several States, and deposits have been discovered in Colorada, Georgia, Indian Territory, Nebraska, New York, Pennsylvania, South Dakota and in new localities in Florida. These deposits, however, have not as yet attained any prominence as producers.

The composition of fullers earth varies greatly, and its value as a cleansing or clarifying agent can only be determined by actual trial—the chemical analysis serving merely as an indication of its value for these purposes. The physical properties of the earth also vary to a considerable extent. The color is brown, gray, to a dark blue and is probably due to the presence of organic matter. Its unctuous feel, and, when dry, its adherence to the tongue, resemble the general properties of clay. In fact, it may be regarded as of similar character, although it lacks plasticity, and has a high percentage of water. It has for its base a mineral of somewhat indefinite composition called smectite, which is, however, a hydrated silicate of alumina, very high in Al_2O_3. The occurrence of fullers earth is rather restricted. It is probably an alteration product formed like kaolin in the decay of rocks, and either found *in situ* as at Rosswein, in Saxony, where it is derived from gabbro or else redeposited in sedimentary series like clay as at Bath, England. Fullers earth is used for various purposes. Originally it was employed in England for fulling wool, whence its present name. In America but little is used for this purpose, the main consumption being utilized in clarifiying mineral and vegetable oils, a class of work to which it is even better suited than the English product. In using the earth for clarifying cotton seed oil, a small amount, 2 to 3%, is ground to a fineness of from 100 to 120 mesh, and is added to the hot oil with thorough agitation. On passing the hot liquid through a filter press the earth and the coloring impurities are removed, leaving the oil almost white in color. The fineness of the earth and the temperature during the time of treatment have an important bearing on its efficiency as a bleaching agent. The use of fullers earth for clarifying oils has practically superseded bone black.

Florida.—(By E. T. Cox.)—Fullers earth occurs in Florida in many counties. The deposits lie along the borders of living or dried up creeks whose bottoms furnish evidence that in past ages the water covered a much broader area than at present. Hills and ridges of sand on the banks of these creeks rise to an elevation of from 20 to 70 ft. The clay has been washed from them, and separated by a natural process of elutriation, the materials being deposited according to gravity. The beds are from 15 to 25 ft. thick, and have considerable sand in the upper and lower layers. The entire deposit rests upon Tertiary limestone. The mining operations are very simple. The overburden of sand is removed, and the earth is mined with pick and shovel. After careful assortment the product is dried at a high temperature in revolving cylinders and ground to various degrees of fineness. It is estimated that fullers earth in Florida can be mined, sacked and prepared for shipment at a cost of $4 per ton or less, f. o. b. at the mines.

A new use for fullers earth suggests itself, which is based on the property possessed by clay, of removing calcium carbonate from spring water. In localities where the water for boiler supply is heavily charged with lime the addition of a small amount of the earth will remove the lime salts, and thereby prevent boiler incrustations. An analysis of Florida fullers earth from Gadsden County gave 60·53% SiO_2; 20·13% Al_2O_3; 2·57% CaO; 2·12% MgO; 14·75% water and loss.

Imports.—The amount of fullers earth imported in 1899 was; lump earth, 3,803 metric tons (3,743 long tons), valued at $23,194, and ground earth, 6,682 metric tons (6,577 long tons), valued at $46,446. The importations for 1898 were, lump earth, 1,866 metric tons (1,837 long tons), valued at $15,921; and ground earth, 5,400 metric tons (5,315 long tons), valued at $65,123. The duty under the Dingley Tariff bill is $1·50 per ton for lump earth, and $3 per ton for ground earth.

American Market.—During the early part of the year there was a fair demand for small lots to cover immediate wants. The tone of the market was steady, and quotations were maintained at 80 to 90c. per cwt. for lump in bags or barrels, 82·5 to 85c. for powdered earth in bags and 87·5 to 90c. for powdered earth in barrels, depending on quantity. In the latter part of March the prices dropped to 75 to 80c. for lump, and 75 to 90c. for powdered. These figures remained constant for the balance of the year.

*India.**—There is a considerable amount of fullers earth mined for domestic use in cleansing hair, dyeing cloth and weighting paper. The principal mines are at Jeysalmir, Rajputana. The clay is found in layers, and is carefully mined and used without further preparation. The Bikanir mines in the same district produced 37,723 tons in 1895, valued at 15,332 rupees. No returns were made for 1897. The quarry here is 36,000 ft. long, 18,000 ft. wide, and 8 ft. deep. It has been worked for a long time, and its large extent offers an indefinite period for future development. At present the mining is of the crudest type, the natives removing the earth with picks. In Manipur a clay of similar nature is spoken of as edible.

* British Government Report, *Review of the Mineral Production of India* for 1897.

GARNET.

THE production of garnet in the United States in 1899 was 2,327 metric tons (2,565 short tons) $72,672, against 2,615 metric tons (2,882 short tons) $82,930 in 1898. The North Creek mines in the Adirondacks furnished 1,656 tons against 1,686 tons in the previous year, the remainder in each year coming from Chelsea, Pa., and Roxbury, Conn.

There have been no new sources of supply developed in the past year, and the Adirondack and Connecticut mines continue to produce practically all of this mineral used in the wood-working and shoe trades. Some African garnet was imported, but upon being given a working trial proved of no value. While garnets are widely distributed, it is only in the Adirondack region that the mineral is characterized by the proper degree of hardness, brittleness, cleavage and cutting qualities required for the various wood-working trades. Considerable of this stock is being used in the shoe trade, although formerly the Connecticut mineral has been in general use for this purpose. The garnet from the Carolinas resembles that from the Adirondacks in physical appearance but proves valueless in actual trial. The value of garnet for abrasive purposes consists in the proper combination of hardness, brittleness and cleavage. In use the grains are affixed to paper, and while they should possess hardness for ease in abrading substances against which they are rubbed, at the same time they should be brittle so that the wearing of the grain will take place by the continual breaking off of small particles from each individual grain, ever presenting fresh and sharp cutting edges so essential for effective work. Should the garnet lack brittleness, the original sharp edges of the grains will rapidly wear smooth and lose their cutting quality.

The New York mines are owned by two companies who supply the somewhat limited market by a few months' operations each summer. The output could be increased with ease, but the demand for this mineral for an abrasive, inferior to carborundum, corundum and emery, although superior to quartz, is so limited that a greater production would lower prices to a point where mining would no longer be profitable.

The mineralogical character and the probable geological history of the Adirondack deposits were given in THE MINERAL INDUSTRY, Vol. VI.

As a ton of garnet will cover a great many sheets of sand paper, this industry has not increased in tonnage to the extent that might be expected from the general activity in all branches of trade. There is, however, a healthier and stronger tone due to the increased business of the furniture, carriage, shoe and agricultural implement trades, and the increased foreign trade in these lines will doubtless lend stability to the market for some time to come.

GEMS AND PRECIOUS STONES.

THERE was nothing new in the mining of precious stones in America during 1899. The glacial deposits of Wisconsin which have been found to contain diamonds have excited some interest from a purely scientific standpoint. Some prospecting has been done on them which has resulted in no new discoveries. The New Sapphire Mining Co., Ltd., continued to work the sapphire deposits of Yogo, Mont., and an attempt was made to consolidate the various turquoise interests of New Mexico.

Diamonds, Brazil.—(By J. C. Branner).—The diamonds of Brazil consist of brilliants and *carbonados* or black diamonds.

From the discovery of diamonds in Brazil in 1772 until the discovery of the South African diamond fields in 1865, the bulk of the world's supply came from there, but the development of the African mines practically abolished the industry. A considerable quantity of the Brazilian stones still appear in the market, but diamond mining is no longer carried on upon a large scale.

The diamonds are not of widespread occurrence, but are confined, so far as our present knowledge goes, to comparatively small districts within the States of Bahia, Mines Geraes, Goyaz and Matto Grosso. They occur for the most part in placers. Some diamond bearing deposits, however, are not stream gravels, but consist of the weathered and decomposed remains of metamorphic rocks. The stones are frequently found held fast in a conglomerate or pudding stone cemented by iron oxide. The beds of existing streams have furnished the greater part of the production.

Mining is carried on upon a small scale as a rule. The methods in use for diverting the streams, handling the gravel and separating the diamonds are of the same character as those employed in the early days of diamond washing. Opinions differ as to the development of the industry by modern methods and machinery. My own observations led me to conclude that modern machinery and methods can be made to pay only in case of the discovery of very large diamond bearing deposits, where separations can be carried on for a long time in a comparatively small area. Brazil is without good roads and without skilled labor, and machinery therefore is expensive to operate or to move readily from place to place. It has been possible to operate the diamond mines of South Africa by means of machinery because the deposits are very limited in area. In Brazil, on the other hand, the diamond bearing gravels spread over thousands

of square miles, and are only locally profitable. The *carbonados*, or black diamonds, are mined in the State of Bahia, and the output passes through the hands of a single firm in the city of Bahia. The finest of these *carbonados* come from Saloboro, in the Serra Itaraca, near Cannavieiras, in the southern part of the State of Bahia, and from Lencoes, Sincora and Santo Ignacio, 306 km. west and a little south of Bahia. There are mines in other parts of the State, some located within ten leagues of the city of Bahia. It is an interesting fact that the rich Salobro district, 70 km. from Cannavieiras, was only discovered as recently as 1881. In 1890 one mine in this district yielded 54,000 oitavos* or drachms of diamonds. The diamond bearing gravel is from 1 to 2 m. thick. The stream beds have not been mined.

The statistics of diamond production are necessarily difficult to collect. Even while the mines belonged to the Crown of Portugal, and all stones coming from them were supposed to pass through the hands of the *intendente* of the district, large numbers went illicitly into the European markets as contraband. The best available statistics show that the total output of Brazil from the discovery of diamonds up to, and including 1899, was 13,105,000 carats.

According to Ernest von Hesse-Wartegg, diamonds were discovered in ploughing, 19 miles to the S.E. of Itschoufu in the Province of Shantung, China. Some years ago United States Consul Fowler, in a statement to the Government,† reported that a correspondent of his considered these fields well worthy of intelligent exploitation. The natives believe, as they frequently find the diamonds after a rainstorm, that they are produced by the action of the rain itself, and will not wash for them, although they are found for some 8 miles in extent on a low sandy ridge.

The report of the De Beers Consolidated Mines, Ltd., of Cape Colony, for the year ending June 30, 1899, shows that the diamonds produced during that period were marketed for £4,038,421. The total expenditure amounting to £2,063,714, left a profit of £1,974,707, exclusive of other revenues. The average yield per load for the De Beers and Kimberley mines was 0·7 carats. The value per carat was 29s. 7·2d. The yield at the Premier mine (Wesselton) was 0·3 carats per load, and the value 22s. 10·1d per carat. There were 4,034,716 loads of blue ground and lump on the floors June 30, 1899. The contract with the syndicate for the disposal of their product expired April 1st of the current year, but it is anticipated that another one will be entered into on even more favorable terms. According to Cecil Rhodes, the holding of the De Beers Company in Chartered shares and in the Mashona land railway had been disposed of to the advantage of the company. The assets of the London & South African Exploration Co., which included the Dutoitspan and Bulfontein mines, had been purchased, as well as the properties of the Kimberley Diamond Mining Co. for a total consideration of £1,725,000. A dynamite factory is in course of erection, which it is anticipated will add to the company's products.

Otto's Kopje Mine, Cape Colony, seemed on the eve of success when the war broke out. It was stated on the relief of Kimberley that the machinery was still in good order, and that work could be resumed quickly.

* Eight oitavos equal one ounce troy. † Consular Report No. 198, March, 1897, p. 384.

It is said that diamonds have been found in an extinct crater in the Witries Hock Mountains in Natal. Diamonds were discovered about 1872 in the stanniferous alluvial near Inverell, New South Wales. While the production has never been large, a maximum quantity of 20,000 carats having been obtained at Boggy Camp, there are possibilities of an increased output on systematic prospecting work.. The diamond bearing wash is covered by basalt, and in places the wash rests on very soft decomposed granite. Overlying the wash in various places, more especially where the basalt comes in contact with the wash, is a cemented gravel in which diamonds have been found imbedded. The opinion of H. N. Porter* is that this was an overflow of lava. The writer believes that an extinct crater in what he terms "carboniferous mudstone" was the center of volcanic activity, and the source of the diamonds found in the wash. The periodical flooding of the low lying crater doubtless prevented the consumption of many of the diamonds, which later would be expelled in the period of activity. The basalt itself he regards as a later flow.

Notwithstanding Mr. Porter's opinion, the Government Geologist does not believe that the original matrix of the diamonds was an eruptive rock. Samples of the alleged volcanic breccia from Bingara were pronounced to be of sedimentary origin. Nor in Mr. Jacquet's opinion was there anything found to suggest the occurrence of a volcanic pipe. A report furnished by Messrs. Thos. Davies, F.G.S., and R. Etheridge, Jr., refers to the quality of New South Wales diamonds as follows: (1) That the diamonds in New South Wales in their physical characters are more nearly allied with those of Brazil than any other country; (2) they have been largely sold in London as such; (3) as regards color they differ practically but little from those of other fields; (4) the general absence of cleavage and macles is a point in their favor; (5) that the greater hardness of New South Wales gems will probably raise the cost of cutting, but this will be compensated for by their extra brilliancy. The diamonds found in this Colony generally average from 5 to 6 to the caret, although gems of 2 to 2·5 carats are occasionally found, and one of 5·62 carats has been recorded. The number obtained per load varies greatly. The Round Mount Co. in 1886 washed 722 loads for 2,685 carats, and from six loads obtained an exceptional yield of 1,080 stones weighing 296 carats, which probably establishes a record as far as New South Wales is concerned.

The Lace Diamond Mining Co., Ltd., formed in 1899, worked the Farm Ruby, near Kroonstad, Orange Free State, which consists of 1,385 morgen, on which 400 claims at about the center of the farm have been taken up. The formation is said to be somewhat different from anything which has yet been found in South Africa. A portion of it was capped by basalt, which was considered an unfavorable feature, though later explorations have shown the occurrence of gems of very good weight and quality in the vicinity of this rock. This overflow, however, is by no means general and is considered to be float. The yellow ground was worked by the De Beers Consolidated mines, which held an option on the purchase of the property, and gave the following results: 5,683 loads yielded 652 carats, equal to about 8·5 carats to the 100 loads. The tailings showed a

* *Transactions of the Institute of Mining and Metallurgy*, Vol. VI.

further 13%, and, owing to the very crude machinery, it was anticipated that the re-treated tailings would show an equal amount. Since the company was formed blue ground was struck at the depth of 50 to 75 ft., though it has not yet been washed. Latest results show an average of about 16 carats per 100 loads of diamonds of fair quality. It was thought that when arrangements have been made for proper supervision of the boys that this yield will be increased, as at the present time they have unlimited opportunity for abstracting the largest stones. The mine is well situated for working, as it is 5 miles only from the Campbell coal mine, from which coal can be delivered at 15s. a ton. Water is abundant.

The chimney at the Monastery Farm mine in the Orange Free State, 30 miles south, in Winberg, is rather small, occupying from 5 to 12 acres. Blue ground was struck at the depth of 81 ft., and resembles in appearance and its mineral constituents the country rock of Kimberley.

The Jagersfontein mine reached its maximum production in July, during which month 22,219·75 carats, worth £45,908, were produced. The value per carat was extremely high.

The total production of the Transvaal, according to Charles E. Macrum, late United States Consul to Pretoria, was 22,843 carats, valued at $212,812·04, of which 12,283 carats of a value of $171,437·06, came from alluvial mines, while from the dikes 10,560 carats, valued at $41,374·98, were marketed. The river stones are generally larger and of better quality than those from the diamondiferous dikes. The average value of the diamonds found at Reitfontein is $3·89 a caret, which is much less than the average value of Kimberley diamonds, and less than half the value of those from Jagersfontein. The diamond bearing area of the Transvaal seems to be quite large.

What may prove to be an important discovery was made at Hatherley, a few miles from Pretoria, where true diamondiferous ground is found 15 ft. from the surface. In general it may be said that much of the country surrounding Pretoria gives evidence of being diamond bearing.

A peculiar feature of the laws formulated in 1898 regulating the mining of diamonds and other precious stones in the Transvaal, was the limitation of the number of employees; not more than four white men being allowed to each owner. A license of £10 per head was imposed on those who intend to open up Government ground, and it was made obligatory to any one discovering precious stones to give notice to the Government within 24 hours under severe penalty. Stringent clauses were also included in the act against illicit diamond buying.

Claims have been made from time to time that diamonds existed in West Australia, and one company has been formed to work the alleged deposits. It went into liquidation the early part of 1899 without the shareholders or receiver being able to obtain much information.

J. Milne Curran* gives some practical hints on the identification of diamonds. While the article does not treat the matter so thoroughly as Mr. Leopold Claremont's contribution to THE MINERAL INDUSTRY, Vol. VII., the notes may be of benefit to prospectors to whom laboratory appliances are inaccessible.

* *Journal of the Royal Society of New South Wales*, 1897.

Zircon is commonly mistaken for diamonds, owing to the surface luster of these stones being often finer than that of a diamond. The test of cutting glass is frequently, though erroneously, relied upon, as a zircon used to advantage will scratch glass easily, while a true diamond will not mark glass if a rounded face is drawn along the latter. In a diamond bearing country the prospector should carry a small plate of sapphire and a splint of diamond mounted on the end of a short length of stout brass wire. The latter is known as a writing diamond and is inexpensive.

The supposed diamond is set in the end of a stick of sealing wax, and while the wax is softened by heat the stone can be so arranged that a sharp solid angle or an edge projects. This can be done after a little practice by heating the stone, and quickly transferring it to the wax. Holding the stick of sealing wax, the stone to be tested is rubbed with a gentle pressure round and round, on the polished face of the sapphire plate. Not more of the sapphire need be used than the space covered by a pin's head. After the friction is continued for half a minute, if the sapphire plate has lost its polish, and with a lens shows the point of contact "eaten into" or "burned," the stone being tested is a diamond, since the diamond is the only stone that can cut into a polished plate of sapphire.

The use of a writing diamond will, however, settle the matter at once. If a prospector draws a diamond over a smooth zircon and over an uncut diamond, the difference is apparent. Even with gentle pressure the writing diamond "catches" or "drags" on the zircon, and a dull cutting noise is made. When the writing diamond is gently rubbed on a real diamond, it (the writing diamond) does not "bite," but glances and splits off in every direction, no friction due to cutting being apparent, and the sound produced is metallic-like and sharp.

It may be well to remember that where zircons are found a considerable number of the stones are red, or some shade of red, and on heating these colored stones become white and remain white on cooling. Red or green diamonds are exceedingly rare. These rough-and-ready tests will also distinguish between small and water-worn topaz and diamond stones which are also confused by prospectors.

C. Margot proposes to distinguish real diamonds from the artificial by the property aluminum when in pencil form has of marking silicious substances, but not the true diamond. The surface of the gem under examination must be cleansed thoroughly, and freed from grease.

In the thirteenth annual report of the United States Commissioner of Labor* there is a comparison between the cost of cutting four three-carat diamonds by hand and by machinery. The cost by machine work was considerably higher than by hand, but time was considerably shortened. Thirty-nine hours' labor cut three four-carat diamonds when machines were employed, but 132 hours' hand labor was required. The cost, however, was $26·25 against $14·84.

Emeralds.—Emeralds are found in various parts of the Republic of Colombia. The famous mines of Muzo, near Bogota, which were worked before the Conquest (the uncut gems supplying a large part of the booty of the sixteenth-century pirates), have been continuously worked under Government leases since the year

* For the year 1898, p. 1288.

1568. No statistics of the output are available. The emeralds are found either in isolated crystals or together with uncrystallized corundum, imbedded in a mountain-side of limestone. The method of working is cutting away the barren surface-rock and then opening reservoirs to sluice off the débris until the limestone is laid bare, when the stones are broken out by expert laborers.*

There was considerable excitement in Bogota during 1899, owing to a local boom in the price of emeralds, although the mines in the vicinity of that city, according to the English company which owns them are unproductive. There was a considerable quantity of uncut gems held by various people. The excitement rose to a great pitch, and these stocks were brought out. The prices mounted rapidly and sales were made at from three to four times the former value of the gems. The craze, however, subsided suddenly, and many unfortunate purchasers were left with stones which they had purchased at absurdly high prices.

The English company's lease of the emerald lands expires during the present year, and as yet no arrangements have been made for its continuation.

The most important occurrence of this gem in New South Wales is at the Emerald Proprietary Co.'s mine at Emmaville. Emeralds were found in the tin drift for many years, but in 1890 they were discovered *in situ*. At this mine emeralds were found on the surface in disintegrated feldspathic material, slate rock and quartz, not very far from the slate and granite. On opening the ground it was seen that a reef or vein of silicious materials occurred in slate-rock or claystone. The reef travels or trends in a northeastern direction, dipping at a high angle to the southeast. At various points, proved to a depth of 100 ft., shoots or pockets occur, containing quartz, topaz, tinstone, arsenical pyrites, fluorspar and kaolin. The shoots almost always carry emerald or beryl associated with the minerals named. Again some shoots consist for the most part of emerald and beryl only.

The greater bulk of the marketable stone did not cut to a greater size than one caret. Some gems realized as much as £10 per caret; the ordinary ones bring much less, being rather light in color and resembling aquamarine. An analysis of a specimen by Mr. James Petrie showed H_2O, 0·62%; SiO_2, 65·20%; Al_2O_3, 17·80%; BeO, 14·40%; CaO, 1·0%; MgO, 0·64%; Na_2O, 0·34%. These mines are now shut down, as the hardened formation encountered in depth does not contain the same quantity nor the same quality of emeralds as was found in the more decomposed rocks.

A considerable quantity of emeralds has been found along the banks of the Tokova River, 50 miles from Ekaterineburg in Russia. Those of superior quality are discovered on the surface, while those found deeper are inferior.

Opal.—The White Cliffs Opal Mining Co., Ltd., still controls the opal region in the vicinity of Wilcannia, New South Wales. The company has found it advisable to increase the percentage to its tributors from 50 to 75%. It is expected that when the percentage was based on the lower rate much of the most valuable mineral was disposed of privately by the tributors. The company has not been able to work satisfactorily on the wages system, and this was finally abandoned. Working here will always be expensive, as there is a lack of water,

* Grainger and Treville, *Transactions of the American Institute of Mining Engineers*, Vol. XXVII.

and the local sanitary conditions are extremely bad. The Paranga opal fields, a few miles distant from White Cliffs, have been practically abandoned. About 350 men are employed on the leases of the White Cliffs Opal Co., Ltd. A considerable number of men are working on abandoned ground, and are there working to as deep as 50 ft. Good opal has been found at 40 and even 50 ft. Local prices have varied according to quality as well as market demand. In the case of special samples $100 an ounce has been obtained.

Here the opal is found in Upper Cretaceous Sandstone, lining joints and filling fissures both vertical and horizontal. The matrix is so friable that it is easily removed, and the gem released. Opal has been found in a strip of country nearly two miles wide and 12 to 16 miles long. In this area miners blindly sink shafts with the hope of finding opal seams. Experience has shown, however, that the probabilities of finding good stone are confined to certain localities that show no especially distinctive features. The opal is not confined to any single level, though appearance would indicate that the conditions under which the opal is found, ceases at a depth of from 50 to 60 ft. from the surface. The matrix is feldspathic and much of it contains sufficient lime carbonate to effervesce with cold hydrochloric acid, and may be classified as a marl.

An extremely large block of opal was found in the earlier part of the year at Opaltown, Queensland. It was 15 in. wide and 11 ft. long. It is stated that $60,000 was offered for it.

Rubies and Sapphires.—The Burma Ruby Mines, Ltd., managed for the first time in its history to pay a dividend of 5%, though their richest mine at Shebontha was drowned out. The books showed a total profit for the year ending February 28, 1899, of £4,322, which was increased to £9,893 by a remission of royalties amounting to 2 lakhs of rupees. The Indian Government has determined to remit the 4 lakhs of arrears still due under the original lease, and to reduce the yearly net rental for a term of years from Rs. 3,15,000 to Rs. 2,00,000, subject to an increase in the Government share of the net profit from 20 to 30%. The agreement is to last for five years and is subject to reconsideration at the end of that term. The mines have been worked by electricity since September 5, 1898. The installation which has reduced working expenses by £500 a month, will shortly be doubled in size. The mine showed a mining profit of £11,470 as compared with a loss of £360 for the same period of 1898.

Sapphires of every known shade from white to royal deep blue, have been found in New South Wales. At Tumberumba it is obtained chiefly in working recent alluvial deposits for gold. Associated with the sapphires there is spinel, topaz, andalusite, and garnet. The spinels are of excellent size and color. Sapphire is also found throughout the alluvial drifts of the tin districts of New England. While sapphires are abundant the production of good gem stones is exceedingly small, and when compared with good gems the New England stones show a want of life.

J. Milne Curran believes that basalt is the original matrix of sapphire, as basaltic areas can always be found in the vicinity of localities where sapphires occur.

H. Warington Smyth, F. G. S.,* describes the ruby and sapphire deposits of Siam. The workings are partly alluvial in river beds, and partly by drifting in layers of a decomposed basalt rock, of which there are frequently two. Basalt is the general country rock, though no gems have been found in it when in an unaltered condition. Poor sapphires, corundum, topaz, zircon, and ilmenite, and at some points garnets occur in the ruby layers. On the upper Mekong River sapphires are found but no rubies of any account. Here also the gems are found in stream beds, and in a definite layer 12 to 20 ft. below the surface.

The ruby and sapphire mines at Pailin, in the Battombong Province of Siam, are held by the Siam Exploring Co., Ltd., one of the affiliated companies of the Siam Co., Ltd. The mines are worked by Shans and Laos, licensed by the company, who are allowed to dispose of the stones themselves. The prices obtained for the stones were poor, but recently have improved.

Turquoise.—Mr. Gustav Eisen, of the California Academy of Sciences, describes the result of archæological investigations of the turquoise mines near Manvel, San Bernardino Co., Cal. The best mines are found in an area of 15 miles long by 3 miles wide. The region is covered with volcanic overflows from a central group of craters. These overflows extend for many miles in all directions. Between the basaltic ridges are found small rounded hills of decomposed rocks traversed at times by dikes of crystalline rocks and quartzites. On the sides of these are pits from 15 to 30 ft. across, and of half that depth. Prospectors have found good stones of good color and quality in these half filled excavations, while ordinary ones are common.

Caverns which have been the habitations of some prehistoric race, indicate that these mines were actively worked at one time. Hammers and axes made from hard basalt or trap, are frequently found. Some of the rock carvings indicate that these mines were known to the Aztecs, but others are unlike any previously described. It is believed, and the conjecture is supported by existing traditions, among the Mojave Indians, that these mines were worked by a tribe which came from the south and had communication with ancient Mexico.

While turquoise has been found widely distributed through Southern California and Nevada, no stones of extreme value have been discovered as yet, though one weighed 107 carats and another 64·5. They were from a point about 18 miles east of Vanderbilt, Cal., and in Nevada. The turquoise is found in what is described as a trachyte or white soft conglomerate through which blue-green streaks and veins pass, here and there expanding into nodules.

Turquoise was discovered in 1894 near Bodalla, New South Wales. Much of the stone is found to be porous and not as compact as it should be. The green tint of most of the stone is objectionable also. A few picked stones are, however, of excellent quality. According to J. Milne Curran it occurs in this locality as (1) thin crust-like seams filling horizontal joints in the slate; (2) in rounded marble-like balls in vugs of the slate; (3) as concretionary matter in similar vugs; (4) in thin lenticular plates in black streaks associated with iron pyrites. The slate rock is dark in color, and is charged more or less with pyrites containing 2 dwt. of gold per ton. An analysis showed H_2O, 21%; SiO_2, 0·5%; CuO, 7·45%; Al_2O_2, 36·23%; Fe_2O_3, 1·26%; CaO, 1·7%; B_2O_5, 31·9%.

* *Five Years in Siam*, 2 volumes, London, John Murray, 1898.

The Cutting and Polishing of Precious Stones.

By Leopold Claremont.

Precious stones were originally used as ornaments in their natural or uncut state by the ancient races and were valued more for amulets and their supposed magic and medicinal qualities than for their beauty. Subsequently the natural crystallographic faces of the gems were roughly polished to enhance the luster and for centuries later they were treated in this way and used for the embellishment of goblets, tankards, idols, thrones, etc., as well as for personal adornment.

As time went on the ancients acquired the means of increasing the number of faces found on many gems, and in this crude way gems were not only polished but received additional faces which are now called facets.

From specimens of precious stones thus crudely cut it is apparent that the one object of the ancient lapidaries was to retain the size of the gems upon which they operated. Therefore no matter in what shape they were found or in what degree they were flawed or otherwise faulty, they were merely covered with polished surfaces without regard to symmetry, brilliancy or quality of color. The gems also were generally drilled through the center in order that they might be strung, sewn on to a robe, or dangled from a pendant or turban. Not infrequently they were beautifully and elaborately carved—as is still done by the natives of many parts of Asia—and in this way many magnificent gems were almost ruined. When they are placed upon the gem market of to-day, which is by no means an uncommon occurrence, it is found that in order to remove the carving and drilling—work done at a gigantic expenditure of labor and energy—so great a loss of weight is incurred before the stones can be made suitable to the requirements of modern jewelry, that they are of considerably less value than rough stones of the same quality and weight.

There is no record of precious stones being cut with any regular system until as late as 1285, when a record occurs of a guild of gem polishers and cutters established in Paris. Records of diamond polishing toward the end of the fifteenth century have been found.

Gem cutting was also an established industry at Nuremberg in 1370, where the lapidaries were termed "table polishers" and gained some distinction in their craft. It was not, however, until nearly 100 years later, 1456, that diamonds were cut in a regular system of faceting.

A Frenchman named Louis de Berquem, a resident of Bruges, was the first to discover a method of so arranging the facets upon a diamond that the reflection and refraction of the light were so greatly increased that he soon became famous throughout Europe. A guild of diamond cutters and lapidaries was established in Bruges in 1470, and many large diamonds which have figured conspicuously in history were entrusted to Louis de Berquem to recut.

The pupils of Louis de Berquem dispersed to different cities, and formed new fields for the industry. Amsterdam, Antwerp and Paris became the centers of the diamond polishing industry and the cutting of colored gems flourished in Lisbon. In Paris the art for a time—notably under the influence of Cardinal Marzarin—progressed with rapid strides, owing doubtless to the reckless grandeur of the French Court. It gradually declined, however, and finally

disappeared from this city during the troubled times toward the end of the eighteenth century.

In Holland the diamond cutting industry took the firmest root, and to this day it remains the staple business of Antwerp and Amsterdam. Until the beginning of the seventeenth century all the colored gems of any note were cut in Lisbon, and the beautiful work of the old Portuguese lapidaries is still to be found in many family jewels. The trade, however, like many other artistic crafts, was entirely in the hands of the Jews, and when they were expelled from the country at the end of the sixteenth century they betook themselves to other cities. Most of them settled in London, and their immigration into England has caused the finest colored stones to be cut in London, and Hatton Garden to be the principal gem market to-day.

There is little record to be found of the methods employed in bygone times for polishing and cutting precious stones, but as the tools and machinery used

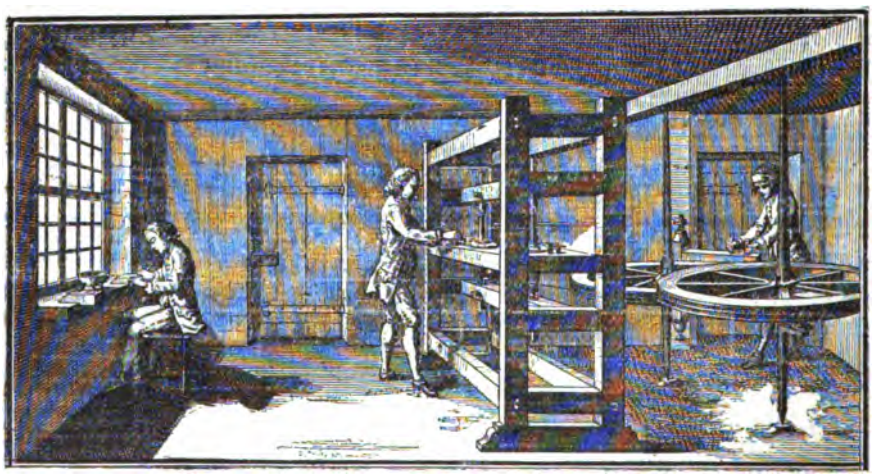


FIG. 1.—OLD METHOD OF CUTTING AND POLISHING GEMS.

by modern gem cutters are of the simplest character, it is probable that little change has taken place in the process with the exception of the introduction of motive power; at all events, during the last 100 years. The method of polishing diamonds before the introduction of steam is shown in Fig. 1, a description of which will be found unnecessary.

The art of the modern gem cutter consists in so manipulating the rough stone as to produce a finished gem of the greatest value. With this end in view he first studies the rough stone, carefully noticing any flaws or other imperfections, which it is his aim to remove, either entirely or partially; or, in the event of this being impossible, to hide as far as lies in his power. At the same time he bears in mind that the desired proportion and as much weight as possible is retained; and in the case of colored stones, the color must be reduced, retained, or adjusted according to the requirements of each individual stone upon which he works. When it is remembered that in nature no two gems occur identical

FIG. 2.—"BRUTING" AND "CLEAVING" DIAMONDS.

FIG. 3.—PREPARING THE HOLDER OF SEMI-MOLTEN METAL FOR EACH STONE TO BE CUT.

FIG. 4.—POLISHING DIAMONDS.

FIG. 11.—POLISHING TURQUOISES.

in every respect, it is obvious that the work of the lapidary is purely and simply one of skill, practice and judgment as the tools and apparatus with which the work is done are very simple. The manipulation of every stone requires brain work, and for this reason machines for cutting and polishing gems are utterly useless.

The method of cutting diamonds is quite different from that of other precious stones. It consists of three processes: "bruting," "cleaving" and "polishing."

Bruting consists in rubbing two diamonds together in such a way that by continual friction each can be made to assume a desired shape. (See Fig. 2.) Each diamond is cemented to one end of a stick or holder about 1 ft. in length, and one is held in each hand of the operator. The stones are then pressed one against the other with a rubbing motion over a trough containing a very fine sieve for catching the dust produced by the rubbing. The holders rest against small projections at the sides of the trough and considerable leverage is brought to bear upon the stones. The dust, called "diamond powder," is collected in a box below the sieve and is used in other parts of the process as a polishing medium. By these means the diamonds are modeled into the required shape and they leave the hands of the bruter shaped in the form in which they are destined to be used as gems minus the flat polished surfaces known as facets, which are subsequently formed by polishing.

It is sometimes necessary to divide a diamond into halves or to remove a small part from a stone which can afterward be cut into a gem as well as the larger piece from which it has been severed. When this is to be done advantage is taken of the natural tendency of the diamond to divide along certain directions (parallel to the faces of the octahedron) and the operation of "cleaving" takes place.

The cleavage of diamonds is a responsible and difficult task and requires a very intimate knowledge of the crystallographic forms in which the stones occur. It is performed in the following way. (Fig. 2.) Upon the end of a wooden holder the diamond is cemented in such a position that the plane of cleavage to be used in the operation is parallel to the length of the holder. The holder is held firmly in position by being fixed into the center of a leaden weight which projects from the bench in front of the operator. The edge of a steel blade somewhat resembling a razor is held upon the diamond with the left hand and with the right hand a sharp blow with a slender rod is then delivered upon the back of the blade. The diamond divides along the plane of cleavage at a single blow and the operation is at an end in a few moments. While this is apparently quite easily performed, in reality it requires the greatest amount of skill. When the diamonds leave the hands of the bruter they are handed to an attendant who is seated at a bench in front of two flaring argand burners. (Fig. 3.) Small brass cups called *"dops"* ranging from the size of a tea cup to that of an egg cup are placed in the flames and a mixture of two parts of lead to one part of tin is placed in them. This soft metal quickly becomes semi-molten and by means of a pair of soft iron tongs the contents of each cup is fashioned into the shape of a cone, upon the apex of which a gem is embedded. Each iron cup filled with soft metal thus forms a holder for the

diamond which is now ready for polishing. From time to time, however, it is returned to the solderer for readjustment, as it is only possible to work upon the small part of a stone projecting from the metal. It is therefore necessary to remelt the metal and replace the stone with a different part of it projecting, until the polishing of the gem is finished.

One man working diligently should be able to solder for six polishers.

The polishing process is an arduous and delicate one. (Fig. 4.) The craftsmen are seated in front of revolving cast-iron wheels, very porous, which turn in a horizontal position at the rate of 2,000 revolutions per minute. The metal cup containing the diamond is held firmly against the surface of the wheel with a heavily weighted iron clamp. Each operator is able to manipulate four of these clamps at once, first examining one diamond and then another, occasionally plunging them into cold water to prevent friction heat from unsol-

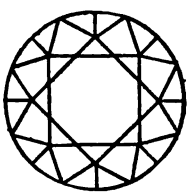


FIG. 5.

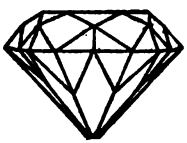


FIG. 6.

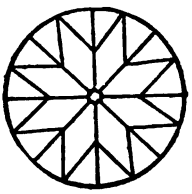


FIG. 7.

FIGS. 5, 6 AND 7.—TOP, SIDE AND BACK VIEWS OF THE "BRILLIANT" CUTTING.

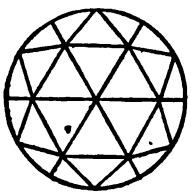


FIG. 8.

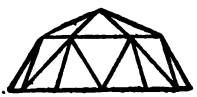


FIG. 9.

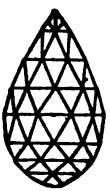


FIG. 10.

FIGS. 8 AND 9.—TOP AND SIDE VIEWS OF THE "ROSE" CUTTING.

FIG. 10.—VIEW OF THE "BRIOLETTE" CUTTING.

dering the stone, which would damage it considerably. Diamond dust which has been prepared for the purpose is mixed with a little olive oil and smeared upon the wheels. This together with the dust which comes from the gems themselves, gets into the pores of the cast-iron wheels and forms an erosive surface which polishes the facets of the diamond. As the metal cups are frequently too hot to be handled with comfort, wooden holders are used to carry them.

There are several forms in which diamonds are cut (see Figs. 5 to 9), and a short description of each is as follows: The "Brilliant" is the usual form, shown in Fig. 5. In the center is a large octagonal facet termed the "Table" surrounded by 32 smaller facets which extended from the table to the edge of the stone, comprising what is known as the "Girdle." Fig. 7 shows the back of a "brilliant" cut diamond shaped to a pyramid, the top of which forms a small

facet known as the "Culette" or "Culet," and there are 24 facets reaching from the Culet to the girdle. Fig. 6 shows the side view of a brilliant.

The "Rose" (Figs. 8 and 9) is another form, used only for small thin stones which would not make brilliants. It consists of a number of triangular facets so arranged as to cover the front of the stone completely. The back of the rose is quite flat and not worked into an apex. The "Briolette" (Fig. 10) is pear-shaped and has no girdle, table or culet, but is covered with facets like those on the front of a rose. They are generally used for pendants.

The working of all other precious stones is very different from that of the diamond. The gems are cut upon a copper disk into the surface of which diamond powder is continually pressed. The operator holds in the right hand a slender holder of ebony or ivory about the size of a pen-holder, upon the end of which the gem is cemented. A conical rest is used to steady the other end of the holder, thus causing the stone to press upon the wheel at any desired angle. The wheel is rotated by a crank operated by the left hand.

When the gem leaves the cutter every thought and judgment necessary for the successful development of a gem from the rough stone has been bestowed upon it, and while all the facets have been accurately placed thereon, it is dull and lusterless and has to be polished.

It is the work of the polisher to apply carefully every cut facet to the surface of a revolving wheel upon which is spread rotten stone or other polishing material. The wheels are made of various metals, copper, tin, lead, gun-metal, bell-metal, brass, pewter, etc., to suit the specific hardness of the gems and an electric motor is used as power. Gems with a smooth convex surface are cut upon a lead wheel upon which emery dust is spread. They are subsequently polished on a metal wheel, or, if they are very soft—as opal or turquoise—wheels are used composed of wood, flannel, leather and silk. These wheels vary in size from 2 to 15 in. in diameter; they revolve vertically and are adjustable to a lathe head. Fig. 11 shows the process of polishing turquoises in this way.

For carving gems very small disks and wheels the extreme edges of which are primed with diamond powder, are used in a lathe, and to divide a stone a small tin disk as thin as paper is used, on the principle of the circular saw. The description of the process of cutting and polishing precious stones sounds very simple, and it even appears simple to a casual observer watching it in progress, but the amount of skill, dexterity and judgment required by a successful gem cutter is very great. A gem of almost priceless value might be completely ruined in a careless or thoughtless moment.

It is greatly to be regretted that thousands of magnificent gem-stones are still cut by ignorant persons without any regard whatever to the study of optics, physics, mineralogy or crystallography, which is quite necessary for the production of high-class lapidary work. It pays to recut stones which have been thus badly treated, for the loss of weight necessarily incurred in the operation is amply repaid by the increased brilliancy secured.

GLASS.

By Robert Linton.

Physical and Chemical Properties.—Glass is the name applied to various fused mixtures of silicates of alkalies, alkaline earths and metals, in the forms in which they are manufactured into articles of commercial value. It is hard, brittle, of irregular fracture and in most cases transparent. The distinctive characteristic of its composition is the presence of at least two bases, one of which is an alkali. Soluble glass is a simple alkaline silicate, and differs so much from the ordinary glass of commerce in its properties and uses that it is omitted from the general classification of the various kinds of glass.

Whether glass is a compound silicate, or a mixture of simple silicates, cannot be ascertained with certainty, as the individual silicates cannot be separated from each other as such. Various phenomena met with in glass making seem to indicate that in melting glass the silicate of an alkali acts as a solvent in which the silicates of the metals, alkaline earths, etc., are held, and remain when the glass stiffens on cooling.* Other salts in varying quantities are occasionally combined with the silicates—namely: Phosphates, borates, fluorides, arseniates, antimoniates, stannates.

The following elements form the bases of the silicates which occur in ordinary colorless, transparent glass:

Alkalies: sodium, potassium, lithium; *alkaline earths:* calcium, barium, strontium; *metallic bases:* lead, thallium, zinc, magnesium, aluminum. Of these elements sodium, potassium, calcium and lead are the bases that form nearly all glass. It will be noted that there is always present an alkali and one of the alkaline earths or metals included in the above classification, which presents two silicates, one built upon a univalent base and one upon a bivalent base.

The analyses on the following page of glass and glassware from different sources show a variation in the chemical composition, due to manufacturing conditions:

* For fuller discussion of the probable structure of glass, see paper read by the writer before the Engineers Society of Western Pennsylvania, April 19, 1895.

Components.	Window Glass.					Plate Glass.				Lead Flint (Cut Glass).			
	American.	Belgian.	German.	English.	French.	American.	French.	Belgian.	English.	American.	Belgian.	French.	English.
	%	%	%	%	%	%	%	%	%	%	%	%	%
SiO_2	72·26	69·48	72·68	71·40	69·65	71·2	72·1	72·4	78·64	63·76	53·70	52·41	51·4
Al_2O_3, Fe_2O_3	1·42	2·59	1·06	1·90	1·82	1·0	.....	.....	2·68	0·90	1·07	0·96	.....
CaO	13·84	13·40	12·76	12·40	13·31	14·2	12·2	13·2	6·09	(MgO 0·31)	0·59	0·77	.....
MgO	.....	0·26	0·26	.....	.....	.....	.....	.....	.....	PbO —21·93	34·91	35·24	37·4
Na_2O	14·01	14·55	13·24	15·00	15·22	13·9	15·7	14·4	11·63	12·86	9·12	10·37	.....
K_2O	.....	.....	.....	.....	.....	.....	.....	.....	1·34		0·30	.....	9·4

Components.	Lime Flint (Prescriptl'n).				Green Bottles.			Champagne.	Jena.				Strass.
	American.	English.	German.	French.	American.	German.	French.	French.	Thermometer Glass.	Combustion Tube.	Verbundglas.	Chemical.	French.
	%	%	%	%	%	%	%	%	%	%	%	%	%
SiO_2	73·96	.....	75·61	69·6	69·82	60·4	68·34	61·9	67·80	66·61	71·95	74·07	38·2
Al_2O_3	0·44	.....	1·01	5·2	2·58	10·4	4·72	4·41	2·50	3·53	5·00	1·20	1·0
Fe_2O_3		.....				3·8	4·42	1·85					
CaO	13·94	.....	7·88	13·0	7·82	20·7	21·34	17·95	7·00	11·81	.....	8·25	.....
MgO	0·41	.....				0·6	.....	6·38	.....	.....	.....	.....	.....
PbO	0·18	.....	4·84	.....	.....	.....	.....	.....	.....	.....	.....	.....	58·0
K_2O	10·85	.....	11·39	8·0	1·50	.....	2·01	1·13	.....	.....	.....	7·47	7·8
Na_2O		.....	.....	3·0	18·28	13·2	4·17	6·16	14·00	6·56	11·00	8·63	.....
ZnO	.....	.....	.....	.....	.....	.....	.....	.....	7·00	1·30	.....	.....	.....
BaO	.....	.....	.....	.....	.....	.....	.....	.....	.....	2·20	.....	.....	.....
B_2O_3	.....	.....	.....	.....	.....	.....	.....	.....	2·00	7·99	12·00	.....	.....

Prof. Weber, of the Royal Polytechnic School at Berlin, made an extensive investigation to determine the best glass for resisting the action of air, moisture, acids and other decomposing agents.* From the results of his experiments he established the formula for the best or "normal" glass to be:

$R'O$, $R''O$, $6SiO_2$, in which $R'O$=alkaline base, $R''O$=metallic or alkaline-earth base, and SiO_2=acid.

It will be found profitable to bear this formula in mind as a check against the use of any one material in excess. Of course other considerations, besides chemical stability and resistance to decomposing influences, must be observed in actual practice—clarity, adaptability to working the molten glass into the finished ware, brilliancy, color, etc.

The use of the different materials in excess affects the quality of the glass in various ways. Excess of alkali causes glass to fade when exposed to the influence of the atmosphere, especially when stored in a damp place. Ugly spots will appear on the surface, showing a smoky appearance when examined by transmitted light and iridescent when the light is reflected. As the alkaline silicate which is in excess, is soluble in water, it is dissolved in part from the glass, and leaves the surface scaly and rough, instead of a brilliant enamel. Too much alkali tends to make the glass cordy. Excess of lime, on the other hand, prevents cords in the glass, but greatly increases the tendency to devitrify. Glass with a high proportion of lime is hard and brittle and requires more labor in working and greater care in annealing than glass with

* *Handbuch der Glasfabrikation*, Dr. E. Tscheuschner.

less lime. With excess of sand it is impossible to melt out the batch thoroughly and the resultant glass is full of stones, seeds, strings and other defects, in such quantity as to render the glass unmarketable. The other materials which are used in smaller proportions, if in quantities exceeding rational practice, always injure the quality of the glass or the pots or furnaces, and the watchful glassmaker must ever be on the alert for the appearance of defects that may be traced to an improperly proportioned batch.

The manufacturer of the ordinary varieties of glass aims to make a product as nearly transparent and colorless as possible. The presence in the raw materials of any metals which form colored silicates and which would consequently mar the transparency of the glass, is scrupulously avoided. It is impossible, however, to keep the materials entirely free from iron. Both oxides of iron, ferrous and ferric, form silicates. The ferrous silicate is of a green color, so intense that a very small portion of it in glass is sufficient to impart a very decided tint. Ferric silicate, on the other hand, is yellow, and colors the glass but faintly when present in small proportion. Since at least a trace of iron will always be encountered in the materials, the difference in the properties of these silicates is turned to good account. A small quantity of some oxidizing substance, usually arsenic, manganese, niter or nickel oxide, is added to the batch and the oxygen given off during the melt converts all the iron present into ferric oxide. If very white glass is required, manganese is used, as it tends to impart a violet color to the glass, which neutralizes the yellowish green tint of the iron. A very small quantity of cobalt also, is generally added, which by reason of its blue tint, renders the neutralization more complete. The use of manganese for window glass, however, is attended with an objectionable feature: the action of light in the course of time effects a change of the monoxide back to the dioxide, and in consequence, the glass acquires a violet tint. This may be occasionally observed in old windows where the glass has been undisturbed for a number of years. At present manganese is rarely used for window glass.

When molten glass stands undisturbed for some hours, crystallization takes place. This is called devitrification and, as the crystals have practically the same chemical composition as the glass in which they form, devitrification, therefore, is simply the crystalline form of the silicates whose amorphous form is glass. The crystals are grouped in various ways, sometimes interwoven throughout the glass and giving it the appearance of porcelain; sometimes in round knots which look like pebbles imbedded in the glass; sometimes in streaks which may be grouped in irregular figures. The effect produced by this crystal-grouping in glass is often very beautiful. Devitrification is rarely a source of annoyance in glass making, as the metal is always worked out as soon as ready, and when remelted the crystals lose their structure and the whole mass resumes the amorphous form. An excess of lime increases the tendency to devitrify and a limited quantity of alumina tends to retard it.

The specific gravities of various kinds of glass are as follows:

Window glass, 2·50 to 2·70; plate glass, 2·40 to 2·50; bottle glass, 2·60 to 2·70; lead flint glass, 2·80 to 3·25.

There is no definite point at which glass passes directly from a solid to a liquid; on being heated it begins to soften at a comparatively low temperature, grows softer as the temperature rises until at length it becomes quite liquid. This characteristic determines the processes of working it. It is cast, blown, rolled or pressed while in the semi-molten, plastic condition, assuming its final form while it is cooling and stiffening. When the article is not completed in the first operation, the glass is reheated and softened as often as is necessary until the work is finished.

In working glass it cools so rapidy that a tension is formed between the molecules of the different layers. The outside layer cools and hardens first, while the inside layers are still soft. As the inside layers cool and harden they tend to contract and draw away from the outside layers, but as the latter are already hard and rigid the result is not a contraction of the whole mass, but the development of tension, small or great, depending on the rapidity of cooling. This tension makes the glass very tough and strong and gives it the quality of enduring sudden changes in temperature, provided the surface be left unbroken; should a scratch or crack in the surface relieve the tension, the glass will fly into a thousand fragments. For ordinary use glass should be annealed, that is, cooled slowly, so that the outside and inside layers will cool simultaneously, and all tension thereby avoided.

Interesting investigations have recently been made by Mon. L. Grenet,* an eminent French engineer, to determine the limits of temperature within which annealing takes place and to ascertain the working temperature of glass of various kinds. His results led to the conclusion that these limits are quite small, and that there is only a small range of temperature where tension develops. For St. Gobain plate glass, the limits are 943 to 1087° F.; for a glass of composition, 71·5% SiO_2, 13 Na_2O, 15·5 CaO, the limits are 941 to 1154°; for an extremely basic silicate compound, 55% SiO_2, 21 Na_2O, 18 CaO to 6 BaO, the limits are 788 to 932°. In addition, the temperatures at which these various kinds of glass soften and lose form, were ascertained to be near the working temperatures, and are 1440°, 1521°, 1350° respectively, for the three glasses above cited. The practical significance of these deductions, if they are correct, is that the entire range of temperature within which it is necessary that great care be taken to cool the glass very slowly and evenly, is approximately but 200° F. Above the maximum and below the minimum it may be cooled quite rapidly without developing tension. Also the well-known fact was observed that devitrification may take place below the temperature of absolute softening. As the softening point is so much above the maximum temperature at which tension may develop, devitrification should never become a source of trouble in annealing.

The action of light on glass changes the color slightly in the course of time. Gaffield exposed several samples to the action of light for a year, and found but one unchanged. The changes in color of the different samples were as follows:

* *Bulletin de la Société d'Encouragement*, etc., June, 1897; November, 1898.

	Before.	After.
French window glass	Bluish white.	Yellow.
German plate glass	Light green.	Bluish green.
English window glass	Light green.	Yellowish.
English crown glass	Light green.	Light purple.
Belgian window glass	Brownish yellow.	Dark purple.
English window glass	Dark green.	Brownish green.
American plate glass	Very light blue.	Light purple.
American plate glass	Very light blue.	Light yellowish green.
American window glass	Bluish green.	Unchanged.

Certain rich colors found in old cathedral windows cannot be reproduced, as light has changed the colors from the original tints, and formed others that cannot be exactly matched in newly made glass. In explanation of this phenomenon Pelouze advances the theory that glass usually contains sodium sulphate and ferrous oxide; the action of light brings about a chemical reaction by which the ferrous oxide changes to ferric oxide and the sodium sulphate reduces to sodium sulphide. When the glass turns purple the manganese is oxidized and the sodium sulphate is reduced.

With the exception of hydrofluoric acid, glass is practically insoluble in acids. Nitric and hydrochloric acids have practically no effect upon it. Concentrated sulphuric acid attacks certain kinds of glass appreciably, others to a scarcely perceptible extent. Alkaline solutions, especially when concentrated and hot, often attack glass to some extent. Boiling water sometimes attacks it in a slight degree, especially if the glass is rich in alkaline silicate. When glass is heated to the softening point, sulphuric acid fumes attack it readily and deposit a smoky, white film on the surface; a source of frequent and serious annoyance when glass is worked in coal fired furnaces. The small quantity of sulphur nearly always present burns to sulphuric acid and at once settles upon the glass. This is called "sulphured" glass and is not fit for use, as the film cannot be removed without spoiling the glass.

Raw Materials.—Normal glass may be represented by the formula: R'O, R''O, 6 SiO_2, in which the R' elements=sodium, potassium, lithium; R''elements=calcium, barium, lead, zinc, strontium, aluminum, magnesium, thallium.

In addition, other ingredients are used in proportionately small quantities, for decolorizing, coloring and for opaque effects.

Decolorizers (oxides): Arsenic, niter, manganese, nickel. *Colors* (oxides): Cadmium, chromium, iron, gold, cobalt, copper, manganese, nickel, selenium, silver, uranium, antimony, tin.

While silica is the formative acid, salts of other acids are occasionally combined, viz.: Phosphates, fluorides, borates, stannates, arseniates. Phosphates and fluorides in certain combinations produce opaque glass.

The above list shows a large number of elements that enter into the composition of glass, yet the great bulk of material is included in the various compounds of sodium, potassium, calcium, lead, barium and silica, and a discussion of glass-making materials need not go beyond these.

Silica is used in the form of sand, and the many deposits of rock-sand in the United States that analyze over 99% SiO_2, furnish our factories with an abundant and excellent supply. The sand rock is crushed, washed care-

fully to remove all impurities, and the finest of the sand gives a product which is in sharp, even grains, best adapted for mixing and melting. Beach sand is not generally suitable for use, as the grains are too smooth and polished from the constant washing and rolling of the tides. The principal sand deposits that supply the glass industry at present are found in Massachusetts, Pennsylvania, West Virginia, Illinois, Missouri, Ohio, Indiana, New Jersey, New York and Maryland. It would be wrong, however, to infer that these comprise the whole or even the bulk of the sand deposits of the country. These particular deposits have been developed because of the glass industry in their respective localities. While good sand is indispensable for glass manufacture, it is safe to say that any locality advantageous in other respects for the manufacture of glass may be supplied with sand at a reasonable price. The most frequent objection to sand for glass making, is too high a percentage of iron, and the value of sand as a glass material can be determined only by analysis or actual use.

A few representative analyses of sand are as follows:

Components.	Nievelstein.	Hohenbocka.	Berlin.	Fontainebleau.	Pennsylvania.	
	%	%	%	%	%	%
SiO_2	99·975	99·71	99·50	98·8	99·06	99·5
CaO	0·010	.....	.....	.....	Trace.	.....
MgO	0·006	.....	.....	.....	0·05	Trace.
Al_2O_3, Fe_2O_3	0·009	.....	0·18	0·7	0·76	0·4

Good sand is very abundant in the United States, and recourse has never been taken to the use of silicate minerals which, with slight additions, will produce a cheap glass, although not of the best quality. The analyses of several of these minerals are as follows:

Components.				
	%	%	%	%
SiO_2	43·75	44·50	88·50	65·75
Al_2O_3	13·89	16·75	5·87	18·28
Fe_2O_3	0·86	20·00	1·75	.....
CaO,CO_2	36·28	.....	.....	.....
CaO	.....	9·5	2·00	.....
Na_2O	2·73	2·6	.....	14·27
MgO	.....	2·25	.....	.....
MnO	.....	0·12	.....	.....
Organic	2·77	2·0	.....	.....

Limestone is found widely distributed and is now generally used as finely ground carbonate. Burnt and slaked lime constantly change their chemical composition by absorbing moisture and carbonic acid from the air, and this changing presents difficulties in keeping the proper proportion for the batch. Carbonate of lime is a stable compound and may be proportioned accurately. Different proportions are used in mixing the carbonate, oxide or hydrate, so that the proper amount of calcium will be in the glass. These proportions are: Limestone, 100; burnt lime, 56; slaked lime, 74.

The limestone used should be analyzed from time to time as the composition varies even in different parts of the same quarry. Iron and magnesia carbonates must be avoided, as the former colors the glass and the latter tends to make it infusible.

Representative analyses of limestone are as follows:

Components.	Prussia.	Pennsylvania.	Germany.	Components.	Prussia.	Pennsylvania.	Germany.
	%	%	%		%	%	%
Ca_2CO_3	95·75	97·23	46·98	SiO_2	0·84	1·01	15·12
$MgCO_3$	1·00	1·48	27·76	Moisture	0·61	.....	2·85
Fe_2CO_3	0·33	0·16	2·77				
Al_2O_3	0·52	0·02	4·00		99·90	100·00	100·02
Alk. li	0·85	.....	0·54				

Barium is used in the form of the mineral barite (heavy spar), $BaSO_4$, or witherite, $BaCO_3$; generally the latter. When substituted for lime it produces a superior quality of glass in respect to homogeneity, brilliancy and fusibility. Its higher cost, however, has prevented its general use. Frequent attempts have been made to replace the alkali wholly or in part by it, but with but little success. While barium resembles the alkalies in some of its properties, its place in glass making is naturally classed with lime.

Two oxides of lead are used as glass-making materials, litharge, or lead monoxide, PbO, and red lead, a mixture of lead monoxide and dioxide of the composition $2PbO+PbO_2$. The latter is preferable on account of its excess of oxygen. Carbon, or carbonaceous compounds in contact with the lead oxides during the melt, reduces them to the metal, which discolors the glass. For this reason lead glass is melted in covered pots and the utmost care is exercised to have the batch materials free from impurities. The excess of oxygen in the red lead serves to prevent the bad effects of traces of carbon or carbonaceous compounds inadvertently present in the batch. These carbon or compounds merely reduce the dioxide to the monoxide, and no metallic lead is formed. Manufacturers frequently find it advantageous to roast their own lead and insure pure material. Muspratt gives the permissible limit of metallic oxide impurities usually found in lead oxides as 0·0014% Cu, 0·005% Sb, 0·0075% Bi. The finer the ware the purer should be the oxides. Excess of lead in glass imparts a yellow tint to it. A sample of yellow glass analyzed by the writer was of the following percentage composition:

SiO_2, 38·35; PbO, 53·52; As_2O_3, 0·27; $Al_2O_3Fe_2O_3$, 0·52; CaO, 0·70; K_2O, 3·63; Na_2O, 3·67.

Lead glass is usually manufactured with potassium as the alkali base.

Zinc oxide is used to a limited extent in glass manufacture. It is more easily procured free from impurities, it clears more quickly and withstands greater heat than the oxides of lead. It imparts a faint yellow tint to the glass, which intensifies as the glass remains in the pot after the melt is completed. This color is usually neutralized by the addition of a small quantity of nickel oxide.

Sodium is used in two combinations, carbonate and sulphate, termed "soda ash" and "salt-cake."

Sodium carbonate decomposes readily in the high temperature of the melt, the carbonic acid passes off and the alkali unites with sand and forms a silicate:

$$Na_2CO_3+4SiO_2=Na_2Si_4O_9+4CO_2.$$

Solvay soda is more free from impurities than Leblanc soda, which usually contains some sulphate and chloride. In the use of Leblanc soda it is generally advantageous to add a little carbon.

Sodium sulphate—salt-cake—is in general use for the manufacture of window and green bottle glasses. It does not decompose readily in the melt and requires the addition of a small quantity of carbon, usually as charcoal or ground coal, to effect the reduction of the sulphate. This forms carbonic acid and sulphur dioxide gases, which pass off, and sodium oxide, which unites with the sand and forms sodium silicate:

$$2Na_2SO_4+C+8SiO_2=2Na_2Si_4O_9+CO_2+2SO_2.$$

From this reaction it requires theoretically about 4·25 lb. of carbon per 100 lb. of salt-cake. In practice it is necessary to use about 50% excess of the theoretical requirement. Salt-cake glass attacks the clay of pots and tanks more than glass manufactured from soda ash. In addition to the economy of salt-cake, it is better adapted for continuous tank melting, as the glass is more liquid and becomes homogeneous in a shorter time.

Sodium borate, or borax, is used to a considerable extent in glass making. It is not decomposed in the melt analogous to the carbonate and sulphate, but remains and produces a glass which is a mixture of silicates and borates. This glass is more fusible than the usual mixture of different silicates; and is nearer absolutely colorless than the silicates or borates alone. When boric acid is present the percentage of lead may be raised considerably without having the clarity of the glass impaired by the yellow tint which excess of lead gives to a silicate glass. For this reason it is used quite extensively in the manufacture of optical glass. The presence of boric acid makes glass a better solvent for the metallic oxides used to color the glass, and it is used considerably for the manufacture of fancy articles of various kinds from colored glass and enamels.

Potassium is used in the form of potassium carbonate, more familiarly known as "pearl ash," or "potash." It is manufactured mainly from the mineral chloride by the Leblanc process similar to the manufacture of soda ash from salt, and is obtained very pure. It is used for the manufacture of the finer grades of glass, as it yields a practically colorless product. A faint bluish tint is always present in glass manufactured from sodium salts. Potash glass has less brilliancy than soda glass, but as brilliancy is attained in the finer varieties of glass by the use of lead, the lesser brilliancy of the potassium silicate is no great detriment. Potassium sulphate is sometimes used but not to any noteworthy extent.

Lithium is occasionally used in the manufacture of fine optical glass. Its rarity and cost debar it from general use.

The substances discussed above include the bulk of materials from which glass is made, with the exception of those used in small quantity as decolorizers, as coloring oxides and as materials for making opaque glass. The selection of materials should be governed by the grade of glass to be manufactured. To use the finest and purest materials for the manufacture of glass not of the highest quality, is extravagance, and to use any materials except the best for

the manufacture of fine ware is no economy. With our abundant supply there is rarely any profit in the use of inferior materials.

Analyses of good fire clays of various sources are as follows:

Components.	England.		Germany.		Belgium.	France.	Pennsylvania.	Missouri.
	Stourbridge.	Birmingham.	Grossalmerode.	Klingenberg.	Andenne.	Forges-les-eaux.		
	%	%	%	%	%	%	%	%
SiO_2	63·30	65·10	47·50	51·10	49·64	65·00	61·98	65·08
Al_2O_3	23·30	22·22	34·37	33·00	34·78	24·00	27·23	28·55
CaO	0·73	0·14	0·50	0·44	0·68	.....	0·32	1·06
MgO	.....	0·18	1·00	.....	0·41	.....	0·15	0·77
Fe_2O_3	1·80	1·73	1·24	4·30	1·80	Trace.	0·35	1·27
H_2O	10·30	10·30	14·00	12·11	12·06	11·00	7·50	9·91
Alkali	1·50	0·18	.....	.....	0·41	.....	2·15	.....

Equally important with the materials forming the glass are those entering into the construction of the furnaces in which the glass is melted. Consideration must be given to two influences which cause constant wear to furnace materials, noting always that the rapid wear not only entails high repair cost, but produces a glass more impure and of a correspondingly inferior quality. These two influences are, the extremely high temperature of the furnaces, tending to melt it down, and the chemical action of the batch materials and the melted glass itself, tending to decompose the materials of construction. Those parts of the furnaces that do not come into direct contact with the melting batch or glass, as the crowns, ports, flues, etc., are constructed of materials selected chiefly in regard to refractory qualities. Where the heat is most intense silica brick is required; in other parts, different grades of firebrick are used corresponding to the requirements. Those parts which come into contact with the melting batch and glass, viz., pots, tank blocks, rings and floaters, are made of fire clay. The two influences above noted conflict at this place; to resist intense heat the clay should contain as high a proportion of silica as possible; to resist the action of the fluxes the proportion of silica, especially of uncombined silica, should be low. When the batch melts, a silicate of alkali is first formed, probably highly acid, and as the melt progresses, silicate of lime, lead or other R″ base is formed and dissolved in the alkaline silicate. This R″ base combines with whatever silica it finds, whether as sand in the batch, sand in excessive combination with the alkali, as free silica in the clay of the pots or blocks, or, possibly silica in combination with the alumina of the clay. In the clearing-up stage of the melt, when the glass is superheated to drive off the impure gases held in suspension, the glass has a strong basic action, dissolving the silica out of the clay and some of the alumina or aluminum silicate. The wear and tear of the blocks and pots is chiefly caused by the action of lime or lead, and by the basic action of the glass itself when superheated. The practical manufacturer seeks to find a rational medium in the proportioning and working of his clay. It must contain a goodly proportion of silica to resist the high temperatures and as much as possible should be in combination with alumina. By thorough mixing and aging the mass is made quite homogeneous. Care must be taken in the manufacture of the various shapes to make them as solid as possible to avoid the formation of cracks and

holes in shrinking, and to dry and burn them slowly and gradually. In general, a mixture of clays is preferable to a mixture of raw and burnt clay of the same composition. Analyses of fire clays are given on the preceding page.

FUEL.*—The fuel used in the manufacture of glass has played a very important part in the development of the industry. The trend of progress has been in the substitution of gaseous fuel for solid fuel.

Gaseous fuel possesses many advantages that render it especially applicable to glass manufacture. Its freedom from ash dust and its easy control of burning and consequent furnace temperature are factors of great value.

Natural gas is the best fuel for glass melting by virtue of its high calorific value and ease in application. Its first use so changed this part of the manufacture that solid fuel has for the most part been driven from the field and modern melting furnaces are principally gas fired.

When natural gas is not available, solid fuel is converted into gaseous form by means of a producer.

CLASSIFICATION OF GLASS. (*a*)—*Commercial Classification* based on trade names:

1. *Polished Plate* embraces all glass cast upon a smooth table, rolled to the required thickness with a roller, annealed and then ground and polished. Under this head comes *thin plate*, a recently developed process.

2. *Rough Plate* embraces all glass cast as above but *not* ground and polished. The principal varieties are *ribbed plate, colored cathedral, rough plate, wire glass, heavy rough plate* for skylights.

3. *Window Glass* embraces all glass blown in cylinders, and afterward cut and flattened out and polished while hot. Chiefly used for glazing, pictures, mirrors, etc.

4. *Crown Glass* embraces glass blown in spherical form and flattened to a disk shape by centrifugal motion of blown pipe. A little is made at the present time for decorative purposes.

5. *Green Glass* embraces all the commoner kinds of glass, and is not necessarily green in color. It is used in the manufacture of bottles, carboys, fruit jars, etc.

6. *Lime Flint* embraces the finer grades of bottles used for the prescription trade, tumblers, certain lines of pressed tableware and many novelties.

7. *Lead Flint* embraces all the finest products of glass making, such as fine cut glass, tableware, optical glass, artificial gems, etc.

This classification may be condensed into four classes: *Plate, Window, Green* and *Flint.*

(*b*)—*A Classification Based on Method of Working.*—Pressed glass, blown glass and cast glass. The first must have the property of retaining the form into which it is pressed and have little shrinkage; the second must have good even cohesion and high melting point; the third must combine considerable liquidity at the casting stage, with even cohesion and easy annealing properties.

(*c*)—*German Classification, Based on Color.*—Gruenes, halb-weisses und weisses glas—green, half-white and white glass.

(*d*)—*Henrivaux's Classification, Based on Lead Contents.*—In the class with no

* See article on Fuel and its Utilization, earlier in this volume.

lead are placed (1) flat glass, window glass, crown and plate glass. (2) Hollow glass, green bottles, prescription, tumblers and hollow tableware, tubes, carboys, etc. (3) Pressed and soluble glass. The lead bearing class is comprised of crystal, optical, enamel and strass.

(*e*)—*French Classification, Based on Commercial Use.*—Bouteilles, verres a vitres, gobleterie, glaces, miroirs.

The commercial classification (*a*) is the most satisfactory for all purposes.

MANUFACTURE AND METALLURGY.—The foundation of good glass is a batch well proportioned and thoroughly mixed. Specific rules and formulæ cannot be established for the proportions of the various materials. Much depends upon the character of the raw materials, the fuel used, and the style and construction of the furnaces. The success of the operation is mainly determined by the skill and intelligence with which these factors are studied and applied, and is primarily a matter of experience. A good batch for one furnace and a certain set of materials will often prove unsatisfactory when used for another furnace differing somewhat in design, or when the materials are procured from other sources. This is apparent in any book of glass-making recipes, which gives the results of experience in different sections and under different conditions. A practical glassmaker in examining such a book finds few that he could use with success. It is interesting to compare the mixes given for this country with those of Europe. The following mixes are taken from Gessner's Glassmaker's Handbook, Tscheuschner's Handbuch der Glasfabrikation, and Henrivaux's Le Verre et le Crystal:

WINDOW GLASS.

Components.	American.				Germany.				French.		
Sand	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0
Salt-cake	44·0	40·0	42·0	43·0	35·0	45·0	45·0	35·0	35·0	40·0	42·0
Soda ash	.....	4·0	.....	.....	.....	.....	.....	6·0	.....	.....	.....
Limestone	36·0	38·0	40·0	34·0	30·0	38·0	32·0	30·0	25·0	35·0	34·0
Carbon	4·0	8·0	6·0	5·0	12·5	3·0	2·0	3·0	1·5	2·0	2·0
Arsen c	2·0	1·0	2·0	0·5	.....	.....	.....	.....	0·5	1·0	.....

PLATE GLASS.

Components.	American.			German.		
Sand	100·0	100·0	100·0	100·0	100·0	100·0
Salt-cake	.....	.....	40·0	98·0	.....	.....
Soda ash	30·0	36·0	.....	.....	35·0	33·0
Limestone	24·0	24·0	38·0	38·0	.....	.....
Slaked Lime	.....	.....	.....	.....	20·0	14·0
Carbon	1·0	0·75	4·0	2·5	.....	.....
Arsenic	.....	1·0	2·0	0·5	.....	.....

LIME FLINT.

	German.			
Sand	100·0	100·0	100·0	100·0
Soda ash	.....	.....	30·0	33·0
Potash	32·0	58·0	.....	.....
Limestone	.....	.....	36·0	36·0
Slaked Lime	18·0	12·0	.....	.....
Saltpeter	.....	8·5	1·5	.....
Arsenic	0·3	0·4	.....	2·0
Manganese	0·2	0·3	0·5	1·0
Cullet	100·0	100·0	100·0	100·0

GREEN BOTTLES.

Components.	American.			
Sand	100·0	100·0	100·0	100·0
Salt-cake	.....	.....	38·0	42·0
Soda ash	38·0	35·0	.....	.....
Limestone	36·0	32·0	34·0	36·0
Charcoal	.....	.....	5·0	8·0

GREEN BOTTLES, WITH CHEAP MATERIALS.

Components.	German.			
Sand	100·0	100·0	100·0	100·0
Wood ashes	160·0	150·0	150·0	176·0
Salt-cake	.....	.....	26·0	.....
Basalt	50·0	.....	.....	.....
Limestone	.....	30·0	30·0	88·0
Alumina	.....	15·0	.....	.....
Heavy spar	.....	10·0	.....	.....
Lava	.....	.....	25·0	.....
Feldspar	.....	.....	.....	291·0
Coke	.....	.....	3·0	.....

LEAD FLINT.

Components.	American.		German.				French.		
Sand	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0
Potash	36·0	34·0	35·0	28·0	30·0	34·0	31·0	33·0	30·0
Red Lead	40·0	48·0	67·0	67·0	66·0	42·0	67·0	67·0	67·0
Niter	5·0	6·0	.....	6·0	3·0	16·0	.....	.....	4·0
Borax	0·3	.....	.....	.....	.....	.....	.....	.....	.....
Arsenic	0·5	0·15	0·3	0·25	0·3	.....	.....	0·6	.....
Manganese	0·25	0·06	0·2	0·2	0·2	0·5	.....	0·45	0·025
Antimony	.....	0·02	.....	.....	.....	.....	.....	.....	.....

OPTICAL GLASS.

Components.	German.			
Sand	100·0	100·0	100·0	100·0
Potash	23·0	26·0	25·0	30·0
Red lead	100·0	100·0	128·0	67·0
Borax	6·5	.....	.....	.....
Niter	.....	6·75	2·0	3·33

CROWN GLASS.

Components.	English.	
Sand	100·0	100·0
Potash	45·0	43·66
Red lead	9·46	.....
Lime	9·46	21·66
Niter	1·89	2·00
Arsenic	.....	0·75

Thoroughness in batch mixing is indispensable for making good glass. The mixing room of a well-regulated factory should be kept scrupulously clean, and its appearance should be as expressive as that of a kitchen. Much improvement has been made recently in the equipment of factory mixing rooms. Formerly all mixing was done by hand and the weighing out done on an ordinary platform scale, which placed the entire responsibility with the foreman. Charging scales are now generally used and give the factory superintendent a better control of the batch proportions. Materials which are used in very small amounts are weighed out by the foreman on a special scale. In the smaller factories mixing is done by hand and if thorough there is little economy if any, in the use of machinery. In the larger factories, however, machinery is used and insures a greater economy and a more evenly mixed batch. The mechanical appliances are few and simple. When the batch is laid out on the floor complete, it is scooped into an elevator and carried to the mixer. After passing through the mixer it is screened, and other conveyors take it to the storage bin or the furnaces. Mixers are of various forms, generally upright with revolving shovels, or a revolving horizontal drum with radial arms to stir and mix the batch. The use of automatic scales is developing. At the Dralle Bottle works in Berlin, Germany, an equipment was installed, in which the various materials flowed from their respective bins into a horizontal spiral conveyor. The materials were proportioned by regulating the flow from the bins. Much difficulty was experienced in operating, as the materials would not flow uniformly, and the apparatus was finally abandoned. An apparatus with the same object in view has been successfully operated by the Chambers Glass Co., at Arnold, Pa. The materials in this case are weighed with great accuracy by an automatic scale.

An important matter in connection with batch making deserves more attention than has usually been devoted to it; namely, the chemical control of the operations. The analyses of the glass, of the materials used for the batch and of the gaseous fuel, if rightly understood and applied, cannot help but give

the factory superintendent a better grasp of the situation and assist him materially in obtaining good results. Slight variations in the composition of the materials are liable to occur; magnesia in the limestone, free acid in the salt-cake, alumina and iron in the sand; and a variation in the proportion of any of these affects the resultant glass.

If the materials are subjected to regular inspection and analysis, such trouble is foreseen and avoided. Glass making is a branch of the chemical industry, where much is done on empirical lines. A glass works laboratory does not require an extensive equipment. Apparatus and reagents for ordinary inorganic analysis, with especial view to the analysis of silicates, is all that is necessary. Good platinum apparatus should be provided, and it is of advantage to have a small, high temperature furnace for conducting experimental melts. Great benefit would result if the exact relations existing between the properties of glass and its chemical composition were determined. Considerable chemical work has been done by chemists who have had up to the present time no practical knowledge of glass manufacture. Their work is of little value to the industry. It is only in the factory itself, where the chemist observes actual glass-making conditions, and the superintendent becomes familiar with the chemical facts that work yielding practical results can be done.

The batch is the mechanical mixture of the materials which form the chemical compound glass, and the series of chemical reactions by which the former is transferred into the latter is called the melt. Choosing for illustration a batch of sand, salt-cake and lime, the reactions are as follows:

1. $\begin{cases} 2Na_2SO_4+C=2Na_2O+CO_2+2SO_2. \\ Na_2O+4SiO_2=Na_2Si_4O_9 \text{ (soluble glass).} \end{cases}$
2. $CaCO_3+SiO_2=CaSiO_3+CO_2.$

Composition of glass—Na_2O, CaO, $5SiO_2$.

Various silicates of soda and lime may form, and silica is soluble in silicate of soda, so we cannot say definitely that the combination forms simple silicates only, of the above structure. The composition, however, should approach that which is assumed above. When the batch is filled into the pot or tank the alkali is quickly reduced to the oxide, and melts to form sodium silicate with some of the sand. In the meantime the carbonic acid gas has been liberated from the lime, and the lime seeks silica, uniting with it to form calcium silicate, which dissolves in the liquid sodium silicate. As soon as the saturation of the bases is complete the melted material has the chemical structure of glass, with suspended, gaseous impurities—the carbon dioxide, sulphur dioxide and steam that have been liberated in the melt. These gases must be driven off before the glass is fit to work, and it is "fined" or "cleared-up" by superheating the mass until it becomes extremely liquid, when it is stirred a few times, if the melt is made in a pot furnace. The gases rise through the liquid mass and escape, leaving a pure, solid body of limpid glass, which is settled and cooled slowly and carefully before working. This stage of the melt is called the "standing off." There are thus three distinct stages of the melt and in a pot furnace it is dependent entirely upon the skill and carefulness of

the shearer, to guide the melting batch through these stages to a successful end. This requires a strict watch on the melt, the furnace and the pots, as there is always danger of breaking these large and heavy crucibles, which are so sensitive to any sudden change of heat. In the use of the continuous furnace, the work is somewhat simplified for the shearer. The batch is charged regularly at one end and the glass worked out continuously at the other end. The problem is largely one of furnace construction—to design the furnace so that the melting batch as it floats from one end to the other, shall at each point meet that degree and kind of heat required for that particular stage of the melt. There are practically no chemical changes in the materials other than those above noted. A very small quantity of alkali volatilizes and passes out into the stack, and small quantities of alumina are dissolved from the clay blocks or pots by the glass; in both cases, however, the amounts are comparatively so small that they produce no change in the chemical composition. To prove this the writer analyzed a certain make of green bottle glass and the materials used in its manufacture. The proportions of the mix were: Sand, 100; salt-cake, 45; limestone, 30; coal, 3·5; cullet (broken glass), 79.

The difference in the actual composition of the glass and that calculated from the analyses of the materials of the mix proved to be slight, as follows:

Components.	Analyzed.	Calculated.
	%	%
SiO_2	72·33	72·42
Al_2O_3, Fe_2O_3	1·41	1·15
CaO	11·34	11·67
MgO	0·26	0·23
Alkali	14·66	14·53
	100·00	100·00

Several years ago a process was brought out in England which aimed to shorten and assist the melt by injecting oxygen into the batch when the melt was partly made. This was accomplished by inserting a platinum tube into the pot or tank when the batch had melted down to a doughy, plastic mass, and blowing a stream of oxygen through the mass for fifteen or twenty minutes. The oxidation of the iron present raised the heat and gave the glass a lighter color. The stirring caused by the passage of the gas aided the expulsion of the suspended gaseous impurities and the time of the melt was appreciably shortened. For pot furnace melting of flint glass, the melt under the ordinary conditions required 20 to 25 hours; with the use of oxygen, 11 to 13 hours. For plate glass the ordinary melt required 15 hours; with the use of oxygen nine hours. For tank furnace melting, a 20-minute injection of oxygen advanced the melt as far as with an hour and a half of ordinary melting. In these tests about 50 cu. ft. of oxygen was used per ton of glass melted. Practical difficulties in the application of the process has hindered its use; the saving indicated by the tests shows that the process will be of considerable value to the glass industry if these difficulties can be overcome.

In addition to the chemical reactions of the batch materials, the chemical action of the firing must not be disregarded. The flame of a glass furnace is in-

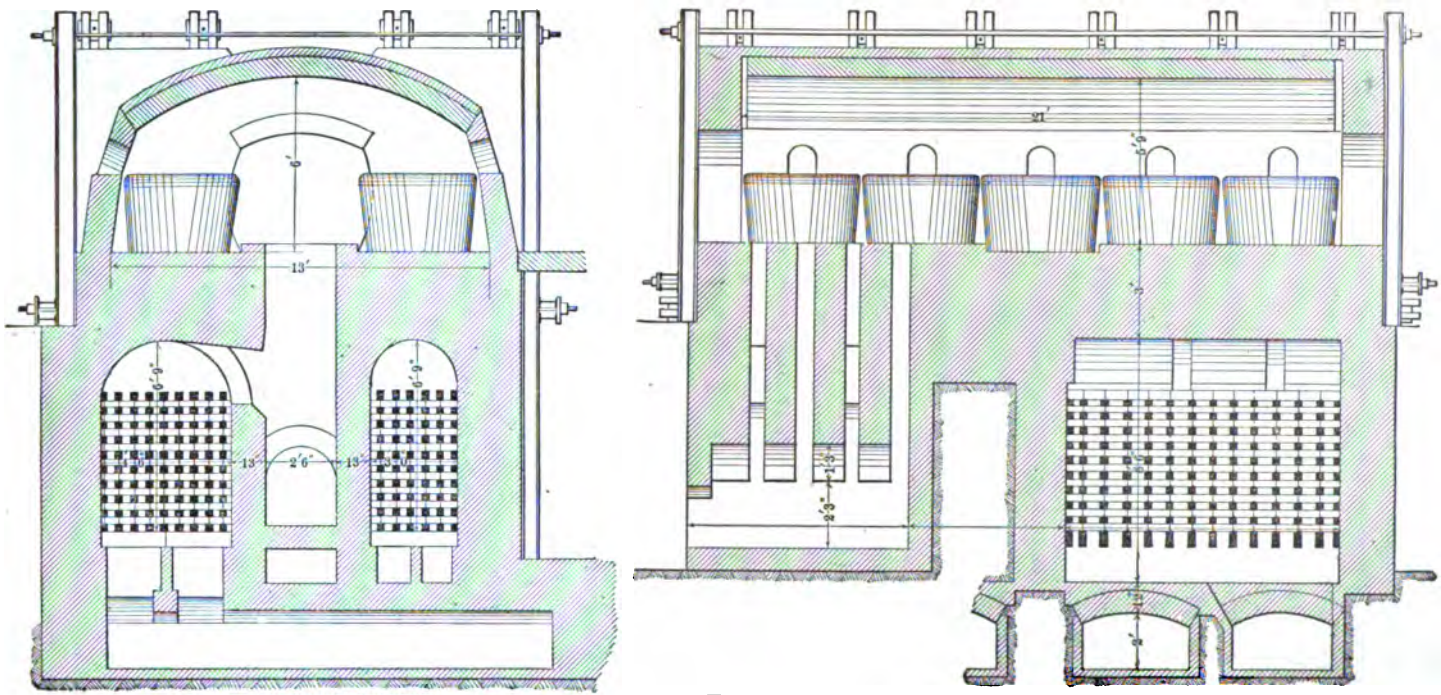
variably reducing, yet its reducing effect differs in degree. When a lead batch is melted, it must be done in covered pots to prevent the flame contact with the batch, else the lead oxide will be reduced and remain in the glass as dirty, gray streaks of metal. Even glass which contains no lead must be melted in covered pots to protect the oxidizing agents from the reducing action of the flame, otherwise their bleaching action will be decreased and the glass produced will not be of the desired white color.

Glass furnaces may be divided into two groups—pot furnaces and tank furnaces. Each of these may be subdivided into two classes, the former into open pot furnaces and covered pot furnaces, the latter into continuous and day tanks. A pot furnace is of rectangular or circular plan in which are placed a number of large fire clay crucibles ranged immediately inside of the breast wall, with room in the center of the furnace for combustion space. The batch is filled into these crucibles through openings in the side of the furnace, and after the melt is completed, the glass is worked out through the same openings, except in the case of plate glass, when the pots are removed bodily from the furnace and their contents cast. The tank furnace does away with the pots and the glass is held on the hearth of the furnace itself, the flame sweeping across the furnace above it. The continuous tank has this hearth full of molten glass at all times; the batch is filled in continuously at one end and the glass is worked out continuously at the other. In a day tank the action is intermittent; the batch is filled in and melted and the glass then worked out, similar to the working of a pot furnace.

Gaseous fuel is indispensable for tank furnaces. The firing is overhead and the flame must be keen, powerful and free from ash or dirt, conditions which cannot be attained by overhead firing from coal direct. Pot furnaces are seldom direct coal fired and it is found advantageous to use gas producers in preference, when gas coal is obtainable.

The tank furnace, especially when continuous, shows a considerable economy in fuel and labor over the pot furnace. The pot furnace supplies glass to the workers one-third of the time only; the remaining two-thirds of the melt requires fuel and labor as well as the productive one-third, while the tank furnace, which requires constant labor and fuel, is supplying glass to the workers the entire time. The calorific efficiency of the continuous tank furnace, or the percentage of the heat value of the fuel that is actually absorbed in melting operations, is about 10 to 12%; that of the pot furnace is 4 to 5%. The commercial economy, while not corresponding exactly to this, is so considerable that it is difficult for pot furnace plants to compete successfully with the modern tank furnace plants. In regard to day tanks, H. L. Dixon estimates a saving of approximately 24% in cost of metal over pot furnace cost; and Henning & Wrede, of Dresden, give figures of operation, showing an increase of output amounting to 40% for the same fuel and labor required by the pot furnace.

With these few remarks on furnaces it will be of interest to notice briefly their general style of construction. It is not within the scope of this article to give details of construction, but merely to indicate by brief description, aided



TRANSVERSE SECTION

LONGITUDINAL SECTION

Fig. 1.

Regenerative Open Pot Glass Furnace.

Elevation.

by drawings, the general lines followed in building the furnaces now in use. The following list comprises the principal furnaces:

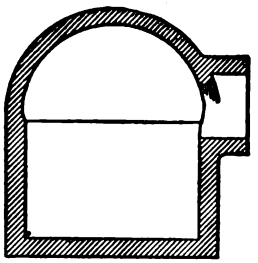
1. Open pot furnaces for bottles and window glass.
2. Open pot furnaces for plate glass.
3. Covered pot furnaces for flint glass.
4. Day tanks.
5. Regenerative continuous tank furnaces for window glass.
6. Regenerative continuous tank furnaces for bottles.
7. Recuperative continuous tank furnaces for bottles.

The open pot furnace for bottles and window glass (Fig. 1) is rectangular in plan, of different lengths according to the number of pots used, and three times the width of one pot. The number of pots varies from six to twelve, generally eight or ten, placed in two rows. The crown is preferably rather flat, but set high enough to allow adequate flame development and good distribution of heat between and above the pots. The regenerators are so located that the occasional broken pot—that disagreeable yet unavoidable contingency—does not pour its contents into them and stop the draught. Pockets are placed underneath the furnace into which glass resulting from accident may be caught and removed. Openings in either end of the furnace serve to remove and replace broken pots. The air and gas flues rise vertically at either end, and terminate in ports at or below the hearth level. The flame travels from one end of the furnace to the other and describes an arc of a circle; by lessening the draught it can be made to fill the furnace completely with its volume. A direct fired furnace has no flues or regenerators; the firing is on a grate in the space between the two rows of pots at the bench level.

The open pot furnace for plate glass is longer than the pot furnace for bottle glass, and is usually built to contain twenty pots. There is an opening opposite each pot large enough to admit of its convenient removal for casting. The air and gas flues come up and terminate vertically, or they come up vertically above the hearth level and turn so as to enter the furnace horizontally. These furnaces are gas fired.

The covered pot furnace for flint glass (Fig. 2) is of circular plan. The pots (Fig. 3) have a hood-like top which projects to the opening in the breast wall of the furnace, and through which the batch is filled in and the glass worked out. The contents of the pots are thus entirely isolated from contact with the flame. To replace a broken or worn-out pot the wall in front is taken down, the old pot removed, the new one set in and the wall rebuilt. Flint pots usually last several months, while the usual life of open pots is about six weeks. The number of pots in a furnace varies from eight to sixteen. As all of the heat available for melting must be transmitted through the walls of the pots, the arrangement of firing differs materially from that of an open pot furnace. In the open pot furnace the firing develops in the combustion space between the pots and above them in one solid flame. Here it must be divided so as to play closely around each pot. The flame is drawn from the center of the furnace to a series of flues at the breast wall, a flue being placed between each pot and the adjoining one. The crown is just above the pots and the stack widens at the

base sufficiently to take in the entire diameter of the furnace. The draught flues empty into the stack through the breast wall or crown. Regenerative firing by reversal of currents is impracticable for this furnace. A certain preheating of the air for combustion by leading it along flues adjoining the gas flues, or underneath the furnace, is advantageous in gas firing. Direct coal firing may also be used.



FLINT GLASS COVERED POT.
FIG. 8.

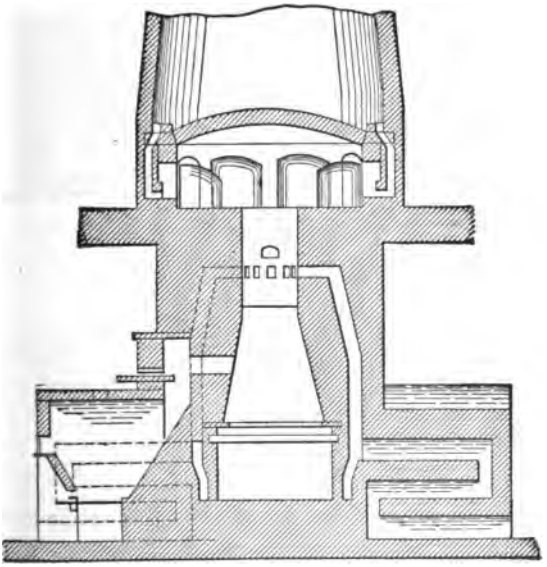


FIG. 2.
COVERED POT FLINT GLASS FURNACE.
MURPHY STYLE OF RECUPERATION.

In the firing of gas-fired glass furnaces, a distinction is made between the "regenerative" and the "recuperative" systems. The purpose of both is to preheat the air for combustion by utilizing the waste heat of the furnace gases. In regenerative firing there are two chambers filled with loosely stacked firebricks called "regenerators," one at each side of the furnace; the hot waste gases, resulting from combustion, pass through the regenerator at one side, and the bricks take up the heat of the waste gases and soon become very hot. The direction of the flame is then reversed and the air for the combustion enters

through the hot regenerator and the waste gases pass out through the regenerator at the other side of the furnace. By a periodical reversal of the direction of the flame the hot, outgoing waste gases are constantly giving off their heat to the regenerators and the incoming air constantly returning it to the furnace. The recuperative firing differs in that there is no reversal of current. The waste gases are led through a series of flues with thin walls; the incoming air passes through a second series of flues in contact with the walls of the first set. The heat of the outgoing waste gases is transmitted through the thin flue walls and is absorbed by the incoming air.

The day tank (Fig. 4) is practically a pot furnace with one large, open pot. The hearth of the furnace—the "tank"—is a square or rectangular box built up of fire clay blocks and well ventilated on the bottom to prevent excessive wear and leakage, which will occur should the glass become too hot and soft on the bottom. The furnace is gas fired, the air and gas is brought up through flues

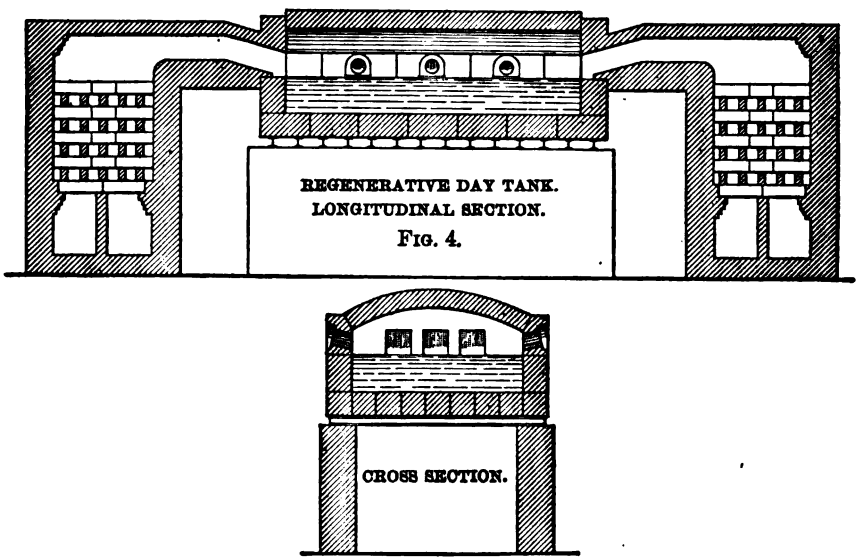


FIG. 4.

at one end and the products of combustion discharged though similar flues at the other end. The furnace may be of the regenerative or recuperative type.

The regenerative continuous tank furnace (Figs. 5, 6 and 7) is a more distinctive type. The tank contains a very much greater volume of glass and it is always full or nearly so. The batch is filled in continuously at one point and drawn, melting as it goes, to the working holes where it is worked out. Before it reaches the working holes the melt is complete and the batch is fully transformed into glass. The window glass tank furnace is oblong in shape, the length is four or five times the width, and the depth varies from 4 to 6 feet. The capacity varies from eight rings or less up to twenty. A 20-ring tank furnishes glass for 60 blowers and represents a capacity equal to six pot furnaces of the ordinary size. It is essential to have the bottom and sides of the tank well ventilated, and of a construction that any necessary repairs during the blast may

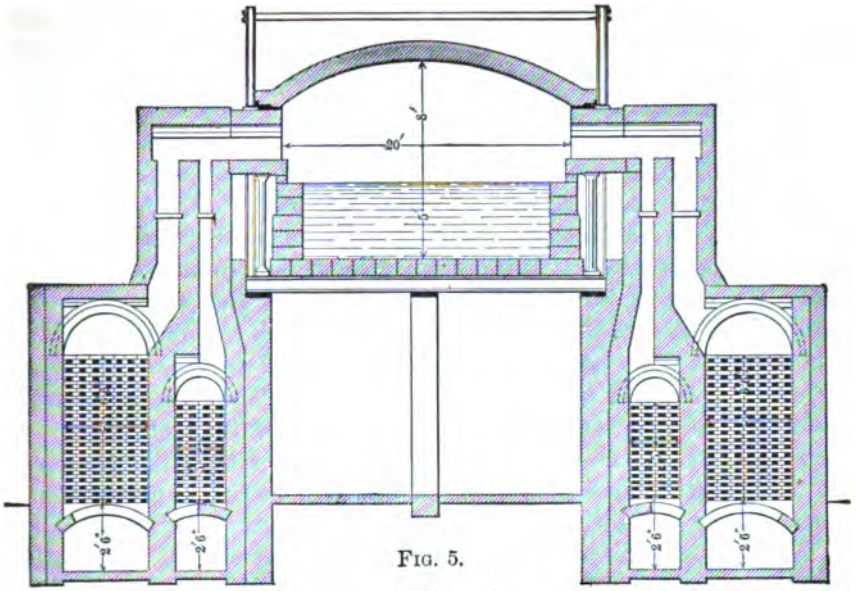
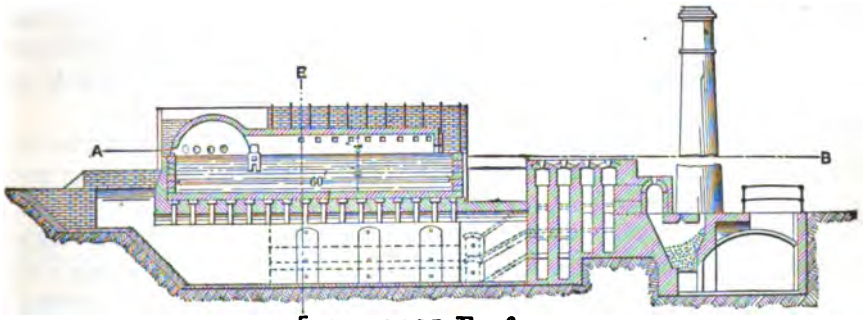
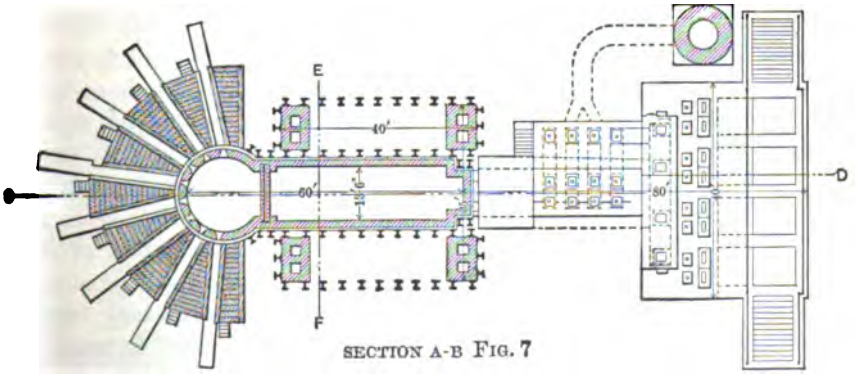


FIG. 5.

SECTION OF WINDOW GLASS TANK FURNACE.
BAUDOUX-PAGNOUL TYPE.



SECTION C-D FIG. 6



SECTION A-B FIG. 7

8-RING CONTINUOUS WINDOW GLASS TANK FURNACE.
GOBBE STYLE OF CONSTRUCTION.

be made with little or no interruption of the work. As all furnaces are out of blast in the summer months by reason of the heat, the bulk of the repairs are made then. The firing is always regenerative, as recuperative firing does not give a sufficiently intense heat. The regenerators are located on either side of the tank foundation and the flues lead from them to ports in the side walls of the furnace above the suface of the glass and just beneath the crown, as in the Bandoux-Pagnoul construction, or to ports that open into the furnace through the crown, as in the Gobbe construction. The regenerators, ports, flues, valves and stack, should be arranged to develop a sheet of flame as nearly continuous as possible. At a point somewhere near the working end one or two floating bridges of fire clay—"floaters"—are set in the bath of glass and secured to the sides of the tank. The glass in flowing from one end to the other passes beneath these floaters and becomes thoroughly mixed and homogeneous. The depth of the floaters is usually one-fourth the depth of the tank.

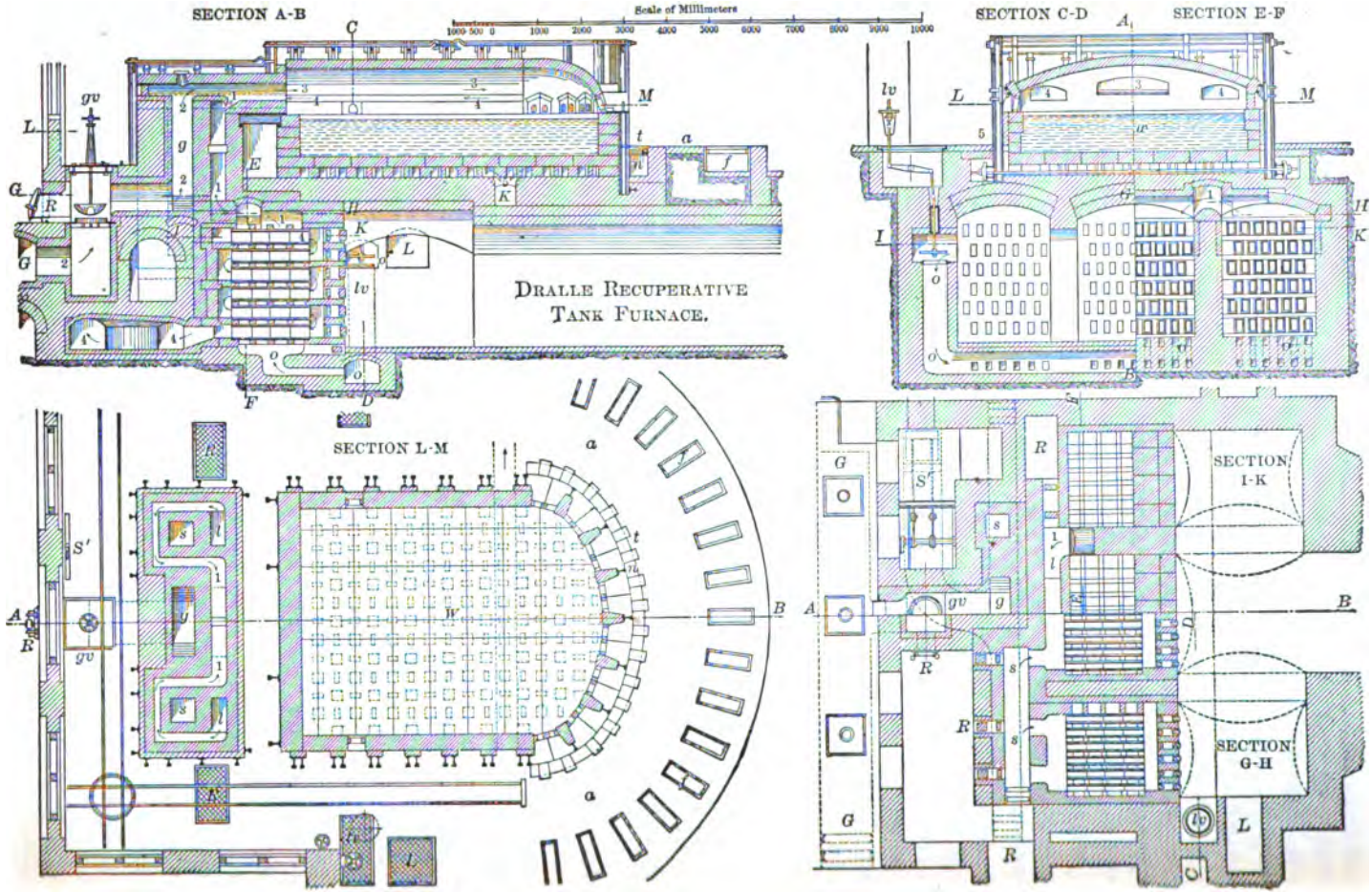
As a rule, the bottle tank is much shorter and wider than the window glass tank. Many are horseshoe shaped and others approach a semicircle. The Siemens Co., at Dresden, have adopted a circular plan for their furnaces and have a number of them in successful operation. It is claimed that the greatest economy in cost of metal is reached by the use of tanks whose dimensions of length and width approach each other, and in designing bottle tanks, economy should be the prime factor, as the finest quality of glass is not necessary for green bottles. There are few ports, often but two, of varied location. In some cases they are opposite each other, as in the window glass furnace; in others they are side by side at the end wall of the furnace; in the latter case the flame enters at one port, turns its direction and is drawn out of the other port.

M. Victor Bertrand, of the glass works at Nancy, has experimented on the use of the continuous tank furnace for making polished plate glass, and has embodied his results in a recent patent.*

The idea of using the continuous tank furnace for this purpose is not a new one, and many patents have been taken out along this line, mostly with the view of melting to the point of fining in the tank furnace, then pouring into pots which are taken to another furnace where the glass is fined and settled, subsequently removing the pots from the second furnace and then casting in the usual manner. Mon. Bertrand's idea is to extend the furnace beyond the tank proper and leave a space for the pots into which the glass is to be cast. This pot compartment being part of the furnace, is at the same temperature, and the glass flows freely into the pots without chilling. The pots are of a special shape and are mounted on strong iron carriages. Mon. Bertrand estimates that a tank furnace 75 ft. long and 18 ft. wide will produce as much glass as can be worked up by 1,500 h.p., and believes that the cost of manufacture will be lessened, and a more uniform quality of glass will be obtained.

Recuperative firing has been applied to some extent in bottle tank furnaces. Its use has been less extensive in this country than in Europe. In most cases the batch used in European bottle factories is much more fusible than

* *Moniteur de la Ceramique et de la Verrerie*, Dec. 1, 1899.


SECTION A-B
Scale of Millimeters
SECTION C-D
SECTION E-F
DRALLE RECUPERATIVE TANK FURNACE.
SECTION L-M
SECTION I-K
SECTION G-H

in American factories. The results obtained by the use of recuperative firing in Europe, with a comparatively fusible batch, have been very satisfactory. Fuel consumption, repairs and labor costs are low as compared with regenerative furnaces, and the quality of glass produced is satisfactory. One of the best styles of recuperative furnace construction is that designed by C. Nehse and Robert Dralle, shown in Figs. 8, 9, 10 and 11. The recuperator is tightly built of tiles that effectually prevent leakage; the flame enters through a port in the center of the back wall and is divided and drawn to down takes on either side. The waste gases in passing to the stack describe a tortuous course through the recuperator, and the air, in like manner, passes through a series of tortuous flues at right angles to the waste gas flues. For the furnace shown in the illustration Mr. Dralle gives the following figures:*

No. of gas producers	2	Height of stack	34·0 meters
Free grate surface of producer	1·6 sq. m.	Inside diameter of bottom	1·2 "
Heating surface of recuperator	175·0 sq. m.	Inside diameter of top	0·85 "
Melting surface of tank	33·0 sq. m.	Coal used per 24 hours	6·0 tons.
Cubical contents of tank	26·5 cu. m.	Average weight of bottles made per 24 hrs.	7·8 tons.

The object of the annealing apparatus is to cool the glass slowly, after it has been worked up in such a way that breakage or damage is avoided. This is particularly important when the annealed ware is in the finished condition.

Scratches offer the most trouble, as glass, particularly when hot, is easily scratched by contact with hard, sharp substances.

The annealing kilns or ovens are continuous or intermittent. The latter is a kiln or arch heated with gas, coal or wood, with a hearth on which the ware is laid. The kiln is filled to its capacity, sealed up, heated and allowed to cool slowly. In the case of hollow ware and flint kilns the firing is so arranged that heat is evenly distributed over the interior while the glass is laid in. There is a tendency to draw at the laying-in opening, which sets currents of cold air in motion over the bottom of the kiln, and hinders proper annealing. The draft is kept at a minimum during the laying-in, so that the kiln will be filled with the products of combustion. When the kiln is filled and sealed, the proper rate of cooling is accurately regulated by adjusting the draught with the stack damper. In annealing plate glass there is a separate kiln for each sheet cast. The kiln is heated to the proper temperature, the sheet is pushed in upon the hearth and the opening sealed tightly. Accurate regulation of the speed in lowering the temperature is effected by control of the draught as in the case of the hollow-ware kilns. Continuous kilns, or "leers," are replacing intermittent kilns to a considerable extent. Plate glass is annealed in kilns when the piece is large and heavy, as it requires very thorough annealing and could not well be handled on any of the leers now in use. Small sheets of rolled glass, hollow ware, flint glass and window glass are annealed in continuous leers. The flint or bottle leer consists of a long, narrow passage with firing at one end, and running through it are parallel endless chains of link belting with a series of iron pans which are filled, one after the other, at the hotter end of the leer and slowly carried forward to the other

* *Zeitschrift des Vereines Deutscher Ingenieure*, June 8, 1899.

end. As they advance they encounter a constantly lowering temperature and by the time they reach the end of the leer the glassware is cooled and annealed. The continuous leer has a considerable economy over the older intermittent style. The capacity of the factory is increased for the same space occupied and the same outlay for construction and fuel. Window glass is annealed continuously in connection with the flattening process. After the sheet of glass has been flattened it is laid in at one end of the leer, which opens into the flattening oven, and drawn slowly to the other end. Two constructions are in use, the Tondeur rod leer and the Zellers chain leer. The former consists of two sets of horizontal rods or bars, so attached to a series of shafts that one set may be raised or lowered below the other set, and slides freely in a series of sheaves. The glass is laid upon this movable set of rods and drawn forward the width of one sheet. The shafts are then revolved sufficiently to drop this set below the level of the other set, which receives the glass, while the movable rods are allowed to slide back to the original position and raised to receive another sheet. The chain leer (Fig. 12) consists of several strands of chain belt which slide freely along channel bar tracks; the belt has numerous links with rollers inserted. The glass is laid on the chain and drawn slowly to the other

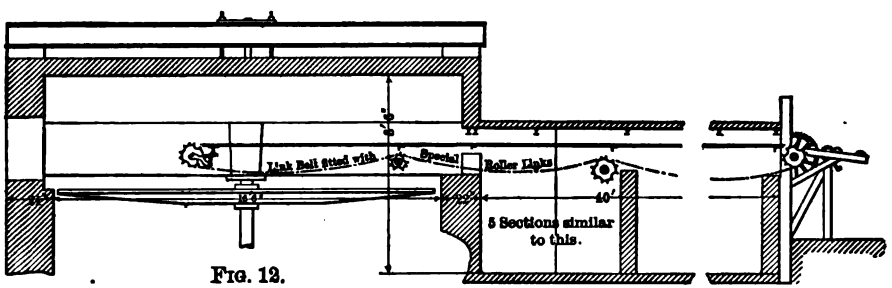


FIG. 12.

CHAIN LEER FOR ANNEALING WINDOW GLASS. T. A. ZELLER'S PATENT.

end. This leer is simple and direct in action; there is less mechanism to control, and the glass, always held at the same distance from the crown, is kept in a more uniform and evenly decreasing temperature than when it is constantly raised and lowered. Before the introduction of the continuous leers, it was believed that the annealing process was necessarily a very slow operation and that glass could not be annealed rapidly. Practice has entirely disproved this supposition, and it is interesting to note that the late investigations of Mr. Grenet on the subject of tension in glass are verified by the results obtained in the improved continuous leers.

The following recapitulation has been arranged to supplement the descriptions of the various manufacturing processes, and will serve to make them distinct:

Polished Plate Glass.—The pots in which the glass is melted have a flange halfway between top and bottom. When filled with molten glass the door in front of it in the side of the furnace is opened, the pot removed and carried by a crane to the casting table, which has a highly polished metal surface. The glass is poured upon the table and rolled flat with a huge metal roller. The

sheet is then carefully slid upon the hearth of the annealing kiln, the opening sealed up and the kiln cooled very slowly, giving the glass the proper temper. The casting and rolling leaves the surface of the glass rough and uneven, and after annealing, the sheet is taken to the grinding room and fastened to a table. An overhead revolving arm carries a number of rubbers, which are let down and rub the surface of the glass until it is level; abrasive materials, as sand and emery, are thrown upon the plate during the grinding. After one side is ground down level, the other side is subjected to a similar treatment. The sheet is now level, but the surface is obscured by the grinding. It is then put on the polishing table and is polished by similar mechanism with felt rubbers and rouge until it acquires a brilliant, polished surface.

The Manufacture of Plate Glass in Belgium has been so satisfactory that a few words on the subject are appropriate. The success is due to excellent technical control, cheap labor, and the prudent business policy of a selling syndicate. Each factory now has its own machine shop and all facilities for making repairs. The new Terneuzen factory has a complete electric plant for lighting and power, including electric cranes for casting, and other factories are now being similarly equipped. The pot room is frequently located underneath the casting hall, having the furnace flues running through it, which serve to keep the room at an even temperature so necessary for drying the clay work. In some factories the mixed clay is allowed to age for a considerable time before using; in others it is worked up quickly, with equally good results. The pots are made in moulds and carefully and slowly dried.

Usually the mixing room has a hard cement floor, on which the batch is laid and hand mixed two or three times before its removal to the furnaces. One of the successful mixes is as follows:

Components.	No. 1.	No. 2.
Sand	180	180
Sodium Sulphate	80	60
Sodium Carbonate	91	.....
Limestone	42	42
Charcoal	1·5	3·0

These mixes are used for making one melt of glass, No. 1 being first filled in and melted followed by No. 2 to complete the filling of the pots. The first mix, containing sodium carbonate, melts more readily and develops less "salt water" than No. 2. The first filling melts flat in about 5 to 6 hours; and about 5 hours additional is necessary to complete the filling; with a final 7 hours to clear up the glass and prepare it for casting.

Furnaces are generally built to contain 16 pots and are of the following inside dimensions: 15 m. long, 5·4 m. wide, 1·8 m. high; air regenerators, 5·2 m. long, 2·0 m. wide, 2·05 high; gas regenerators, 5·2 m. long, 1·3 m. wide, 2·5 m. high; gas flue, 1·0 m. diameter; gas ports, 0·33 × 1·35 m.; air ports, 0·35 × 1·50 m.

Between the regenerators is built a strong wall 0·40 m. thick. Each anneal-

ing kiln is constructed to contain two plates, except for extra large plates, when the kiln contains but one. They are direct coal fired or use producer gas.

It is very difficult during the summer to secure sufficient labor for the casting halls, consequently the greater part of the casting is done during the winter, and a stock of rough plate is accumulated which is ground and polished in the summer. This labor difficulty encourages the increasing use of machinery, particularly for the casting operation. All the factories have large engines, rarely less than 1,000 h.p. each; an amount sufficient to operate five large grinding tables, three large polishing tables and the smaller tables and machines.

Mon. Bertrand* measured the temperatures of annealing with a Richard Freres nitrogen pyrometer, and found that the temperature of the kiln when sealed up was 700° C.; one hour later it dropped to 600°; and after another hour to 500°. From this point the temperature diminishes more and more slowly until cold.

Rough Plate, Cathedral, etc.—These styles of glass are manufactured in a similar manner, omitting the polish. *Wire glass* for skylights is made by imbedding strong wires in the glass during the process of rolling, which adds to its strength.

Window glass is a blown article. A lump of glass is gathered on the end of a blow pipe, moulded and blown to a pear shape, which by heating, blowing and swinging in the air is elongated to a cylinder. This is cracked longitudinally and taken to a flattening oven, where it is again heated, flattened out on a smooth fire clay tile, polished with a wood block and annealed. It is stronger and less easily scratched than plate glass, as it has not been ground or polished.

Bottles and other hollow ware are blown in moulds. Plain round bottles are turned in the mould while being blown, others are simply blown out to fill in the side of the mould by main force of breath. The glass used is gathered on blow pipes, smaller in size than those used for blowing window glass. The work has heretofore been done by hand. Recently, machines for blowing fruit jars have been successfully operated, and their use will doubtless be extended.

Lamp Chimneys are blown either free in the air or in moulds, similar to bottles, or are machine blown.

Tableware and similar articles are pressed. A small lump of glass is gathered on an iron rod and dropped into a press holding the mould of the required article. A plunger operated by hydraulic power drops and presses the glass until it takes the form of the mould completely and retains the shape into which it is pressed.

Goblets and Thin Glasses are blown or pressed and the beautiful designs are cut into them by a rapidly revolving wheel. Lead glass is used for this purpose, as it is soft, cuts easily and takes a brilliant polish.

Colored glass and opaque glass are produced by the addition of various substances to the mix.

* *Moniteur de la Ceramique et de la Verrerie*, Dec. 1, 1890.

The principal colors are produced as follows:

Black is produced by using pyrolusite, copper, iron or cobalt oxides in excess; *blue*, by using cobaltic oxide or cupric oxide; the former in very small quantity—0·1%—produces a deep blue, the latter in small quantity gives a delicate sky blue; *green*, either by ferrous oxide, giving a dull green; by cuprous oxide, giving a bright emerald; or by chrome oxide, giving a beautiful grass green; *violet*, by pyrolusite, or black oxide of manganese, free from iron; *gray*, by neutralizing the violet color of manganese as above, with oxides of iron or copper; *ruby*, by the suboxides of gold and copper. The details of its manufacture are quite complicated; *orange*, by a selenate in connection with a reducing agent, or by a mixture of ferric oxide and pyrolusite; *yellow*, by ferric oxide, by cadmium sulphide, by silver chloride, by oxide of uranium; the uranium oxide gives in addition an exquisite iridescent effect to the glass.

In the manufacture of most articles blown from colored glass, the colors are not used solid but are "flashed." A small lump of the colored glass is first gathered on the blow pipe, and a desired amount of clear glass is added to the outside of the first lump. Sometimes the process is reversed, the clear glass being gathered first and the colored glass added to it. This latter method is economical, as it requires a smaller quantity of the more expensive colored glass.

Best results are obtained in coloring by using fine glass and pure materials. Except in the case of cobalt, iron and sodium sulphide, these oxides cannot be used with a batch containing salt-cake, as the carbon added to reduce the salt-cake reduces the oxides as well. Colors may be added to a salt-cake glass after the melt has been completed, but this method does not yield the best results. Lead glass dissolves colors more readily and shows a more brilliant effect than lime glass, and borax aids materially in dissolving and holding the colors. Success in color making is acquired by experience and careful observation only, and much pleasure is afforded in the pursuit of this line of work. To seek new shades and delicate tints, knowing that the colors once obtained are permanent, is a rare opportunity for the combination of artistic instinct and technical skill, and while we admire the rare old windows in many a famed cathedral of older lands, we cannot fail to recognize the fact that windows now designed and made by a few of our leading houses are equal, if not superior, to anything that has yet been produced.

Opaque glass is made by the use of phosphate of lime, fluorspar and feldspar, cryolite, or tin oxide. The opacity is probably produced by the evenly distributed suspension of those undecomposed substances throughout the mass of the glass. Opaque glass made with fluorspar or cryolite contains fluorine, and glass made with phosphate retains phosphoric acid with the silica. The use of cryolite, formerly quite general, is somewhat restricted at the present time, as the fluorine attacks the pots in which the melt is made and soon ruins them. Tin oxide is largely used in the manufacture of opaque enamels. Representative analyses of cryolite, fluorspar and phosphate glass are as follows:

Cryolite Glass.		Fluorspar Glass.		Phosphate Glass.	
	%		%		%
SiO_2	59·85	SiO_2	76·11	SiO_2	49·58
Al_2O_3	10·71	Al_2O_3	3·04	CaO	15·77
ZnO	8·13	CaO	3·78	Na_2O	12·48
Na_2O	14·07	Na_2O	9·21	K_2O	1·44
F	6·18	K_2O	4·26	B_2O_3	9·15
As_2O_3	1·06	F	3·65	P_2O_5	11·58

Optical Glass.—The essential characteristics of optical glass especially for telescopic lenses are: (1) Transparency, in order that the rays of light shall not be obstructed. (2) Homogeneity, to insure the unchanged direction of the rays of light. (If the different layers of glass have different degrees of tension the rays will be unequally refracted.) (3) Hardness, for grinding and polishing and (4) resistance to the action of the atmosphere, temperature and moisture.

The manufacture of optical glass is accomplished in a furnace, ordinarily built to contain one well dried pot which is slowly heated in a kiln for four or five days, and then brought into the melting furnace where it is heated in five or six hours to the melting temperature. A filling of broken glass is first made and when melted the sides of the pot are well glazed with it. The batch is added in parts and melted down repeatedly until the pot is full. The glass is then "stirred" by placing a potato on the end of an iron rod and forcing it to the bottom of the pot. The liberated steam and gases rise rapidly through the mass, and help to collect and carry off the gases liberated from the batch in melting. When the action has ceased the furnace is heated to its highest temperature and held at this heat for from 6 to 8 hours at the end of this time the glass is well fined, yet still contains a few suspended bubbles of gas. To remove them and to mix the contents of the pot thoroughly the glass is slowly and carefully stirred with a previously heated fire clay roller. It is tested from time to time and when no more bubbles are observed the melt is considered finished. Before stirring, the temperature has been allowed to fall a little, and during the operation it is gradually lowered, until the glass is quite stiff. The roller is then removed, the pot taken out of the furnace, placed in an annealing kiln, and slowly cooled for three days. When taken out of the kiln, the glass is broken up into a number of pieces, which are carefully examined; and the faulty ones with seams, bubbles or impurities are rejected. The good pieces are charged in disk-shaped clay moulds, and placed in a kiln where the temperature is raised slowly and the glass melted to the form of the mould. These glass disks are placed in an annealing oven and cooled very carefully and slowly. The temperature of the oven is regulated with great care, the thermometer showing the temperature also regulating the flow of gas for firing and in this way the manipulation is very delicate. The annealing is often extended as long as four weeks.

The annealed glass is tested by polarized light for tension, and if present the glass is again heated and annealed until it has been entirely removed. A temperature of 465° C. removes all tension, and at 370° C. the glass is so hard that no change in condition is possible, consequently the necessity for very

gradual cooling lies within a limit of approximately 95° C. The glass, finally free from tension or defects, is ground and polished; the greatest accuracy being required for the proper curvature for the lenses and absolute uniformity in all parts.

Compound Glass.—One of the most striking recent improvements in glass manufacture is the so-called "Verbundglas," or compound glass which originated at the works of Schott & Genossen at Jena, Germany. The purpose in making this glass is to counteract the effects of changes of temperature upon glass. The glass is made of two layers, one flashed over the other, each layer having a different coefficient of expansion. In ordinary chemical ware it is difficult to make a flask that will not crack when heated too quickly; if made of compound glass, however, the outer layer has a lower coefficient of expansion than the inner and expands more slowly.

At the ordinary temperature a slight tension exists between the layers, and when the heat is applied to the outside of the flask the outer layer becomes heated more rapidly than the inner, yet on account of its lower coefficient of expansion it does not expand more rapidly than the inner layer. As a result of this arrangement a rapid heating does not tend to crack the compound glass as much as if it were made of one kind only. Compound glass is very useful for lamp chimneys and globes; it is used also for thermometer tubes for very fine instruments, as the tension that exists between the two layers tends to minimize the irregularity in the expansion of the glass.

Foerster gives the composition of the inner layer of compound glass as: Al_2O_3, 5·00%; Na_2O, 11·00%; Mn_2O_3, 0·05%; B_2O_3, 12·00%; SiO_2, 71·95%.

This analysis is stated to be the composition of the boro-silicate thermometer glass as well. The composition of the Jena normal thermometer glass is as follows: SiO_2, 67·3% ; ZnO, 7·0% ; CaO, 7·0% ; Al_2O_3, 2·5% ; B_2O_3, 2·0% ; Mn_2O_3, 0·2%; Na_2O, 14·0%.

The Venetian Glass Industry at the present time is in a very flourishing condition. The principal products are: Novelties, bric-a-brac, mirrors, tableware, beads, enamels, and aventurine. The glass used is very soft, and can be fashioned into the daintiest of designs. The body of the glass is not as clear and colorless as other European flint, but the tone obtained which is neither entirely clear, nor yet cloudy, imparts a certain attractiveness, and makes an especially good setting for the brilliant colors, red, blue, violet, green and yellow, that are used in it to mark the decorations. The glass is rarely cut, etched or engraved.

The Venetians are the masters of the art of working glass when soft and they produce the most beautiful effects in glass that is spun or interwoven into lace designs, etc., so that the term "artists" used by the glass workers is well chosen. In the production of a single piece it is sometimes reheated fifty times.

The principal tableware factory—lime flint—is the Vetreria Veneziana, where all varieties of pressed ware, cut glass, novelties, etc., are manufactured. They manufacture also large quantities of mirrors using glass imported from Germany and Bohemia, which are silvered and framed only. The

manufacture of beads is still a most important speciality, and they are imported to all parts of the world. Considerable business is done in artificial gems.

The Venetian enamels are of the finest quality and are largely produced by the Salviati works; they are extensively used in the production of mosiacs. Analyses of a number of these mosiacs made by Dr. Schwarz, of Graz, are as follows:

ANALYSES OF VENETIAN ENAMELS.

Components.	Yellow.	Carmine.	Blue.	Violet.	Purple.	Green.
	%	%	%	%	%	%
SiO_2	51·74	46·95	49·90	52·08	52·40	42·53
Sb_2O_5	7·91	1·42	3·75	3·15	5·26	7·25
As_2O_5	.....	9·96	1·12	4·29	2·35	.....
PbO	19·94	18·98	21·45	9·53	4·04	32·21
Cu_2O	.....	.....	.....	0·22	.....	.....
CuO	.....	.....	0·67	.....	0·10	2·54
CoO	.....	.....	0·91	0·09	0·15	.....
FeO	.....	.....	1·12	.....	.....	.....
Fe_2O_3	0·70	0·63	.....	0·76	1·60	0·89
MnO	0·42	.....	1·59	.....	.....	.....
Mn_2O_3	.....	0·15	.....	5·28	11·50	0·23
CaO	4·42	3·80	5·50	4·97	7·61	2·72
MgO	0·91	1·40	1·26	0·86	0·86	0·52
K_2O	2·26	9·99	8·25	9·59	1·41	2·75
Na_2O	12·88	5·43	4·50	8·57	11·84	8·88
B_2O_3	.....	1·29	.....	.....	.....	.....

Opaline Glass for Art Windows.—The beauty of modern art windows lies in the fact that instead of painted designs the colors are furnished by various shades of glass carefully selected for the purpose. The first results in this direction were due to John LaFarge, who found that colored opaque and opalescent glass produced a curious and very beautiful semi-transparent effect.

In the manufacture of opaline glass the white opaque glass is ladled from the pot upon a slab, and colored glass of one or more kinds is ladled upon it; the whole is then pressed out with a heavy iron roller. The different glasses intermingle in irregular streaks, with thick and opaque edges and thin and more nearly transparent centers, affording every imaginable gradation of color and shade. There is a special variety called drapery glass, in which the colors are poured and rolled, producing a drapery effect.

In constructing windows of opalescent glass a sketch is first made indicating the design and the color scheme. From this a full-size drawing is made in which special attention is given to the lead lines which mark the design, no painting whatever being done except those places corresponding in the design to faces, hands or feet. Three tracings are then made of this full-size drawing, two on paper and one on glass. The paper tracings are cut up into patterns for the small pieces of glass which enter into the composition of the window, while the glass tracing serves as a transparent frame to receive the colored pieces as the window is built up. The glass is selected piece by piece, great care being given to colors and shades; when the assembly is complete it is removed and leaded. It is then set up to the light again and carefully inspected; any portions that do not have a sufficiently deep tone are "plated," that is, backed up with other thicknesses of glass, which enhances the beauty of the effects. The semi-opacity of the glass gives it a soft, silky effect, with a certain solidity of appearance. Artistic designers make windows of opaline glass that rival in beauty the famous ones of old.

GOLD AND SILVER.

THERE was a large increase in the world's production of gold in 1899, the details of which will be found in the subjoined tables.

Australia took the first place as a producer, followed by the South African Republic and the United States, which is now only third in the list. The world's production of fine gold amounted in 1899 to 15,108,804 fine oz. or $312,307,819, as against 13,874,322 oz., or $286,803,462 in 1898, showing an increase of 1,234,482 oz., or $25,504,357.

The production of gold may be expected to increase steadily for a number of years to come owing to the following causes: (1) Enlargement of existing plants; (2) discovery of new gold fields; (3) discovery of new deposits in old districts; (4) development of low grade mines discovered in the past, but unworked because of high cost, though now worked owing to better and cheaper processes, cheaper material and transportation; (5) working of placer deposits with steam shovels and dredges; (6) increased production of copper and other ores carrying gold. The exhaustion of old mines is probably more than offset by the ability to mine cheaply at much greater depth.

PRODUCTION OF GOLD IN THE UNITED STATES.

State or Territory.	1896.		1897.		1898.		1899.	
	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)
Alaska	99,444	$2,055,700	130,624	$2,700,000	136,430	$2,820,000	247,944	$5,125,000
Arizona	124,770	2,579,000	130,624	2,700,000	116,110	2,400,000	124,577	2,575,000
California	737,036	15,235,900	725,669	15,000,000	740,208	15,300,000	716,014	14,800,000
Colorado	719,264	14,867,971	947,249	19,579,637	1,138,584	23,534,531	1,282,471	26,508,675
Idaho	104,263	2,155,300	96,759	2,000,000	99,178	2,050,000	84,664	1,750,000
Michigan	1,800	37,200	(c)	(c)	(c)	(c)	(c)	(c)
Montana	209,207	4,324,700	217,534	4,496,431	253,890	5,247,913	233,127	4,819,157
Nevada	116,620	2,410,538	145,138	3,000,000	145,138	3,000,000	114,750	2,371,882
New Mexico	23,017	475,800	22,738	470,000	23,222	480,000	24,190	500,000
Oregon	59,313	1,226,000	65,534	1,354,593	58,862	1,216,669	61,684	1,275,000
South Dakota	237,978	4,919,000	256,410	5,300,000	276,730	5,720,090	280,600	5,800,000
Southern States (b)	12,785	264,300	12,062	249,737	12,731	263,153	13,062	500,000
Utah	91,908	1,899,900	89,305	1,845,938	114,777	2,372,442	169,631	3,506,582
Washington	19,626	405,700	21,754	449,664	29,028	600,000	36,284	750,000
Other States	1,413	29,200	3,136	64,795	3,760	77,722	2,164	44,725
Total domestic	2,558,433	$52,886,209	2,864,576	$59,210,795	3,148,642	$65,082,430	3,391,196	$70,096,021
Foreign	409,315	8,461,023	584,963	12,091,599	1,065,552	22,024,960	1,423,449	29,422,691
Grand total	2,967,737	61,347,232	3,449,559	71,302,394	4,214,194	87,107,390	4,814,645	99,518,712
Total domestic—kg	79,576	.....	89,092	.....	97,933	.....	105,471	.....
Total foreign—kg	12,731	.....	18,194	.....	33,142	.....	44,274	.....
Grand total—kg	92,307	.....	107,286	.....	131,075	.....	149,745	.....

(a) 1 oz. gold = $20·67; 1 kg. = $664·60. (b) South Carolina, North Carolina, Georgia and Alabama. (c) Included with other States.

GOLD PRODUCTION OF THE WORLD.

Countries.	1897.			1898.			1899.		
	Fine Ounces.	Kilo-grams.	Value.	Fine Ounces.	Kilo-grams.	Value.	Fine Ounces.	Kilo-grams.	Value.
America, North:									
United States	2,864,576	89,092·4	$59,210,786	3,148,642	97,932.9	$65,082,430	3,391,196	105,471·0	$70,096,021
Canada	291,583	9,068·6	6,027,016	662,796	20,613·9	13,700,000	1,018,371	31,674·6	21,049,730
Newfoundland	3,000	93·3	62,010	3,000	e 93·3	62,010	3,000	e 93·3	62,010
Mexico (a)	344,518	10,715·0	7,121,189	398,487	12,398·5	8,236,720	448,882	13,960·1	9,277,351
Central America	25,399	e 789·9	525,000	24,435	e 760·0	505,096	23,470	e 730·0	485,158
America, South:									
Argentina	6,661	207·0	137,700	6,661	e 207·0	137,700	6,661	c 207·0	137,700
Bolivia	16,617	517·0	343,500	16,617	e 517·0	343,500	16,617	e 517·0	343,500
Brazil	70,736	2,200·0	1,462,120	76,618	2,383·0	1,583,700	76,613	2,383·0	1,583,700
Chile (b)	68,102	2,118·0	1,407,623	60,000	1,866.2	1,240,000	54,657	1,700.0	1,129,820
Colombia	188,679	e 5,868·2	3,900,000	179,003	5,567·3	3,700,000	164,490	5,115·9	3,400,000
Ecuador	6,430	e 199·9	132,900	1,911	59·0	39,500	1,911	e 59·0	39,500
Guiana (British)	101,504	3,156·9	2,098,095	99,105	3,082·0	2,048,397	108,269	3,367·5	2,238,040
Guiana (Dutch)	39,982	1,025·8	681,784	27,532	856·0	568,808	26,972	838·9	557,532
Guiana (French)	59,860	1,861·7	1,237,310	79,547	2,474·0	1,644,260	80,072	2,490·5	1,655,088
Peru	22,506	e 700·0	465,220	31,572	c 982·0	652,593	31,329	e 990·0	657,905
Uruguay	1,863	e 57·9	38,506	1,863	e 57·9	38,506	1,863	e 57·9	38,506
Venezuela	51,152	1,591·0	1,057,379	48,226	e 1,500·0	996,900	46,619	e 1,450·0	963,670
Europe:									
Austria	2,174	67·6	44,927	2,299	71·5	47,520	2,299	e 71·5	47,520
Hungary	98,645	3,068·0	2,038,993	88,994	2,768·0	1,839,506	88,994	e 2,768·0	1,839,506
France	8,574	276·0	183,430	8,584	267·0	177,448	8,584	e 267·0	177,448
Germany (d)	12,089	376·0	240,890	3,569	111·0	73,771	3,569	e 111·0	73,771
Italy	10,160	316·0	213,014	6,041	187·9	124,878	6,041	e 187·9	124,878
Norway	32	1·0	665	32	e 1·0	665	32	e 1·0	665
Portugal	537	16·7	11,098	219	6·8	11,098	219	e 6·8	11,098
Russia	1,042,017	32,408·2	21,538,490	1,196,634	37,217·0	24,734,418	1,159,214	36,056·3	23,963,017
Spain	1,833	57·0	37,888	1,929	e 60·0	39,873	1,929	e 60·0	39,873
Sweden	3,643	113·3	75,299	4,048	125·9	83,672	4,048	125·9	83,672
Turkey	386	12·0	7,975	375	e 11·6	7,751	375	e 11·6	7,751
United Kingdom	1,691	52·6	34,962	306	9·0	36,321	484	e 15·1	e 10,000
Africa:									
So. Afric'n Republic	2,744,010	85,342·6	56,718,679	3,777,009	117,470·3	78,070,761	3,529,826	109,782·6	72,961,501
Rhodesia	*Nil.*	*Nil.*	*Nil.*	20,981	652·5	433,682	54,241	1,687·0	1,121,170
Soudan	2,701	e 84·0	55,880	2,701	e 84·0	55,880	2,701	e 84·0	55,880
West Coast	48,363	1,504·1	999,658	34,845	1,083·7	720,248	33,865	e 1,053·3	700,000
Madagascar	19,351	601·9	400,000	3,151	c 98·0	65,110	3,151	e 98·0	65,110
Asia:									
Borneo	5,369	167·0	110,977	5,369	e 167·0	110,977	5,369	e 167·0	110,977
China	321,296	e 9,992·8	6,641,190	321,296	e 9,992·8	6,641,190	321,510	e10,000·0	6,645,612
India (British)	353,147	10,983·4	7,299,554	375,704	11,684·9	7,765,807	405,683	12,618·2	8,385,467
Japan	34,509	1,073.3	713,300	38,260	1,190·0	790.826	58,055	e 1,805·6	1,200,000
Korea	52,927	1,646·1	1,094,000	55,432	1.724·0	1,145,769	55,432	e 1,724·0	1,145,769
Malay Peninsula	25,000	777·5	516,750	25,000	e 777·5	516,750	25,399	e 790·0	524,997
Australasia, 7 cols.	2,539,491	78,981·8	52,491,279	3,013,763	93,732·3	62,294,481	3,831,937	119,185·6	79,206,180
Other countries (f)	21,771	677·1	e 450,000	21,771	677·1	e 450,000	24,190	752·4	e 500,000
Totals	11,506,084	357,858·2	$237,833,984	13,874,322	431,515·8	$286,803,462	15,108,804	469,929·9	$312,307,819

(a) Figures are based on exports of ores, matte, etc., and coinage. (b) Computed from exports. (c) As reported by the *Statistique de l'Industrie Minerale*, 1899. (d) The statistics have been changed in the present volume so as to represent the production from domestic sources. The statistics given in former volumes represented the production of refineries. The output by refineries in 1899 was 2,605 kg. (e) Estimated. In all cases, unless specified to the contrary, the figures have been taken from official sources or have been collected by THE MINERAL INDUSTRY, directly from the producers. (f) Comprises Abyssinia, Servia, Persia, Dutch East Indies, Formosa, and Philippine Islands.

The world's production of silver in 1899 decreased by 4,528,999 oz., and was in 1899 174,723,363 fine ounces of a commercial value of $104,100,163. This decrease was due mainly to smaller output in Mexico and Spain, the output of the United States having increased somewhat. There was an increase also in Australia.

There was produced in the United States in 1899 3,391,196 fine oz., or $70,096,021 of gold, as compared with 3,148,642 fine oz., or $65,082,430 in the preceding year.

UNITED STATES.

The statistics of gold and silver production in the accompanying tables and compiled as to totals from returns by all the American refiners and exporters of gold and silver bearing material, and reports of the Bureau of Statistics of the

Treasury Department as to gold and silver imported in ores and bullion; but in the distribution of the totals to States where produced, statistics compiled by H. A. Lee, Commissioner of Mines of Colorado, have been used for that State,

PRODUCTION OF SILVER IN THE UNITED STATES.

State or Territory.	1896.		1897.		1898.		1899.	
	Troy Ounces.	Commercial Value. (a)	Troy Ounces.	Commercial Value. (a)	Troy Ounces.	Commercial Value. (a)	Troy Ounces.	Commercial Value. (a)
Alaska.....	150,000	$100,650	250,000	$149,475	150,000	$87,390	275,000	$163,845
Arizona.....	2,000,000	1,342,000	1,332,292	796,577	2,250,000	1,310,850	2,000,000	1,191,600
California....	600,000	402,600	757,300	452,790	650,000	378,690	600,000	357,480
Colorado.....	22,500,000	15,097,500	21,278,202	12,722,227	23,502,601	13,692,615	23,114,688	13,771,731
Idaho.....	5,400,000	3,623,400	6,000,000	3,587,400	6,284,744	3,661,492	4,800,000	2,859,840
Montana	15,720,000	10,548,120	16,807,346	10,049,112	14,818,662	8,633,352	16,850,755	10,039,680
Nevada	1,200,000	805,200	1,500,000	896,850	800,000	466,080	575,000	342,585
New Mexico.	700,000	469,700	350,000	209,265	450,000	262,170	435,000	259,215
Oregon	61,100	40,998	84,802	50,703	128,326	74,763	140,000	83,412
South Dakota	450,000	301,950	500,000	298,950	325,000	182,345	350,000	208,530
Texas.....	525,400	352,543	600,000	358,740	500,000	291,300	450,000	268,110
Utah.....	8,842,810	5,933,526	6,669,754	3,999,804	6,570,256	3,827,773	7,153,107	4,379,685
Washington..	274,900	184,458	242,781	145,159	275,000	160,215	300,000	178,740
Others.....	64,600	43,347	64,815	38,753	50,443	29,388	63,284	37,705
Totals ..	58,488,810	$39,245,992	56,457,292	$33,755,815	56,755,032	$33,065,482	57,126,834	$34,096,168

(a) The average value in 1896 was 67·1c. per oz., 59·79c. in 1897, 58·26c. in 1898, and 59·58c. in 1899.

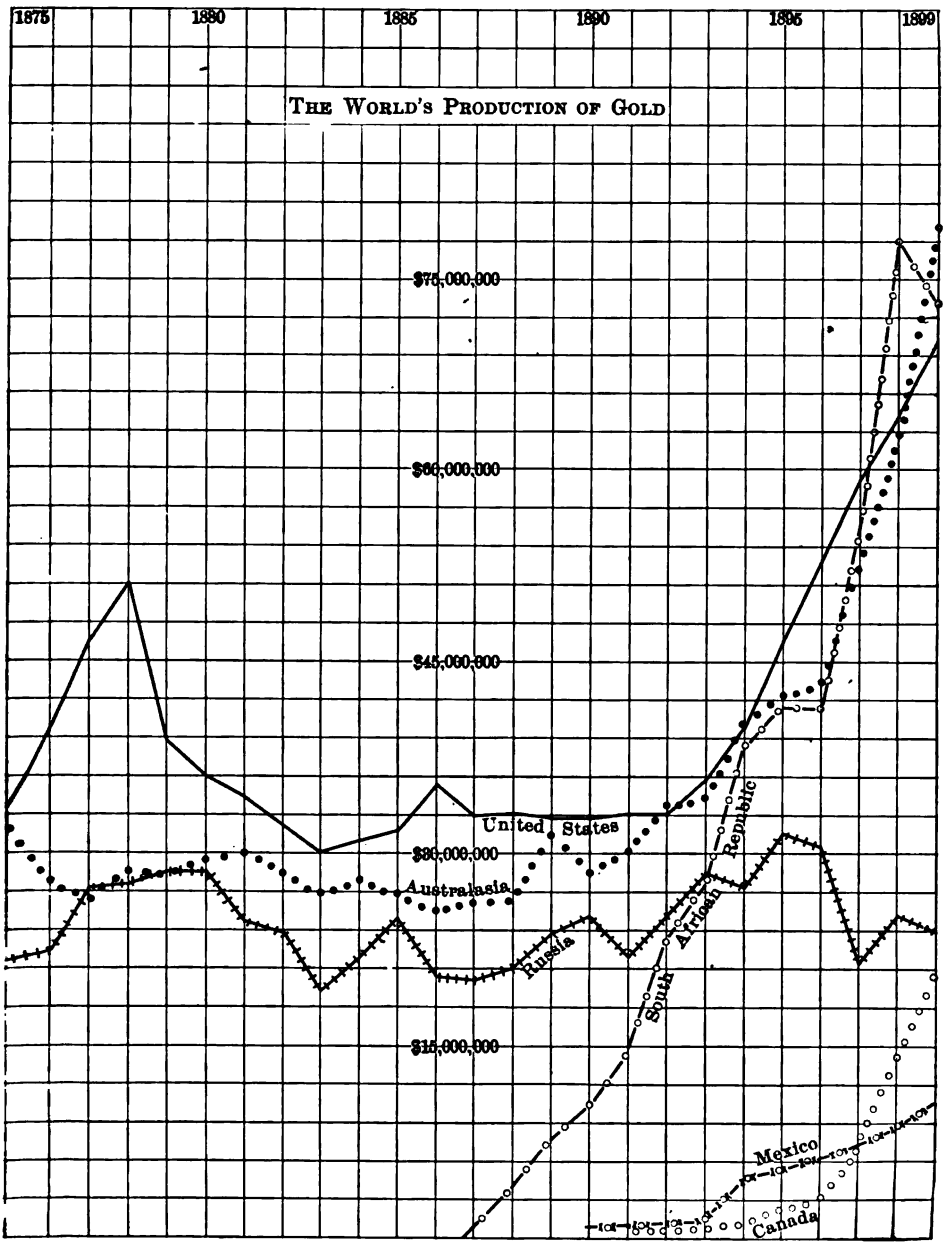
SILVER PRODUCTION OF THE WORLD. (a)

Countries.	1898.			1899.		
	Troy Ounces.	Kilograms.	Commercial Value.	Troy Ounces.	Kilograms.	Commercial Value.
America, North:						
United States.....	56,755,032	1,765,264·9	$33,065,482	57,126,834	1,776,829·1	$34,096,168
Canada	4,434,065	137,913·3	2,616,110	3,078,837	95,761·8	1,834,371
Mexico (a).....	56,859,076	1,768,501·6	33,546,855	55,032,838	1,711,699·1	32,788,565
Central America.....	1,623,575	e 50,500·0	957,909	1,446,795	e 45,000·0	862,001
America, South:						
Argentina.....	383,561	e 11,930·0	226,301	383,561	e 11,930·0	228,526
Bolivia	10,432,685	e 324,490·4	6,155,084	10,432,685	324,490·4	6,215,784
Chile (b).....	5,829,542	181,318·2	3,439,430	5,772,791	179,552·4	3,439,430
Colombia.....	1,646,131	e 51,200·0	971,217	1,646,131	e 51,200·0	980,764
Ecuador.....	8,100	e 251·9	4,779	8,100	e 251·9	4,826
Peru (a).....	5,781,552	179,824·0	3,411,116	5,725,270	e 178,074·4	3,411,116
Europe:						
Austria.....	1,295,843	40,304·9	764,847	1,295,843	e 40,304·9	772,068
Hungary.....	604,407	18,799·0	356,600	604,407	e 18,799·0	360,106
France.....	461,045	14,340·0	272,017	461,045	e 14,340·0	274,690
Germany (c).....	5,572,701	173,329·0	3,287,893	5,572,701	e 173,329·0	3,320,215
Greece.....	1,303,136	40,533·0	768,850	1,294,917	40,276·1	771,512
Italy.....	1,396,556	43,437·4	823,968	1,396,556	e 43,437·4	832,068
Norway.....	172,515	5,372·0	101,784	172,515	e 5,350·0	102,784
Portugal	3,842	119·5	2,267	3,842	e 119·5	3,481
Russia.....	278,515	8,668·0	164,324	260,809	8,112·0	155,390
Servia.....	18,326	570·0	10,812	18,326	e 570·0	10,919
Spain.....	7,362,579	229,000·0	4,343,922	5,448,019	169,451·0	3,245,930
Sweden.....	65,363					
Turkey.....	225,282	2,033·0	38,563	65,363	e 2,033·0	38,943
United Kingdom.....	211,393	7,007·0	132,916	225,282	e 7,007·0	134,223
Asia:		6,575·0	124,722	211,393	e 6,575·0	125,947
Dutch East Indies.....	1,286	40·0	759	1,286	e 40·0	766
Japan.....	1,660,213	51,638·0	979,326	1,660,213	e 51,638·0	989,154
Australasia.....	14,817,795	460,881·0	8,742,499	15,326,768	476,712·0	9,131,688
Other countries (d).....	48,236	e 1,500·0	28,453	48,236	e 1,500·0	28,733
Totals.....	179,252,362	5,575,335·5	$105,364,505	174,723,363	5,434,353·0	$104,100,168

(a) Statistics compiled from exports and coinage. (b) Represents exports of silver in all forms. (c) The figures are for silver produced from domestic ores only. The output as reported in former volumes of THE MINERAL INDUSTRY represents the total output by refineries. In 1899 this amounted to 467,593 kg. (d) The output is mostly from China and Persia. (e) Estimated. Unless specified to the contrary, the statistics have been taken from official sources or have been collected directly from the producers by THE MINERAL INDUSTRY.

while those of E. B. Braden, United States Assayer at Helena, have been used for the States of Montana and Utah.

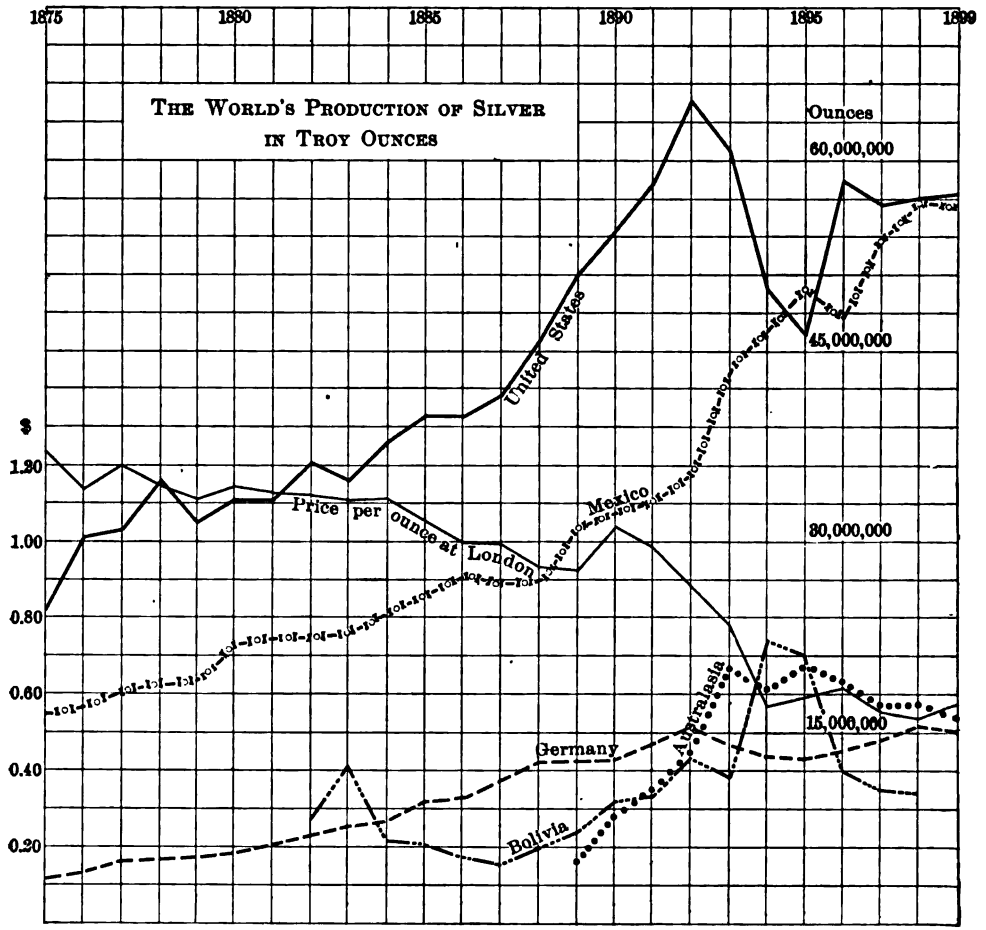
Alaska.—The production of this Territory in 1899 amounted to 247,944 fine oz., or $5,125,000 as compared with 136,430 fine oz., or $2,820,000 in the preceding year. Not only did the quartz mines of Douglas Island produce



well, but there was much activity in placer mining on the Yukon River, many disappointed workers at Dawson having crossed back into American territory,

and moreover the discovery of the marvelous Cape Nome beach placers has contributed to swell the output.

The Alaska-Treadwell Gold Mining Co. has 540 stamps. During the six months ending May 31, 1899, 250,408 tons of ore were crushed, most of which came from the 110-ft. level. The yield was $441,282·25 in free gold, and 4,912·9 tons of concentrates, containing $236,372·88, a total of $677,655·13. The yield per ton was $2·71. The operating costs were $1·28, divided as follows:



Mining, 77·32c.; milling and concentrating, 27·76c.; chlorination (3,196·6 tons), 6·76c.; sulphuret expense (1,718·57 tons), 6·1c.; general expense at Douglas Island, 3·84c.; office expense at San Francisco, 2·44c.; office expense at London, 0·45c.; office expense at Paris, 0·09c.; legal expense at San Francisco, 0·28c.; consulting engineer expense, 0·66c.; bullion charges 2·6c. The amount of ore in sight was estimated at 4,177,000 tons. Chlorination of the sulphates was discontinued, the concentrates being shipped to Tacoma, Wash., where they are treated by the Tacoma Smelting Co.

The Alaska-Mexican mill ran an equivalent of 357·08 days for the full 120 stamps out of a possible 363 days. It crushed 166,054 tons of ore, or 3·87 tons per stamp per day. The cost of milling, including that of saving the concentrates was 29·76 tons crushed. The operating expenses were divided as follows: Milling, 29·76c.; chlorination (497·1 tons, sulphurets) 2·99c.; sulphuret expense (2.948·69 tons) 15·5c.; general expenses, Douglas Island, 5·26c.; San Francisco office expense, 1·45c.; London office expense, 0·33c.; Paris office expense, 0·11c.; consulting engineers' expenses, 0·54c.; bullion charge, 1·32c., making a total of $1·6796, not including the construction of the new main shaft and repairs to a railway track, which increased the total to $1·7269. The net profit for the year was 37·54c. per ton of ore crushed. The output of ore was 166,034 tons, which yielded a total of $358,161·37, or $2·15 a ton. Like the Alaska-Treadwell concentrates during the latter part of the year were shipped to Tacoma, where they were smelted by the Tacoma Smelting Co. The costs amounted to $8·2647 a ton, divided as follows: Labor, $1·1388; treatment charges, $5; freight, $1·8204; unloading charges, 6·22c.; wharfage, 24·32c., as against $10 per ton when the sulphurets were chlorinated by the Alaska-Treadwell Co.

A gold vein was discovered on Taku Creek, 125 miles by rail and water from Skaguay, near the entrance to the Atlin country. It is said to be 8 ft. wide. Four tons of it taken to Juneau averaged $35·50 per ton, a considerable amount of concentrates running $87. By far the most important discovery in Alaska during the year was that of the rich beach placers of Cape Nome. Cape Nome is situated south of Cape Prince of Wales, which latter point is the nearest locality in America to Asia. The beach here is low, and averages some 250 ft. to low tide water. As the richest portion has proven to be the beach covered by low tide water, it is anticipated that sands which at present cannot be worked will prove highly remunerative when dredged. Back from the beach the *"tundra"* for some miles is rich in gold. The precious metal is contained in a thin stratum of reddish mineral close to the surface, and it is rarely that bed rock is more than 20 to 24 in. deep. The country back of Cape Nome is strewn with quartz float and it is possible that good quartz veins will be found in time. According to Paul F. Travers it was common for a man to take out by means of a rocker, converted from an ordinary soap box, as much as $25 a day. At the time of his visit 2,000 people were working side by side for 10 miles up the coast. The beach last summer was worked in three lateral lines and the lowest nearest the low tide line proved to be the richest, although the most difficult to work. Blankets were generally used to collect the gold from the rockers, but he found that Astrakhan cloth was the most efficient. The black sand which holds the gold looks like iron fillings, and being of high specific gravity it is almost impossible to separate it from the gold without the aid of quicksilver. The Government reserves the beach from low water mark to 60 ft. above high water mark as a highway. This is divided into claims and held by occupation. There was considerable comment on the action of the War Department in granting permission to certain applicants to dredge the beach below low water. The Secretary of War stated that unless he was

instructed differently by Congress he would relieve all citizens applying for dredging rights from the obstacles interposed by the United States Statutes, so long as their proposed operations would not interfere with navigation. It is thought by many Cape Nome miners that these dredgers would be bound to trespass on property now held by possession, and trouble is predicted.

Arizona.—The production of gold increased from 116,110 oz., or $2,400,000 in 1898 to 124,577 oz., or $2,575,000 in 1899.

The Congress mine, as usual, worked steadily. The McCabe produced a considerable amount of gold with some copper. The ore from here was shipped mainly to Silver City, N. M. The United Verde copper mine is one of the largest gold producers in the Territory. Ores were still shipped from Tombstone, and there was some talk of reopening the mines below water level. A fair portion of the value of these ores, particularly in depth, is in gold. Tailings piles in various portions of the Territory were worked by the cyanide process. The activity in copper mining will doubtless bring about a large increase in the production of the precious metals, both as some of the copper ores contain the precious metals, and owing to increased smelting facilities the reduction of a larger quantity of silicious gold and silver bearing ores will be brought about.

California.—The year 1899 was not so dry as the previous one, yet the production of gold did not show the anticipated increase, being 716,014 fine oz., or $14,800,000 against 740,203 fine oz., or $15,300,000 in 1898. Notwithstanding decreased production the industry, generally speaking, was in a most healthy condition. Considerable development work has taken place in Amador County, particularly at the Oneida mine. Here the Exploration Co. and Messrs. Seligman have opened up the vein at considerable depth and it is intended to sink to over 3,000 ft. Several mines are making preparations for deep sinking in this district. The Lincoln and Keystone mines are said to have improved their condition. There is some talk of opening up the Bunker Hill mine at Sutter Creek. There has been great activity in river dredging in Butte County, six dredges being placed in operation on Feather River, while others are being built. Arrangements have been made for transmitting electrical power to drive the dredges, thus effecting an economy as the wood consumption has been high. The Utica Co. at Angel's Camp in Calaveras County has run 180 stamps continuously during the year. Its electrical power plant supplies power to various other mines in the vicinity. The Melones Consolidated Mine at Carson Hill was actively developing and partially erected a 120-stamp mill. The Mariposa Commercial and Mining Co. has actively developed the Princeton, Josephine and Pine Tree properties on the Mariposa grant. Work has taken place on a large scale and expensive and fully equipped mining plants erected at various points. The company has 44 square miles of territory and a strike of over 10 miles of the mother lode. The Standard Consolidated Mining Co., in Mono County, has replaced its old mill with a modern structure. It paid $60,000 in dividends during the year. In Nevada County, the Champion, Providence, W. Y. O. D. and Pennsylvania mines have been operated satisfactorily. The North Star Mines Co. was formed to take over the business of the North Star Mining Co. on June 30, 1899. This company was originally incorporated

in 1884 with 100,000 shares, of which 40,000 were sold at $1 per share for working capital and 60,000 were issued for property arbitrarily valued and entered in the accounts at the same rate. In 1894 the capital was doubled and 100,000 new shares were sold for working capital at various prices, for the most part at $5 per share. The total production to June 30, 1899, was $3,446,295·08. Receipts from interest account amounted to $22,293·74 and from sundries $3,089·48. Dividends amounting to $534,850 were declared and a balance of $183,948·43 in cash and supplies was delivered to the North Star Mines Co. The Gaston Ridge or California mine was developed by a tunnel over 2,000 ft. long. The ore body is said to be 40 ft. wide and the ore to be of fair grade. Gold mining has been active during 1899 in San Diego County. A large cyanide plant was built at the Golden Cross mine. The Julian and Banner mines were worked continuously. In Sierra County four new mills were built, and quartz mining at Downieville, Forest City and Allegheny was active, several abandoned mines having been reopened. While the scarcity of water has prevented the successful operation of a hydraulic mine the heavy storms in the past winter give an assurance of a successful year's operation in 1900. The production in Siskiyou County was about $1,000,000 from placer mining and $600,000 from quartz. The largest producers are the Old Black Bear and Gold Ball mines as well as the Jilson, near Henley. At the Brown Bear mine at Deadwood in Trinity County, a tunnel over one mile in length is being driven to cut the vein 400 ft. below the present opening. The Maple Creek Mining Co. has purchased the Dutch Creek mine and is spending large sums in development and for erection of a plant. The Raw Hide and Dead Horse mine are being worked at a depth of 1,600 ft. in Tuolumne County. The Longfellow mine at Big Oak Flat has been satisfactorily developed. The Jumper mine has been a steady producer during the year. During the year 463 quartz and 63 placer mining claims were located in the county. In Yuba County, the Pennsylvania-Jefferson mine which had been abandoned, was reopened and developed systematically. It is now producing $15,000 to $20,000 a month.

Colorado.—Eighteen hundred and ninety-nine was the record year in this State, the production of gold having increased from $23,534,531 to $26,508,675. The greatest increase came from the Cripple Creek district, although Leadville's gold production showed a considerable advance. The production of silver decreased slightly, being 23,114,688 oz., against 23,502,601 oz.

Clear Creek County.—The production of gold decreased from $605,528 to $546,824·85, while the silver fell from 1,569,012 oz. to 1,502,900 oz. The silver production comes mainly from mines in the vicinity of Georgetown and Silver Plume. A remarkable thing in mining in this county is that the leasing or tribute system which was so popular some years ago, has been partially abandoned, miners preferring to accept the certainty of day's wages.

Gilpin County.—The product of Gilpin County in 1899 was $1,996,060·56 and 340,652 oz. of silver, being a trifling increase over the preceding year when $1,983,514 in gold was produced. The production during the latter half of the year was somewhat impaired, owing to improvements and extra machinery being erected at some of the chief producers.

Lake County (Leadville).—The production of gold in this county was valued at $2,196,497·55 against $2,073,036 in the previous year. Silver increased from 7,068,727 oz. to 7,230,118 oz. As in the previous year, the increase in the production of silver was due to increased activity in working the sulphide shutes between Carbonate and Iron Hill as well as the reopening of some producers which had been closed down. It is thought that the production would have been far greater had not a snow blockade occurred from which relief was not experienced until the end of May. Almost every mine in the camp was shut down for three months and had hardly begun to operate to their full capacity when the smelter's strike interfered again with their continuous operation. The Mahala mine, which had been a large producer, closed down during the year, it is stated on account of the low grade of its ore. The Home Mining Co. has opened up remarkably large bodies of ore at the Starr, Bon Air and Penrose shafts, after draining this portion of the basin. The Wolftone has been a steady producer. The Iron Silver Mining Co. had a successful year's operations, from 250 to 300 tons of iron sulphides being shipped daily. The zinc-lead-iron sulphide ore will be worked more actively during 1900, as an increase has been made in the concentrating plant. The Ibex mine has been worked actively.

Mineral County (Creede).—The production of silver decreased from 4,177,184 oz. to 3,796,899 oz. The Commodore mine was the largest producer. This property has paid dividends aggregating $432,000. It is stated that work will be carried out to relieve the Last Chance and Amethyst mines from water.

Pitkin County (Aspen).—The production of silver here increased from 3,977,270 oz. to 4,158,708 fine oz., while the gold decreased from $71,001 to $52,233·09.

San Miguel County.—The gold production, which came chiefly from Telluride, fell off from $1,572,677 to $1,376,704·68, and from 2,129,082 oz. of silver to 1,208,395 oz. These changes were principally due to a decrease in production of silver by the Smuggler-Union Consolidated Mining Co. and the decrease in gold produced by the Tomboy mine.

Teller County (Cripple Creek).—The production of this county increased from $13,507,349 in 1898 to $16,058,564·34. The largest producers were the Portland, Stratton's Independence, Gold Coin, Isabella, Strong, Jack Pot and Vindicator mines. The tonnage treated was 467,122. The total amount of dividends declared by public companies during the year was $4,322,000. Several mines joined the list of dividend payers. Among these were the Alliance, Squaw Mountain and Orphan Belle. Those which resumed dividends during the year were the Consolidated, Elkton, Isabella, Monument, Raven, Garfield, Gold King and Jack Pot. The works of the Economic Gold Extraction Co. near Victor were completed during the year. Its capacity is 350 tons daily. Rolls are used in crushing and the ore is roasted in automatic furnaces. The chlorination process has been adopted.

Georgia.—The Dahlonega Consolidated Gold Mining Co. has erected a 130-stamp mill with stamps weighing 850 lb. There are a double set of rock breakers, screens intervening between them. The final crusher delivers the ore in size of about 1-in. cube. The ore is then conveyed by belt conveyors to the ore

bins. Suspended feeders are used and the mortars are arranged for front and back plates. There are 54 sq. ft. of copper plate in front of each battery. Mercury traps and hydraulic classifiers are used; 48 Frue vanners are employed and chlorination works have been erected. In this a Holtoff-Wethey roasting furnace and automatic cooling floor has been adopted. The chlorinating barrels have a capacity of 5 tons and the plant is able to treat 30 to 50 tons of concentrates daily.

Idaho.—Very few changes have occurred in gold mining in Idaho during the year, so that the production has not materially differed, being $1,750,000 in 1899, against $2,050,000 in 1898. The production of silver from 6,284,744 oz. ($3,661,492) decreased to 4,800,000 oz. ($2,859,840).

The De Lamar Mining Co. still remains the most important gold mine in the State. From April 1, 1898, to March 31, 1899, it produced gold and silver valued at $449,547. The production of silver in the Cœur d'Alêne district shows a pronounced falling off, being 2,682,135 oz., as against 3,399,524 oz. in the previous year. One of the largest previous producers, the Bunker Hill & Sullivan Mining Co., was obliged to suspend operations owing to the destruction of their plant by strikers early in the year. A new mill was ready for operation in December. Stimulated by the present high price of lead, it is anticipated that the output will be increased during the present year. The company, however, paid $201,000 in dividends against $228,000 in the previous year. The De Lamar Mining Co., under the careful management of D. B. Huntley, worked with considerably lower cost during the year ending March 31, 1899, than in previous periods. The total cost, including developing work and all other expenses was $7·66 per ton as compared with $9·53 and $11·14 in the two preceding years. The cost divided as follows: General mining cost, including labor, supplies and its proportion of management expenses, $3·4176 per dry ton; prospecting cost, including only labor and supplies, $1·6361 per dry ton; cost of mining and marketing the shipping ore, $0·1430 per dry ton; general milling cost, including labor, supplies, repairs, experimenting and its proportion of management expense, $2·2665 per dry ton; mill clean-up, for labor, freight, smelting and refining charges on resulting product, $0·0805 per dry ton; other expenses, including taxes, insurance, freight and refining charges on cyanides, scouting for mines, etc., $0·335 per dry ton; total average cost, $7·8692 per dry ton. This is a reduction of $1·66 below that of last year. Analysis of milling cost shows that the total cost was $2·2665 divided into labor $1·0412, and supplies $1·2253. The supplies were further subdivided as follows: Chemicals, 62·86c. per dry ton; lubricants, 0·57c.; illuminants, 0·22c.; fittings, 0·28c.; tools, 0·40c.; iron and steel, 0·28c.; bolts and nuts, 0·04c.; blacksmiths' coal, 0·53c.; fuel, 29·46c.; pipe, 0·15c.; belting, 0·17c.; lumber, 2·56c.; sundries, 6·76c.; assay office proportion, 1·55c.; stables, 1·44c.; office and incidentals, 1·79c.; freight and expressage, 12·91c.; legal expenses, 0·56c.; total, $1·2253.

The Twin Springs Placer Co. added considerably to the production of Elmore County during 1899. This company owns 3,500 acres of placer ground along the Boise River, 42 miles from Boise. The main flume is 9 miles long, 6 ft.

2 in. wide and 7 ft. high, carrying 5 ft. of water. The grade is 10·8 ft. to the mile and the flume is estimated to carry 10,000 miners' inches. An inverted siphon bringing the water from Sheep Creek has attracted some attention. The length is 1,780 ft. The discharge end is 35 ft. lower than the intake. The lowest point on the "siphon" is 410 ft. below the intake. The upper diameter of the pipe is 6 ft. and the remainder 4 ft. The upper section is 0·375-in. steel; middle section 0·5-in.; bottom section 0·525-in., and the elbows 0·875-in. There are four expansion joints and air outlet valves are provided every 50 ft. of elevation. For three months of the year there was ample water for six giants with 8-in. nozzles under from 370 to 420 ft. head. The gold obtained is clean, of medium weight, a large portion being recovered in the sluices and is from 840 to 912 fine. The gravel is from 15 to 70 ft. thick with values well distributed from top to bottom. The gold varies from $0·15 to $0·70 per cu. yd. The company earned in 1899 nearly 8% on its capital of $1,500,000.

Montana.—The gold production of this State decreased from 253,890 oz. ($5,247,913) in 1898 to 233,126,717 oz. ($4,819,156·95) in 1899. The silver production, however, increased from 14,818,662 oz. ($8,743,011) to 16,850,754·85 oz. ($10,039,679·74). Yet there was considerable activity in gold mining and doubtless 1900 will show an increase in production. Not only have a number of new quartz mines been opened up, but there has been an increase in general dredging and alluvial work. There has been a pronounced improvement in the Libby district in Flathead County and the Bare Gulch mines in Clark County. An increased production of gold may be anticipated from Madison and Jefferson counties, as a smelter has been erected at Clinton Bridges. The increased silver production is doubtless due to the comparatively large increase in copper product, as well as to the stimulus of small establishments being started in various parts of the State. The Montana Mining Co., Ltd., for the half year ending June 30, 1899, treated 37,652 tons of ore, yielding $239,987 or $6·37 a ton, with an expense of $221,957 or $5·89 a ton, leaving a profit of $0·048; 28,459 tons of tailings were treated, yielding $77,144 at an expense of $49,744, giving a profit of $47,400. The expenses of the tailings treatment amount to $1·75 per ton, divided into transport and treatment $1·40 and redemption of cost of tailings plant $0·35, and a dividend of 6d. a share was declared on April 7, 1899, leaving a balance of £24,687. The slimes are still reserved for treatment by some process to be determined in the future; they average about $2·70 per ton. Several mines in the Winston district, 27 miles from Helena, showed important developments during the year. At the East Pacific mine a tunnel 2,035 ft. long struck pay ore at a considerable depth. At the junction of the tunnel and the vein a shaft is to be sunk 1,000 ft. deep. The ore is sent to the East Helena smelter. The Peck concentrating plant has been in operation during the past year. Much of the ore came from Rimini. On low grade ore the concentrator has been able to pay better prices than the smelters, though the concentrates are eventually shipped to the latter.

The New Elkhorn Mining Co. mined at its Montana mine 12,365 tons of ore during the year ending December 31, 1899; 1,700 tons of ore were sold to the smelters returning 532,429 lb. lead, 214,909 oz. silver, and 231 oz. gold, with

a total value of $104,946. The remainder of the ore was sent to the mill which in addition received 299 tons selected from waste dump. The mill report shows 11,424 tons crushed and panned, 2,144 tons of salt being used in treatment. The proportion saved was 83%. The product was 375,963 oz. silver and 244 oz. gold. The batteries were in service 223 days and the average amount crushed per stamp per day was 2·05 tons. The pans were in service 352 days. The cost of mining and of milling per ton of ore hoisted and per ton treated, respectively, was as follows: Superintendence, $0·7833; mine labor, $0·0269; ore house labor, $0·7246; fuel, $3·0993; explosives, $0·8090; timber, $0·1250; illuminants, $0·0768; general supplies and repairs, $0·5362; surface expenses, $0·1892; taxes, insurance and legal expenses, $0·2238; total, $15·5941. Milling per ton: Superintendence, $0·5217; labor, $3·3058; quicksilver, $0·8904; salt, $2·1406; fuel, $1·3973; chemicals, $0·0661; general supplies and repairs, $0·7214; surface expenses, $0·2034; taxes, insurance and legal expenses, $0·2423; total, $9·4890.

Nebraska.—At Crete 40 tons of gravel washed yielded $12, or approximately 30c. per cu. yd.

Nevada.—While the Comstock mines still produce a certain amount of gold the largest individual producer in the State has been the De Lamar mine in Lincoln County south of Pioche. A certain amount of production is to be credited also to Tuscarora and to the Philadelphia mine in El Dorado Cañon. The production at the Silver Peak mine has been tied up by litigation and has been confined to treating old residues by the cyanide process. It is anticipated that when the present legal questions have been satisfactorily settled a large production will be made in this part of Esmeralda County. An important strike was reported at the Chainman mine in White Pine County. It seems probable that this State will have increased railway facilities in a few years. If so, there can be hardly any doubt that both gold and silver production will increase to a remarkable degree.

New Mexico.—The Cochiti district came to the front during the year and several plants were erected for treatment of the ore, and several are in process of erection. It is proposed to work the placers on the San Pedro grant. A number of small discoveries have been made, but on the whole, with the exception of Cochiti and Pinos Altos very little has been done.

Oregon.—The mines in the vicinity of Baker City have been worked actively, and at several points dredging has been carefully conducted on the alluvial gold deposits. At Sumpter there are several promising prospects, and a smelter has been erected.

South Dakota.—According to the State Mine Inspector, the production in 1899 of various mines in this State was in short tons as follows: Homestake, 2,674,336; Highland, 936,000; Horseshoe, 562,100; Holy Terror, 900,000; Hidden Fortune, 200,000; Golden Reward Chlorination Works, 648,000; Golden Reward Smelter, 2,300,000; Black Tail Mines, 60,000; Rapid City Chlorination Works, 24,000; Allen, Small and Associates Cyanide Plant, 19,000; Spearfish Cyanide Plant, 63,000; J. R. Mill, 6,000; Central City Cyanide Plant, 8,000; Cockran Mill, 9,000; Placer Mines, 10,000; other mines and mills, 10,000; ore shipped

East, 70,000; total, 9,131,436. For the dividends paid by these various mines the reader is referred to the table in the latter part of the volume of dividends paid by American mines. Certain of the properties belonging to the Deadwood & Delaware Smelting Co. were sold to the Golden Reward Mining and Milling Co. The purchase price of the 3,000 acres of mineral ground and the smelting plant at Deadwood, which is taken over, is said to have exceeded $2,000,000. The Golden Reward smelter had a successful year's operation, although no dividends were declared. It charges $8·50 for ores which contain high silica contents. It returns 90% of the silver as well as the customary amount of gold. The smelter and chlorination plants handle 425 tons of ore per day; 500 men are employed. The Homestake mine, under the supervision of C. W. Merrill, has erected a cyanide plant capable of handling from 800 to 1,200 tons daily. A number of other small cyanide plants have been put up, which do not embody the best construction nor follow in their operations the most approved practice. Among them is a plant at the St. Elmo mine, at the mouth of the Black Tail Gulch for treating tailings, the Dead Broke, the Kicking Horse, and others in Central City and Spearfish. The refractory ores of this district are almost ideal material for cyaniding. The uncertain ore occurrence, however, precludes the erection of large plants.

The capital of the Homestake Co. was increased in July to $21,000,000 from $12,500,000 in order to control the Highland Mining Co., the Black Hills Canal and Water Co., and the Black Hills & Fort Pierre Railway. Dividends amounting to $963,000 were declared during the year. The capacity of the milling plant is being increased to 3,200 tons of ore daily. According to T. J. Grier, the superintendent, this supply can be continued for over 20 years, from the ore blocked out above the 800-ft. level. The Hidden Fortune mine, a new discovery, produced a number of carloads of extremely rich telluride ore.

Utah.—The production of gold in 1899 increased somewhat in Utah, mainly owing to the increase in ores smelted, since there are few direct gold producers in the State, except at Mercur. The total amount of gold produced, including that refined by the smelters, was 169,630,888 oz., valued at $3,560,581·66. The total production of silver in 1899 was 7,183,107 oz. against 6,570,256 oz. in 1898. The Golden Gate mine at Mercur, Utah, does not state its production, but it is understood that it exceeds that of the Mercur mine. This latter mine produced from ore treatment and re-treatment of tailings $472,999 in 1899.

The Ontario Silver Mining Co. sold 3,095 tons of ore and 7 tons of mill cleanings during 1898, which were valued at $177,088. Some of the company's tailings averaging 10·6 oz. of silver per ton were treated at the Marsac mill, yielding $66,726 in silver. The net loss for the year, however, amounted to $20,443. Very little work was done in the mine, though the ore stoped yielded the expense of maintaining the mine in proper condition. The flow of water through the long drain tunnel rose during the year as high as 8,000 gal. a minute.

The Daly West mine, of Park City, works the Ontario vein on the western extension of the "Daly," though the main source of ore supply is a contact vein between quartzite and lime. A shaft has been sunk to 1,440 ft. Ore was first cut on the 900-ft. level, where the main exploration was carried out.

Although the vein could have been reached by direct cross-cut within 300 ft. of the shaft, indications, which were followed have resulted in the cross-cut extending to a length of 1,500 ft. Late in 1899 the ore zone was cut on the 1,400-ft. level, 700 ft. north of the shaft. The vein here was strong and well defined but of low grade. The Anchor mine has been worked up to and beyond the boundaries of the Daly West, the Ontario-Daly fissure has been worked up to its eastern boundary. Mining operations, after some litigation, were resumed on March 1st; shipments began in June, which have varied from 400 to 650 tons per week, and the earnings are said to be fully $60,000 per month. Ore shipped averages 23·65% lead, 62·5 oz. silver, $1·20 gold, 3% copper, 13·5% iron and 25·24% zinc. The concentrates run 35·63% lead, 52·64 oz. silver, $1·05 gold, 1·74% copper, 13·5% iron and 13·30% zinc. The concentrating plant is equipped with a crusher, two Cornish rolls, 6 double Harz jigs, 2 Huntington mills and 6 tables. The Union Pacific has agreed to build a spur to the mine and arrangements with the Ontario Co. for an extension of its drain tunnel to the Daly-West will result in an increased economy in the handling of ore, concentrates and water.

Washington.—The Republic district has been actively developed during the year, and though at first the ore proved somewhat rebellious and the cost by the Pelatan-Clerici process which was used at that time was excessive, experiments have shown that by a modification of the ordinary cyanide process the ores can be reduced successfully. The production during the year has not been large, but the development work has exposed fair bodies of high grade ore. While no mines in the district have been found to equal the Republic a number of extremely promising prospects have been discovered. A 200-ton mill is in process of erection. The construction has been supervised, and the management will be undertaken by D. J. Jackling, formerly of the Golden Gate mine at Mercur, Utah. Experiments made by him lead him to guarantee a high extraction of both gold and silver. If results substantiate his judgment there is no doubt there will be increased activity in this and adjacent districts in the next few years.

Gold and Silver Mining in Foreign Countries.

North America.—*Canada.*—British Columbia.—Trail Creek, as in previous years, was the greatest gold producer in this Province. All the mines showed an increased production though a decrease in tonnage value. It is stated that the reserves in both the Le Roi and War Eagle mines are somewhat exhausted and as a consequence these as well as the Center Star and Hall mines, the affairs of which latter property are in such a condition as to demand reconstruction, shut down in February. It is stated, however, by some persons that the shut down was due to a desire to settle labor troubles which had been existent for some time.

The Ymir mine had a successful year's operation, the profits amounting to £20,031.

In a general way it can be said that the Slocan lead silver properties have all shown much lower average grade than was at first expected. This was due

largely to the fact that only such properties were developed which had been high grade on the surface, and as the occurrence is in shoots of varying richness, the averages shown by subsequent mining are lower than the original gouging. In fact the mining is not of a very advanced type and, probably, if the mining costs throughout the district were published they would show why low grade properties do not pay.

The erection of concentrators was a natural consequence, but there was no precedent except the Cœur d'Alêne practice. This is applied unconditionally to all Slocan ores where concentration is possible. The difficulties have been realized only, but not overcome.

The efficiencies are low, much loss being occasioned by gray copper (tetrahedrite) and the irregular quality of ores. The jigging is done by Harz or Cullom jigs. The English method of jigging with little sizing prevails throughout. Slime jigs are employed, treating material of such fineness that only pulp of extreme fineness goes to the round tables. Recently Wilfley tables have been introduced, but thus far all fine material has been treated without previous sizing. The light lead and heavy zinc material is of difficult separation throughout. During the labor trouble, which has closed down mining practically since June, 1899, the dumps milled at different places were of lower grade than any ore ever milled in a lead concentrator anywhere. There is no market for the zinc or siderite. Systematic development has been resorted to only in rare cases, it being thought advisable to proceed with caution in view of the natural difficulties of the country. As is well known much snow and irregular weather occur here. Timber is not scarce, but much is required in many of the properties.

Dawson.—The production of gold in the Yukon region in 1899 while difficult to ascertain exactly is estimated by the Canadian Geological Survey at 774,069 fine oz. ($16,000,000). Owing to the complaints of miners the Government deemed it advisable to reduce the royalties, and the amount exempted from tax was increased to $5,000 annual output on each claim. Government officers were prohibited even in their private capacity from locating any ground on Dominion land. According to J. B. Tyrrell the rocks underlying this well-known gold bearing area are micaceous and sericitic schists and quartzites with many large and small veins of quartz. In places the schists are also cut by extensive dikes of dark green basic rock, and by lighter colored acid porphyry, but it is not probable that these latter have an influence on the gold contents. The schists are undoubtedly sedimentary rock of Paleozoic, probably of Cambrian, age which have been highly metamorphosed, the quartz veins having probably formed in them during metamorphism. They are not greatly disturbed, but appear to lie in wide and gentle folds. Near the summits they are approximately horizontal. The Klondike schists are probably the S. E. continuation of the Birch Creek and 40-Mile schists which have already been traced to some extent in Alaska. Further south in Canada they have not been clearly followed, though it is not improbable that many of the schists, quartzites and limestones underlying the country along the Dalton trail and westward to near White River are of approximately the same age. The Kasha River

country is probably underlain by mica schists not unlike those of the Klondike regions. As yet very few traces of gold in these rocks have been found. The gold is concentrated in the recent sand and gravel on the banks of rivers and in the beds of the smaller streams as well as in stream terraces and moraines on the sides of the valleys. The gold is scattered through the gravel, but is more especially within a short distance of bed rock, where nuggets or particles of gold varying from 10 oz. in weight to very minute scales are obtained. The decomposed rock covering the hillside contains a certain amount of gold and this can in many cases be washed and recovered by hydraulic processes.

Mexico.—The gold production in Mexico in 1899 was 448,832 oz., valued at $9,277,351, an increase over 1898, thus fulfilling the prediction in an editorial in the *Engineering and Mining Journal*, January 28, 1893, that should the railway systems of Mexico be extended the production of Mexico would startle those who believed that Mexico has no gold mines. Part of the gold produced in the Republic has been through the recovery by the several smelting companies, but that recovered by cyanide process and amalgamation, particularly the former, is becoming larger continually. The El Oro district in the State of Mexico, practically on the border of Michoacán, has attracted much attention and though the projected flotation of the Esperanza mine fell through, the El Oro mine, a very large property, has been taken over by substantially the same people. It is believed that a consolidation will be made by several companies in this locality with a view to working in a more economical manner. Both the El Oro and Esperanza mines have complete plants, while on some others extensive machinery is being erected. The Grand Central Mining Co., of Sonora, showed at its last annual meeting a net profit of £92,633, leaving a balance with the amount carried over from the preceding year's earnings of £115,976. After spending £9,617 on improvements there was a credit balance of £106,858. Dividends of 2s. per share or 10% were declared in December, 1898, and in March and July of 1899. The manager reports there are still 105,000 tons of ore in sight. A number of other new properties have been opened up in various parts of the Republic, particularly in Sonora and Sinaloa. Silver production has fallen off somewhat, being 55,032,838 oz. in 1899, against 56,859,076 oz. in 1898. This is probably due to the lead fluxing ores running somewhat lower in silver than in former years. Railway extensions, which seem imminent now in several directions will probably bring about an increase of production in future years.

Nova Scotia.—The production of gold in Nova Scotia during the year was 27,772 oz. There was little activity in gold mining as many mines lay idle owing to their owners' anxiety to sell rather than develop and work them. At Wire Harbor it is said that the Ploud Lode has been struck with its original richness. The Tanjier mine shows somewhat favorable results. At Waverley prospects seem promising. Some prospecting has been done at Renfrew, Uniacke and Oldham. Some attention has been paid to the low grade ores of Moose River and at the Dufferin mine on Salmon River a large plant has been erected.

E. R. Faribault of the Geological Survey of Canada draws a parallel between

the ore occurrence in Nova Scotia and that in Bendigo, Victoria. He believes that during the folding of the measures that rich quartz and large saddle lodes of quartz were formed at favorable places along the planes of the bed on the anticlinal domes of the folds. There can be no doubt, he says, that slate is the most favorable country rock for gold segregation, and that the presence of this metal in payable quantities will more or less depend upon the occurrence of this rock. The saddle reef deposits at the Salmon River, Richardson and Mooseland mines have attained a maximum width of 15 to 25 ft., while other bodies not at present being worked, are of even greater width. Many of the pay streaks have been proved rich and some have been traced from the surface along the gentle incline for as much as 1,800 ft. with extraordinary uniformity. At times two or three pay streaks have been determined in the same vein in the pay zone, lying parallel and under one another.

Ontario.—The Sultana mine at Bald Indian Bay, Lake of the Woods, was floated in England during the year. An interesting work was undertaken in sinking in the lake, and it was proposed to increase the stamping capacity.

The veins of Bag Bay, Ontario, show an interesting occurrence* of gold in small quartz veins though the impregnation extends to considerable distance in altered granite. The wall rocks are granite. Felsite dikes charged with pyrites are generally present in the vicinity of the veins, and the felsite shades frequently into phonolite. The altered granite which MacKellar terms mikadoite, resembles sericite. This is greenish white to green, massive and unctuous and has a tendency to shade away into quartz. Throughout this altered granite are quartz and occasionally chlorite seams. The ore is not free milling but is cyanided satisfactorily.

Alluvial containing gold is found on the banks of the Vermillion River and though preliminary tests showed the gravel to be low grade, J. W. Evans† thinks it probable that some valuable deposits may be found. In the majority of deposits the gold is found in the first 10 or 12 in. of soil, which is a reddish sand composed of quartz, granite and green stone particles and containing garnet and magnetic iron with some coarse gravel. The gold is very fine, running as high as 200 colors to the pan though assaying only $0·15 to $0·60 per ton. If the deposits are worked it is probable that some difficulty will be met with in saving the gold. It will be necessary doubtless to use undercurrents of large area. There is some doubt, however, that the deposits will be of a great extent, as from the composition of the sand it would seem that it was the result of surface weathering of granite and other rocks.

SOUTH AMERICA.—*Brazil.*—According to E. Hussak,‡ the official geologist of the State of Minas Geraes, the second mine in Brazil, ranking after the Morro Velho only, is the Passagem mine of the Ouro Preto Co., which from 1884 to 1893 yielded 76,360 oz. of gold from 287,626 metric tons of ore. The vein is composed of quartz and tourmaline, the former being almost milk white; mispickel, pyrite and pyrrhotite are accompanying minerals. It may be considered a bedded deposit, being parted from quartzites by a salband 39 in. wide of graphitic schist

* *Transactions of the American Institute of Mining Engineers*, February, 1899.
† *Transactions of the Canadian Mining Institute*, Vol. II., 1899, p. 105.
‡ *Zeitschrift für Praktische Geologie*, 1898, p. 345.

on the foot wall, while that on the hanging is rarely graphitic. The salband on the hanging wall is overlain by a bed of itabirites while above this again is a brown porous conglomerate locally called *canga.* The ore occurs in lenses varying between 6·5 and 50 ft. wide. It is poorer where wider and also when more silicious. The main portion of what may be called ore is probably of intrusive origin, breaking through the metamorphosed sedimentary rocks (quartz schists) and forming a distinct contact zone on both walls. A peculiar feature of the ore occurrence is that a considerable amount of bismuth is amalgamated with the gold and as no bismuth mineral has been found it is believed that the mineral is a natural alloy of gold and bismuth.

(By J. C. Branner.)—There was a time when Brazil led the world in the production of gold. Its production came almost exclusively from placer mines which have long ceased to yield any large output. The amount of rock mining done by the Brazilians was very small, the greater part of the total amount of rock mining having been in the hands of the English, German or French companies. The earliest and one of the best paying mines owned by an English company was the famous Gongo Soco mine in the Province of Minas Geraes between Santa Barbara and Morro Velho. Shortly after this, the mine, having ceased to be profitable, was abandoned. The St. John del Rey Mining Co., commonly known in Brazil as the Morro Velho Co., was formed in London in 1830, and the Morro Velho mines were opened in 1834. This is to-day, and has for many years been, the most important gold mine in Brazil. During a recent visit to Minas Geraes, a gentleman who is a reliable authority in regard to the mining business of that State, informed the writer that of "the three most important gold mines in Brazil, one of them was paying well, another was not paying at all, and the third was sometimes paying and sometimes losing." The paying mine referred to in this statement is the Morro Velho mine. There are in all about twenty gold mines in operation in the State of Minas Geraes. Of these three are placer washings, and the remainder are quartz and itabirite mines. Most of these mines are small and the methods of working them are antiquated and often not adapted to the ores. This is not true, however, of the large mines in which the best machinery is used and the treatment of the ores is up to modern standards. The gold mining State, *par excellence,* is Minas Geraes, but gold is by no means confined to that State. Indeed there is hardly a single State in which it is not found. The difficulties to be encountered in gold mining enterprises in that country are great or small according to the acquaintance with the people, the customs, the laws and the language. At a distance, these matters seem of small importance, but on the ground they can only be successfully met by men of experience, judgment and education.

Chile.—Guanaco, formerly in Bolivia, but now by the fortunes of war in Chile, is described by Sidney H. Loram.* The field, which contains more than 100 mines, is 80 miles by railway northeast of Taltal, in a waterless desert. Supplies come from southern Chile via Taltal, and water is piped 20 miles from the Cordillera, and commands when it gets there, an average price of $7 paper or $2 gold per 1,000 liters.

* A paper read at the meeting of the American Institute of Mining Engineers, September, 1899.

Through the country rock, which is called porphyry and syenite by Mr. Loram, are many intrusions of trachyte and there are also evidences of strong hydrothermal action leaving an extremely irregular ore deposit. The quartz, which is the main vein filling mineral shows evidence of having been shattered and recemented by heavy spar, quartz, trachyte and a distinctly later deposition of quartz.' Though the "Mantos" or irregular ore bodies rarely extend below 120 ft. from the surface, scattered bunches of rich gray copper have been found from 270 to 330 ft. deep. The gold is extremely fine and can be seen but rarely, though the ore may be rich. Mr. Loram comes to what seems a correct deduction that the gold existed in the quartz prior to its brecciation. A second form of gold which obviously is a precipitate from a solution, is found in spangles on the surface of the barite crystals. The auriferous solutions also penetrated porphyry and trachyte to a small extent.

Las Bordas, in Atacama, has a fair producer in the Elisa mine which yields annually about 350,000 oz. of silver from some 12,000 tons of ore. The vein is about 5 ft. wide on an average, and the minerals are comparatively few, cerargyrite predominating, with occurrences of the rarer minerals of silver and mercury. The vein is worked to a depth of 1,620 ft., on the dip the hanging wall being a soft sandy tuff (locally called *Cantera*) while the foot wall is an amygdaloidal augite-porphyrite, which the miners term *Manto Negro*. The vein stuff is a talcose decomposed rock never more than 10 ft. thick, called in miners' parlance *Mantos Bordos*. The silver deposition seems to occur on lines where the most crushing has happened.

Colombia.—The production of the Frontino and Bolivia mine up to June 30, 1899, showed a falling off, the returns being £55,223 against £61,933 for the previous half year. Work on the property has been interrupted considerably by the revolution; the directors take a hopeful view of some of the development work now going on.

Ecuador.—An American company is working in the district of Zaruma, which was discovered about 1549, but which was abandoned after the wars of Independence. The veins are quartz which yield about $5 a ton to amalgamation, leaving $11 a ton in the tailings, part of which is recovered by cyanide treatment. There are a number of undeveloped mines in this district, and it is stated that with good communication they would become highly productive. The average grade of the ore is said to be from $15 to $20 a ton and the width of the veins from 40 in. to 3 ft. The district has water power, and forests of good wood.

AFRICA.—*Madagascar.*—According to F. W. Turner, Acting Vice-Consul,* gold was discovered in the Imérina some 14 years ago and from that date until after the French occupation of the Island, a considerable amount has been taken out of the ground, notwithstanding the fact that under the Hova rule the unauthorized search for the precious metals was (in the case of natives) punished by a long imprisonment in chains, and Europeans were not allowed to prospect. Alluvial gold is to be found all over Madagascar in more or less paying quantities, but no serious prospecting for reefs has as yet been attempted. In spite of the severe

* *Diplomatic and Consular Report*, No. 2334, Annual Series.

penalty imposed on the natives for embarking in the gold dust trade, at least 1,800 oz. were dispatched monthly from the capital to Tamatave for shipment, and probably the combined exports from Mananjary and Majunga exceed that weight, to say nothing of the dust exported by the Arab-Indian traders from the less known southwestern parts.

The value of the gold production, as given by the French Commission of the Mineral Industry, was $400,000 in 1897, and only $65,000 in 1898. Contrasting the value for 1898 with the estimated value of the 21,600 oz. or more obtained yearly from the country according to F. W. Turner, under date of August, 1899, $400,000 presents a variation so wide that it is impossible to reconcile the conflicting statement. Perhaps the surreptitious manner in which a great part of the gold was obtained would prevent the appearance of its production in the published statistics of the French Government.

Rhodesia.—The gold production of Rhodesia during the year, according to the Rhodesia Chamber of Mines, was 65,303 oz. 13 dwt., which came from the Geelong Gold Mining Co. and Antenior gold mines, Selukwe Gold Mining Co., Filabusi gold fields, Criterion Development Co. and the Tati Concessions, as well as from small producers. The greatest month's production was in March, when 6,614 oz. were turned out. The Chairman of the British South Africa Co. stated at the annual meeting held December 15, 1899, that mining was proceeding in Rhodesia notwithstanding the Boer war. He stated that the outlook was promising, as the labor difficulties which had been prominent had disappeared and that the railway had provided facilities for cheap and profitable working. He added that there was no longer room for doubt that a great mining industry would be established shortly owing to the assistance which would be afforded by the large funds at the command of the company.

Zambesia.—Dr. Karl Peters, the explorer, reporting to the company named after him, stated that he believed he had found near Inja-Ka-Fura, the ancient Ophir. The ruins there, he states, are of a decidedly Semitic type. The surface indication of the quartz lodes in the diorite formation is said to be promising. A number of claims were located within 12 miles of the Zambese River. The conditions for mining are said to be favorable.

South African Republic.—The production for 1899 (4,069,166 crude oz., valued at $72,961,501) fell below that for 1898 (4,295,602 crude oz., valued at $78,070,761). Had hostilities not begun and had the greatest uneasiness not been felt during September and before war was actually declared, 11th of October, there can be no question that the Rand, exclusive of other producers in the South African Republic, would have stood as the world's first gold producer. Estimating conservatively that 450,000 oz. monthly would have been produced had affairs remained in a normal condition, the 3,913,810 oz. produced in the first nine months would have been swelled by the end of the year by 1,245,314 oz., which would have made the gross total produced by the mining companies 5,315,470 oz. Unfortunately the war lessened this to the figure first cited.

Mining conditions have remained unchanged. There has been greater economy shown in the reduction of working force, which many technical men

consider inadvisable. Mechanically there have been improvements reducing the cost of handling ore and the separation of low grade waste material from the ore before milling, the economical treatment of slimes, and the general adoption of methods for the treatment of this material have not only decreased cost but increased production. The deep level mines concerning which a few years ago there was some doubt as to their ultimate success, have proven remarkably successful, and it is now thought that what would have been considered deep level mining a few years ago will be comparatively surface work to that which will be executed in future years. There are some engineers who believe that an ultimate depth of 10,000 to 12,000 ft. vertical will be reached. From the standpoint of the mine owners of the Transvaal a statement made in 1899 by Geo. Rouliot, which represents the view of the Chamber of Mines, is of extreme interest. The following is a condensed abstract of it on the questions which were pertinent.

The native labor association has been somewhat handicapped by independent speculators, who have raised the cost of premiums to nearly 1s. per ton of ore milled. In 1898, 88,500 natives were employed, and fully 12,000 more were required. Intermediary handling of these men has raised their cost, speculators having purchased them with a view of future profits in disposing to the companies. Mr. Rouliot acknowledges that valuable aid has been given the companies by the State Attorney, and that owing to restrictions of the pass law desertions or escapes are comparatively less. Great trouble has occurred through the accessibility of liquor to the Kaffirs; the quality is poor and the results demoralizing. Mr. Rouliot states there is no intention of introducing the Kimberley compound system in Johannesburg, and furthermore no intention of being the sole purveyors of liquor to the natives. Reductions have been made by lowering the coal rate and by changing certain articles previously held under a high tariff to a lower class, yet the benefit to the mining industry has not been large. Mr. Rouliot points out that the Netherlands Railway charge 2·08d. per ton per mile for coal, whereas in the Free State and in Cape Colony it is 0·59d. per mile, and in Natal 0·94d. per mile. He does not see how legitimate costs can be so high considering conditions are about equal. Although the Government was memorialized to cancel the monopoly in the manufacture of explosives it has not seen fit to take action. The price, however, was reduced 10s. a case, half being borne by the Government and half by the monopolizing company, while the Government relinquishes £62,500 on a basis of estimated consumption; yet the benefits are apparent only, as taxation in other directions has been increased. Mr. Rouliot states substantially that the Government is disingenuous in making this reduction, and moreover, that the Government at the expiration of 10 years, being 15 years from the establishment of the company, has the right to take over the factories and its business.

Mr. Rouliot points out, by a careful analysis of results on the Rand, that the amount collected by the Government will amount to nearly 1% on the issued capital, and that the expense will amount to 9·3d. per ton of ore milled. The Chamber of Mines consider this tax unjust, as without it the revenue of the country was approximately £3,500,000; sufficient, in his estimation for all

purposes. Mr. Rouliot defended the action of Johannesburg financiers in not subscribing and endorsing the Government loan which was issued in 1898. He states that it was owing to the poor financial policy and to the uncertainty in the minds of investors, owing to the extreme powers conferred upon the First Volksraad in modifying or changing existing laws; moreover, no statement was made as to the disposition of the funds sought to be raised. If the Government had commenced by placing the control of its affairs in expert and authorized hands, all would have worked cheerfully to the end.

One of the most important companies in South Africa is the Consolidated Gold Fields of South Africa, Ltd. Its annual meeting was held November 14, 1899. For the year ending June 31, 1899, a profit of £1,006,000 was reported. There were some changes in the capitalization of the company owing to the increase of 550,000 ordinary shares. Over 2,000,000 ordinary shares had been converted to warrant to bearer in order to effect transactions on the Continent. The greatest producing companies owned by this consolidation are the Simmer and Jack Proprietary and the Robinson Deep, which in themselves return dividends amounting to £415,000 to the parent company. An interesting address was delivered by John Hays Hammond, in which he predicted greatly increased profits for the company, provided labor was obtainable and that certain conditions existing during the Transvaal administration were changed after the war. Technically he announced himself strongly against machine drills in stoping; he stated that there would be a considerable reduction in the expenses of white labor, not due to decreased salaries but to increased efficiency. If England is successful in the existing war, which seems certain, drunkenness, which causes inefficient labor among the natives, and illicit gold stealing and buying would then be eradicated, owing to the precautions which would be adopted by the companies. He thinks that a saving of 6s. per ton will be made, comparing the costs of 1901 with those of 1897.

A complaint of those interested largely in Rand mines is the excessive cost of dynamite there. According to a report from the late United States Consul Macrum from Pretoria, some £600,000 has been invested in a factory near Pretoria for the manufacture of explosives. Some £700,000 to £800,000 of stock is carried at all times. The company has the sole right to import, manufacture and sell explosives for the further period of about nine years; the Government receives a royalty on all sales. The mining companies pay now about £3 15s. for a No. 1 dynamite, but they claim that if the monopoly were cancelled it could be delivered at the mines at £2 5s., including a reasonable duty on the importation of the article.

The Rand Central Ore Reduction Co. passed its usual dividend, though it reduced its indebtedness to the extent of £30,000. A considerable amount of slimes which had been previously carried as an asset on the company's books, was written off, as it was shown by experiments that they could not be treated economically. The business in the treatment of by-products has increased and a blast furnace has been erected. Lead sheets, pipes, foil and litharge have been produced. The debentures which fell due on the 10th of May, 1899, were

extended to the same date in 1901, when it is anticipated the company will be in position to retire them.

The Ferreira Gold Mine, Ltd., during 1898, returned dividends amounting to £270,000 or 300% on the original capitalization of £90,000; 131,713 tons of ore were crushed against 125,325 tons in the previous year. The value of the gold won was £553,960 11s. 6d. The improvement over the previous year was mainly due to an extraction of 92·1% against 84·2% which was owing to the recovery of the gold from slimes. A large amount of development work was done during the year, and the mine is in a thoroughly healthy condition. The total amount of dividends has been £1,133,500 or 1,465% of the capital of the company.

The Geldenhuis Deep is one of the best examples of deep level mining. There has been a steady increase in the production per ton of ore treated, as well as a decrease in cost. In the first three months of the company's operations it lost over 10s. per ton; in the succeeding year the profits amounted only to a trifle above 6d. per ton, while in the year ending December 31, 1898, the profits were a shade below £1; 735,000 tons of ore are said to be in sight, and George Webber, the general manager, predicts increased profits in future.

The Crown Deep won £200,183 13s. 5d. during the year ending December 31, 1899. The company parted with a portion of its property to the Robinson Central Deep; £150,000 were paid in dividends, or 50% on the capital of the company; £588,647 tons of ore are said to be in sight. A falling off in the grade of ore was experienced in the year, due to the employment of machine drills in the stoping, as native labor was scarce, as well as the necessity under the circumstances of drawing upon the low grade surface dumps for a supply of ore.

The mines in the Transvaal are controlled probably by 10 firms and corporations, not including independent interests. The Consolidated Gold Fields controls properties with a nominal capitalization of £19,850,000; Wernher, Beit & Co. are a close second with £18,474,567 capital in companies which this firm controls; J. B. Robinson, the next greatest owner, dominates incorporations with a nominal capital of £14,367,500; the Barnato Bros. control a capital of £9,675,000; G. & S. H. Farrar's amount to £6,545,300; S. Neumann & Co. to £5,416,500; A. Goerze & Co., Ltd., £4,530,000; G. & L. Albu, £3,700,000; Henderson Transvaal Estates, £3,510,000; Freeman Cohen's Consolidated, £2,550,000, and various groups in which none of the previous combinations are interested show a total capitalization of £12,483,750. The mere nominal capitalization of these companies reaches the great sum of £101,102,617, and as the market value on the average is considerable above par the monetary power wielded by these generals of industry can be appreciated.

Martial law was declared October 15, 1899, and the Government took immediate steps to secure a number of gold mines, promising the shareholders a full accounting of profits after the war. Mines owned wholly in England were not touched. Some 10 mines were worked, which were divided into two classes. (1) Those worked under Government supervision, but as nearly as possible by the company's own staff; here the production has been maintained at practically

the former rate. (2) Those worked under direct Government control with Government Mining Engineers as managers, and an entirely new staff.

Pumping was allowed at some 30 mines, the shares of which were held in numerically greater proportions on the Continent than in England. The Transvaal Mining Engineers claim a reduction of expenses below those under previous management, although they state that the reserves had been practically depleted in the working of the property for four or five months previous to the declaration of war. The elimination of high salaries, curtailment of staff, and reduction in the cost of Kaffir labor might bring about an apparent reduction in cost, but as little or no development has taken place, the work being confined to stoping exposed bodies, this reduction is fallacious. According to the report of the Transvaal Engineers the mechanical condition of the various plants was not so good as would have been expected from the reports of the companies' engineers.

ASIA.—*Celebes.—Dutch East Indies.*—This island shipped in 1899 to England and Germany selected lots of ore running from 4 to 7 oz. gold per ton, valued at $100,000. Considerable development work is under way for 1900. The three principal companies on the island are the Palelah Co., the Totok Co., and the Soemalata Co. The first two companies have gold mills of 10 and 20-stamp capacity respectively and treat ore averaging 1 oz. gold and very little silver. The bullion and high grade concentrates will be shipped abroad for treatment. The Soemalata Co. is equipped with a shaft matting furnace and will treat ore assaying 2% copper and from 0·75 to 1 oz. gold per ton, shipping the auriferous matte to England or Germany.

China.—Gold is said to have been found at several points near Port Arthur both in quartz veins and in deposits along the seashore. The Russian Minister of Agriculture and Crown Lands deputed the chief of the Kamchatka surveying expedition to make an examination. He reported favorably on the deposits. The Mining Bureau in China, which has been placed under charge of Pritchard Morgan, is about to make a new experiment in prospecting. Miners are to be employed for a certain period under salary to prospect assigned sections of the country. At the expiration of their contract, which is not to be less than two years' duration, they are to be allowed to take up any lands or mines on their own account. It is believed that this will stimulate the men's efforts to make discoveries.

India.—The Mysore Gold Mining Co. produced during the year 155,786 oz., which realized £606,947 6s. 4d. There was a balance of profit, after paying royalties amounting to £30,080 16s. 3d. of £404,254 6s. 9d. Dividends free of income tax amounting to £350,000 were declared, or 140% on the nominal capital of the company, against dividends of £375,000 or 150% in the previous year. The mill crushed 92,343 tons at a total cost of £23,771 16s. 10d., which yielded 146,468·8 oz. The cyanide plant treated 57,815 tons of tailings for a yield of 7,290·7 oz. bullion, and 23,743 tons of slimes for 2,032 oz. gold. The consumption of cyanide varied from 1·2 lb. per ton when treating slimes to 1·77 lb. when treating tailings. The tailings averaged about 3 dwt. 16 gr. per ton, from which 2 dwt. 5 gr. were recovered. The consumption of zinc was about

0·15 to 0·21 lb. per ton. There is said to be 196,000 tons of good grade ore in sight.

The Champion Reefs Gold Mining Co. produced 154,802 oz. of bar gold, equivalent to 150,076·6 oz. of standard gold, from which £585,048 4s. 3d. were realized. Dividends to the amount of £275,000 were declared, making a total of dividends paid of £1,090,500 from a total production of gold, valued at £2,396,870 15s. 4d. During the year 93,121 tons of ore were milled, producing 128,197 oz. of bar gold, an average yield of 1 oz. 7 dwt. 12 gr. of gold per ton; 25,865 tons of mill tailings were treated by Wheeler pan process, producing 5,739 oz. of bar gold, an average yield of 4 dwt. 10 gr. of gold per ton; 99,806 tons of tailings were treated by the cyanide process, producing 20,866 oz. of gold 850 fine, an average yield of 4 dwt. of gold per ton, composed of 77,330 tons of mill tailings, producing 16,543 oz. of gold, an extraction of 58·6% and 22,476 tons of slimes from pan process of assay value, making a total of 154,802 oz. of bar gold, being an increase of 19,390 oz. over the previous year. A correspondent of the Madras *Weekly Mail* takes a pessimistic view of the Wynaad district, stating that no well defined reef has yet been proved to exist there, and no payable ore has been crushed.

Japan.—New gold fields were discovered in Hokkaido in the North Island. It is said the auriferous area covered nearly 100 square ri, or 25 square miles in the Yessachi, Soya and Mombetsu districts of Kitami Province. About 7,000 miners are on the spot, and have produced since May, 1899, to the end of the year, gold to the value of about $500,000. It is thought that the Government intends to remove the disabilities which at present prevent foreigners participating in mining enterprises, so that the investment of their capital in Japanese mines may be encouraged.

Malay Peninsula.—The production of gold in Pahang in 1898 shows a falling off, though the Raub mine produced 12,306 oz. against 11,584 oz. in the previous year, and the production of the Silencing mine fell off from 8,256 oz. to 6,568 oz., as did that of the Penjum mine from 6,400 oz. to 3,796 oz. A 40-stamp battery will be erected shortly at the Raub mine.

AUSTRALASIA.—*New South Wales.*—A bill was presented to the Legislative Assembly recently to provide for leases for gold dredging on private or crown property. It allows an applicant having the Warden's sanction, after having made a deposit, to enter upon private property and mark out an area, the size of which will depend upon the number of men that he agrees to employ, though the Minister for Mines may arbitrarily refuse to grant the lease, notwithstanding the applicant has complied with the conditions of the Act. Such provisions have worked badly in the past, several Ministers for Mines having been openly accused of corruption. A second bill introduced opens private property which was previously exempt from intrusion by the prospector for prospecting or mining copper, diamonds, opals, platinum, bismuth, cinnabar, wolfram and chromite. The Legislative Council, who are largely identified with the squatting, or land holding and sheep raising industry, eliminated the right of entering for these minerals from the Mining Act of 1899, and the Amendment of 1896, the principal reason being that copper is so thoroughly distributed

throughout the Colony that scarcely a station property has not copper prospects. A third bill is intended to provide a fund for the benefit of relatives of those killed in mine accidents and for the relief of those permanently disabled therein. The revenue is to be derived from (1) the fees paid by workers in and about mines; (2) contributions from mine holders; (3) from the Government's funds. Each miner or worker about a mine must contribute 10s. annually, the mine owner must contribute a similar amount, and the Government is likewise to pay in the succeeding year after the adoption of the Act, a sum equal to that contributed by the mine owners in the previous year. When the miner is permanently disabled he is to receive 12s. weekly, after having been certificated by the authorities at the place. In case of death his widow, while unmarried, is to receive 8s. per week, and 2s. 6d. for each child until it reaches 14 years. Whether married or not any sister or mother unmarried and dependent upon the deceased for support shall receive 8s. weekly. In case of death £12 shall be paid for funeral expenses.

Gold mining in this Colony showed a remarkable increase in production during the year, although it is difficult to determine from whence it came. In 1898 the product was 292,940 fine oz., while in 1899 it reached 476,699 fine oz., being the highest annual production by far of any year since 1889, and only exceeded twice in the history of the Colony, though during part of the year many of the mines suffered from the drought and were obliged to suspend operations. The latter half of the year rains occurred and the mines were able to recommence crushing, which accounts in part for the increase. One of the best examples of an economically worked mine in this Colony and which is cited because the machinery was of colonial manufacture and the management likewise colonial, is that of the Myalls United Gold Mining Co., near Tomingley, 35 miles from Dubbo. According to W. R. Thomas* the entire cost of the 40-stamp mill, 5-drill air compressor, hoist and cyanide plant was £9,669 12s. 7d. Of this the railway freight and carriage amounted to £1,247 19s. 8d., while the labor in erection amounted to £992 8s. 1d. The cyanide plant with a capacity of 2,200 tons monthly cost, including everything, £2,088 7s. 4d. The total cost of developing the property to insure economical and effective methods of working amounted to £26,414 9s. 9d. The maximum cost has been 15s. per ton of ore treated. Labor costs from 7s. 6d. to 9s. per day, gelignite (dynamite) 1s. 3d. per lb., cyanide about 1s. 2d. per lb., fuel (wood) 7s. per cord, candles 5d. per lb. The costs of the cyanide treatment amounted to 3s. 0·8d. per ton, divided as follows: Labor (filling vats), 1s. 0·3d.; supervision, 3·7d.; boys, 1·0d.; laborers (extra), 1d.; tip timbers, 0·1d.; steam, 1·2d.; lime, 0·5d.; coke, 0·6d.; chemicals, 0·5d.; zinc, 1·7d.; cyanide, 1s. 1·7d.; sundries, 0·5d.; total, 3s. 0·8d.

The Beacon mine at Coramba Creek, which had been promoted by the Exploration Co., Ltd., was shut down finally in December and its property sold at auction. The 20-stamp mill, hoisting machinery, aërial tramway and cyanide plant brought only £1,300. The nominal capital of the company was £200,000, and the total dividends paid amounted to £15,000. A great many gold dredges were constructed during the year following the initiative of C. W. Garland on the

* A paper read before the Institution of Mining and Metallurgy, Feb. 15, 1899.

Macquarrie River. , Results in some cases were profitable though it is doubtful if the profits will ever justify the increase in value of shares.

PRODUCTION OF GOLD IN AUSTRALASIA.

Colonies.	1898.			1899.		
	Crude Oz.	Fine Oz.	Value.	Crude Oz.	Fine Oz.	Value.
New South Wales (a).....	341,722	292,940	$6,055,070	509,418	436,699	$9,026,568
New Zealand.....	280,176	254,416	5,258,779	389,558	356,222	9,363,100
Queensland.....	918,106	647,487	13,383,556	947,626	668,076	12,809,131
South Australia.....	.....	22,377	462,533	32,857	28,750	594,268
Tasmania.....	.....	68,624	1,418,258	76,622	74,706	1,544,173
Victoria.....	837,258	788,429	16,296,390	854,500	804,644	16,631,988
Western Australia.....	1,050,180	939,490	19,418,712	1,648,876	1,462,840	30,236,907
Totals	.....	3,013,763	$62,293,298	4,454,457	3,881,037	$79,206,130

(a) There seems some reason to doubt this increase in New South Wales as it was synchronous with increased ore shipments to Illawarra and Cockle Creek from West Australian mines. The monthly productions have been exceedingly uneven, as have the West Australian ore shipments. It is possible, therefore, that part of this increase, which seems phenomenal in a year of mines inoperative owing to drought and exhaustion, may be due to statistical duplication.—Editor THE MINERAL INDUSTRY.

New Zealand.—The gold production in New Zealand increased from 280,175 oz. in 1898 to 389,558 oz. in 1899, a value of £1,513,173 in 1899, against 1,080,691 in the previous year. The production for 1899 was the greatest for 43 years. While the Waihi Gold Mining Co., Ltd., increased its output as did several other mines, the greater portion of the increase came through the impetus in gold dredging. During the year many public and many private companies were formed for dredging, particularly on the Molineux and Clutha Rivers. While returns have been fair from there, and in instances have been extremely profitable, considering the capital invested, without doubt many investors will suffer through undue inflation in values. Mining in New Zealand owing to its superior climatic conditions, its abundance of water and extensive timber reserve, will doubtless remain in a healthy condition for many years, though with the exception of a few mines English investments have been unremunerative.

Queensland.—During 1899 the Charters Towers regained its place as second best producer in Australasia. The total recovery was 512,042 oz. of gold, valued at £1,216,750, obtained from treating 209,803 tons of quartz by amalgamation, 693,538 tons of tailings by the cyanide process, 5,541 tons by chlorination, 202 tons by smelting and 3,916 oz. were recovered by alluvial workers; £303,262 were disbursed in dividends, while the calls amounted to £75,448. The Brilliant & St. George United was the greatest producer, having treated 34,224 tons for 41,383 oz., and a recovery of 20,936 oz. of bullion from the cyanide works; £95,400 paid in dividends; Kelly's Queen Block paid £34,000; the Brilliant £45,500, and the Daydawn Block £37,500.

The Mount Morgan Gold Mining Co., Ltd., for the year ending May 31, 1899, shows total returns from gold obtained $3,320,715 (counting £1=$5); from miscellaneous sources $5,715, a total of $3,326,530. The expenditures were mine expense and construction, $1,852,480; royalty and dividend tax, $124,025; reserve fund, $4,880; dividends, $1,750,000; total, $3,731,385, showing a deficit of $410,660. The directors' report says: "Comparing the balance with our balance last year, it will be seen that the credit balance of $374,545 has turned into a debit balance of $31,410, showing a loss in floating balances of $410,660. But for this we can show new mundic works, $309,335; electric light and power plant,

$31,450; additions to dams, $5,050; railway line to works, $32,795; increase in stock of firewood and material, $35,000, a total of $413,675. The mine report shows 158,283 tons ore and 105,047 tons waste taken from the open cut, and 13,296 tons ore from the underground workings. The development work shows a total of 2,246 ft. drifts and cross-cuts."

The following ores were treated during the year: Oxidized ore, 40,803 tons, yielding 39,210 oz. or 0·95 oz. per ton; low grade oxidized, 130,776 tons, yielding 52,808 oz., an average of 0·4 oz.; per mundic, 30,069 tons, yielding 72,224 oz., an average of 2·41 oz.; tailings and slimes, 2,854 tons, yielding 1,837 oz., an average of 0·64 oz. per ton. The total being 204,502 tons for 166,079 oz., an average of 0·81 oz. per ton. During the year 41·5 tons of slag, ash lead scrapings and by-products were treated for a smelted return of 1,462 oz. gold. A series of investigations has been undertaken in the manufacture of chlorine, an important subject, as during the year the following quantities of reagents were used: Manganese, 922 tons; salt, 873 tons; sulphuric acid, 2,925 tons, the total cost being slightly over $85,000. The manganese which came from Gladstone averaged 71·24% MnO. The salt came from South Australia, while the sulphuric acid was made on the spot; 768 tons of sulphur from Japan and 100 tons of niter being consumed. Gay-Lussac and Glover towers have been erected. The mundic ore has required 11% less chlorine than in the previous year, while the quantity used on low grade oxidized ore has increased. The Richards furnace is said by Mr. Richards to be working admirably; ore roasted showing less sulphur than the reverberatories and requiring less chlorine, the costs likewise being diminished. The Richards' process consists in using chlorine water as a lixivating agent in open vats. There seems nothing distinctly patentable about the furnace. It is strange that this company has not adopted the use of liquid chlorine, as the consumption on the class of ore treated at Mount Morgan with perfect apparatus should not exceed one pound of liquid chlorine to the ton; the cost would not exceed $0·30 as against $0·42 under present figures, making a total annual saving of $24,000, to say nothing of the trouble of handling 922 tons of manganese, 873 tons of salt, 768 tons of sulphur and 150 tons of niter in addition to the manufacture of 2,925 tons of sulphuric acid. During the six months ending November 30, 1899, 118,658 tons of ore were treated, being an increase over the previous six months owing to the completion of the mundic works. The average of the ore treated was 14·75 dwt., being 1·5 dwt. less than the preceding year. The mundic contained a small amount of copper, and it will be necessary, as this increases in depth, to make some minor additions to the works in order to effect its recovery. According to the report of the metallurgist, the mundic works returned only 9·74 dwt. from ore carrying 1 oz. 5·4 dwt. per ton, being a recovery of 38·3%. This seems very unsatisfactory and looks as if the treatment of arsenical pyrites at this works has not been mastered.

There was a falling off in the production of the Croydon district, as there was also at Gympie, but the total production of the Colony increased from 647,487 fine oz. to 668,076 fine oz.

South Australia.—There was very little activity in South Australia proper, but some promising claims were opened up in the Northern Territory.

Victoria.—The gold production in this Colony increased from 788,429 fine oz. to 804,644 fine oz. in 1899. The increase came mainly from greater activity in working alluvial ground. It is the law in Victoria that when a mine or a portion of a mine is let on tribute, the company or owner becomes responsible to the tributers to the extent of one-half the nominal day's wage should the tributing venture prove unsuccessful. Several ventures during the year proved unfortunate and directors of companies have declined to let tributes unless the men contracted themselves out of the provisions of the New Mines Act of 1897. The Attorney-General has given his opinion that it is legal for the worker to renounce these rights, but the mine owner notwithstanding this, has feared the penalty prescribed for the breach of the section and has refused to take the risk. As a consequence, men have petitioned the company in the case of the Last Chance United Co., of Ballarat to be allowed to work the mine on a division of the gold. The men doing the mining and delivering the ore to the mill; the company crushing the ore and supplying all tools, the tributers providing explosives, the company and tributers to divide the battery gold, while the concentrates are to be the property of the company alone.

The deep leads or ancient river beds capped by disintegrated rock or volcanic overflow have attracted much attention during the past year, extensive capital having been raised both locally and abroad to prosecute work. The Government has granted pecuniary aid in a number of instances and has granted important concessions in relation to annual rentals and labor conditions. The Loddon and Chiltern Valley systems are the most important in point of production, but it is estimated that there are over 400 miles of deep channel at present unworked in Victoria. About 35% of the gold output of the Colony comes from this source.

West Australia.—There was an abnormal increase in the production of West Australia mines in 1899. The total production of the Kalgoorlie district for the year ending December 31, 1899, was 857,503 oz. 10 dwt. (equivalent to 782,415 fine ounces) from 467,048 tons, an average of 1 oz. 16 dwt. 17 gr. per ton, the production for the year practically equalling that for all previous years. The total production of the Colony for the year was 1,462,840 fine oz., as against 939,490 in the previous year. As in 1898 some mines produced the greater portion of the output at Kalgoorlie. The production of the Associated Gold mines was 108,270 crude* oz., the Golden Horseshoe 103,643 crude oz., the Great Boulder Perseverance 43,015 crude oz., the Great Boulder Proprietary 83,727 crude oz., Hannan's Brown Hill 90,934 crude oz., Ivanhoe 104,000 crude oz, Lake View Consols 238,557 crude oz. The value of the product for the year ending December 31st was £3,323,824 (16,173,728), making a total to date of £6,648,322 ($32,350,735). The report of the general manager for the year ending August 31, 1899, gives some details of the new plant. It states that trouble was experienced in working the Brown Straight-Line furnaces owing to the breakage of the wire ropes used for haulage, and the burning out of

* The value of a crude ounce as estimated by the Kalgoorlie Chamber of Mines, is £3 17s. 6d.

the doors at the discharge end of the furnaces. The first difficulty was overcome by substitution of chains for the rope, while coal or producer gas can be so much better regulated that there is no excessive fire. It had been thought by results obtained in an experimental plant that it was necessary only to crush through a 20-mesh screen, roast the ore and extract the gold by ordinary cyanide leaching methods, but in treating on a large scale it was found that finer crushing was required, and that the roasted pulp required a longer time and contact with the cyanide solution than was anticipated. After 12 days in the vat the material tended to cement together, making it difficult to wash and discharge. Crushing through a 30-mesh screen was then adopted, and dust separation effected. The fines were agitated for four hours with cyanide solution and then passed through a Dehne filter press. The extraction in this way was 95% of the gold in these fines. Krupp ball mills were used successfully for dry crushing; a No. 5 mill crushed 50 tons of ore daily. The cost of amalgamation, including bullion expense, was 6s. 10·5d. per ton. The cyanide costs on treating sands were 5s. 11d. per ton, and for treating slime 7s. 6d. The cost of treating sulphide ore was £3 16s. 9d. per ton; 2,620 tons were shipped to the Smelting Co. of Australia, Ltd., of Illawarra, New South Wales, which averaged 38·976 oz. per ton. The cost of shipping smelting bullion was £7 4s. 5·5d. per ton. The cost of amalgamation was divided as follows:

Superintendence and amalgamation, 5·036d.; engine driving and firing, 2·883d.; feeding, 1·498d.; attending concentrators, 13·998d.; handling tailings, 3·728d.; general repairs, 4·677d.; assaying, retorting and melting, 0·854d.; breaking and tramming, 1s. 3·645d.; pumping return water, 9·877d.; mercury, 0·874d.; fuel, 4·4541d.; water, 1s. 5·411d.; general stores, 0·631d.; electric light, 2·727d.; realization charges, 7·096d. The cost of cyanide treatment was divided as follows: Superintendence and attending solutions, 5·858d.; filling and emptying vats, 1s. 10·629d.; general repairs, 1·511d.; assaying, retorting and melting, 2·236d.; zinc, 0·941d.; fuel, 0·394d.; water, 1·004d.; lime, 1·604d.; potassium cyanide, 1s. 7·166d.; general stores and charges, 1·366d.; electric light, 1·179d.; compressed air, 7·841d.; realization charges, 5·246d.; total, 5s. 11·245d.

The cost of slime treatment was divided as follows: Superintendence and attending solutions, 6·614d.; discharging presses, 3s. 0·931d.; compressed air, 10·123d.; general repairs, 4·148d.; assaying, retorting and melting, 4·193d.; zinc, 1·147d.; fuel, 0·521d.; filter cloth, 2·240d.; potassium cyanide, 1s. 1·896d.; general stores and charges, 2·492d.; electric light, 2·815d.; breaking down old slimes, 1·545d.; realization charges, 3·201d.; total, 7s. 3·201d. The cost of stoping for a total of 85,889 tons was 12s. 0·107d., divided as follows: Mining, 6s. 6·947d.; trucking and mullocking,* 2s. 0·653d.; hoisting, 10·713d.; timbering, 10·620d.; general stores and charges, 4·949d.; tool sharpening and repairs, 2·726d.; assaying expenses, 1·910d.; tool, power and sample boys, 3·215d.; superintendence, 6·374d.

During the year under question dividends amounting to £1 10s. per share were paid. The latter part of 1899 much dissatisfaction was expressed at

* Separation of waste.—Editor THE MINERAL INDUSTRY.

the falling off in the production, shares rapidly depreciated and the management was changed. The Associated Gold Mines, Ltd., was one of the greatest disappointments of the year. Not only did the plant erected for treating the sulphide and telluride ore prove incapable of handling the material, but the shipments of high grade ore to the smelting works at Freemantle fell off, to the astonishment of the shareholders. From a yield in October and November of over 1 oz. per ton to as high as 2 oz. per ton the milling ore fell to 11 dwt., and the shipping ore showed a proportionate decline. In December the sulphide ore treated at the company's plant fell to 10 dwt. 14 gr. per ton, with a consequence that shares declined sharply owing to lack of public confidence in the management of the property. Dividends amounting to £123,750 were paid during the year. Hannan's Brown Hill mine finally arrived at successful basis. During the year ending September 30, 1899, 42,025 tons were treated for a yield of 87,762 oz. of gold, and £176,250 were paid in dividends, leaving a balance of £54,527 16s. 9d. From this reserve a dividend of £52,500 was declared payable January 10, 1900. The company has acquired considerable property during the year. Alluvial troubles were still existent in 1899 in a more acute form than ever. Though both the Wardens and the Supreme Court had decided against the rights of individual miners to enter upon lode leases they had still persisted in acting upon the rights which they had asserted.

The most important development in Kalgoorlie during the year has been that at the Great Boulder Proprietary. The continuance of sulphide and telluride bodies has been proven down to the 1,100 ft. level. To the east and west of the main shaft bodies of high grade ore have been encountered from the 800 ft. level downward either by cross cuts or by bore holes.

Austria.—According to A. Irmler* the gold deposits of Brezna had been worked for more than a century. The veins occur in granite and are traced over 1,000 m. in length. The gold is associated with antimonial minerals.

Bokhara.—The head waters of the Pandj River dividing Bokhara from Afghanistan were visited in 1898 by Albrecht von Krafft.† The gold bearing conglomerates of the Tertiary Age, whose constituent rocks were the débris from diabase, tuffs, red felsite, porphyries, red and gray granites, diorites, porphyrites and gneisses, were examined by the expedition and the cemented material, silicious and partly calcarious was found to be auriferous, the gold occurring in thick flakes. Auriferous quartz veins on authority of the Russian engineer Pokorski, run through the crystalline rocks in the immediate vicinity of the conglomerates, which veins are probably the source of the gold in Eastern Bokhara. Ancient workings are to be found as huge open casts and enormous dumps. The natives of the present time have abandoned vein mining, to work the auriferous sands along the river banks. Eight thousand natives work annually about 6,450 tons of sand from bedrock, yielding 5,792 oz. avoirdupois of gold, which is nearly all exported to Afghanistan. The sands have been proven to be poor down to about 26 ft. below the surface, so that

* A paper read before the Geological Institute of Vienna.
† *Zeitschrift für Practische Geologie*, pp. 37-43.

work is all done below this level, long tunnels being run up stream, in order to effect drainage. The beds extend back from the river a considerable distance, as it is the opinion of the writer that the industry would pay handsomely if organized on a large scale by Europeans, the attendant conditions being favorable. Mr. W. Rickmers reports that the gravel contains about 4% of large boulders of from one cu. m. upwards in size. The gold is almost invariably scaly, and nuggets of any size are nowhere found. The largest pieces discovered in Safet-Daria weighed 12 g. The richness of the sand increases with the depth, reaching its maximum at the bed-rock, and the sands found in the gravel terraces of the river banks are richer and contain coarser gold than the deposits of the river bed. The bed-rock is about 15 m. below the surface, although it has been seldom reached. While the gravel becomes richer in depth the horizontal distribution seems to be regular and even. Samples give an average result by washing over felt of 0·8 g., or about 48c. per metric ton. Thirty cents a ton, it is estimated, will cover the expense of excavating, washing and dumping gravel.

Russia.—The gold production of Russia during 1899 was reported at 1,159,214 oz. against 1,196,634 oz. in 1898. It is expected that the completion of the Trans-Siberian Railway will render accessible a large area of gold bearing land. The Government, with a view to opening up some of the gold bearing lands on the northwestern coast of the Okotsh placed them on auction in the early part of the present year; while the effort to bring in capital was doubtless well meant the restrictions were so binding that it is doubtful if capital can be secured. To stimulate the mining industry, the Government suspended duties on machinery, required in gold mining, for a period of 10 years, extending to January 1, 1909. This includes concentrators of all systems for working gold bearing sands, contractors for the treatment of slimes, centrifugal concentrators for treatment of dry crushed ores, amalgamating apparatus of every description, including amalgamated copper plates and other appliances, and all machinery necessary for the wet extraction of gold from ores and tailings, including vats for the cyanide or chlorination processes; the necessary connections and the precipitating materials, including dynamos, if an electric method is used. Reagents are also admitted free; credentials must be furnished the customs.

D. A. Louis* describes the Miask gold deposits of the Ural Mountains. Here both alluvial and quartz are worked, although both in a somewhat primitive manner. The method of working the alluvial deposits has been frequently described, but at Tcheliabinsk within 15 miles of the Trans-Siberian Railway there are no less than 18 well-recognized lodes or groups of lodes, most of which might be called strong fissure veins. The country rock is granite but berecite, porphyry and diabase are associated with it. The fissures are filled with granitic material accompanied by white opaque quartz, iron stained near the surface, but containing undecomposed sulphides at a depth of from 100 to 150 ft. The ore is low grade, running from 1·66 dwt. to 7 dwt. per ton, and is only from 7 to 27 in. wide, though in exceptional cases the veins are

* *Transactions of the Institution of Mining and Metallurgy*, 1899.

from 4·5 to 6·5 ft. The ore after breaking is hoisted with a horse-whim and crushed under edge runners with steel faced rolls 4 ft. in diameter and 10-in. face driven by an engine. The pulp passes through a 0·125-in. screen to amalgamated plates set at an incline of 1 in 7, in sets of four plates, with a drop of 2 in. between each plate. The output of the mill was 25 tons a day, and the yield of gold averaged about $1·60 per ton; 80c. or 33% being lost in the tailings.

Progress in Gold Milling in 1899.

Cost of Milling in Nova Scotia.—Some interesting figures are given by W. L. Libbey* on the course of mining and milling gold ores at Queenstown, Nova Scotia. When it is considered that the vein does not average 14 in. of crushing material the results seem remarkable. The detailed figures are as follows: labor, including blacksmith and top plat man, $1·97 per ton, sent to the mill; timbers, $0·07; shovels, $0·0006; picks, $0·0036; coal, $0·013; hauling rope (depreciation estimated), $0·001; candles, $0·065; charcoal, axles, steel, iron, including rails, lumber and miscellaneous, $0·074; explosives, $0·184, a total of about $2·50 per ton. This is for a period of six months; for the three months following it was somewhat less, being $2·44 per ton.

A New Method of Feeding Water to Mortars.—This method was described by Bernard MacDonald.† Instead of feeding the water through a hose passing through the mortar housing, the mortar is tapped at six points, there being a delivery pipe between each die and between the end dies, and the end of the mortar, at about half the height of the die when new. The disadvantages of the old system claimed by the inventor are that more water is consumed than is actually required, being discharged before it has mixed thoroughly with the pulp, and, furthermore, that the fine slimes are settled at the bottom, reducing the discharge capacity of the mortar, and bringing about the further pulverization of slimes already sufficiently, if not too fine. On the other hand he claims that these disadvantages do not exist with the improved feed, but the pulp is discharged as soon as fine enough to pass the screen, increasing capacity while decreasing sliming. A third advantage claimed is that the gold under this system settles readily to the bottom between the dies, or between the dies and the sides of the mortar, thus protecting the metallic particles from undue abrasion. Each feed pipe is 0·75 in. in diameter, and is fitted with a globe valve. The main supply pipe to which the branch pipes are tapped is 3 in. in diameter. At the Dufferin mine, Nova Scotia, where the mortar was used the water was delivered under 30 ft. head.

It is claimed that a similar device has been used before, but discarded as would seem quite probable. Most mill men, including the writer, will hesitate to adopt MacDonald's hypotheses that the greater amount of water fed in the ordinary way will increase the slimes as frequently by decreasing the flow of water they have increased the fineness of pulp while necessarily reducing the crushing capacity. Mortar saving is largely due to coarse gold being present, which will be

* *Journal of the Mining Society of Nova Scotia*, 1898.
† *Transactions of Canadian Mining Institute*, Vol. III., p. 54.

saved even to a high percentage though no mercury is fed. In the case of gold being fine, it is of course better, as in Gilpin County practice, to keep the pulp in the mortar as long as possible, thus ceding capacity to extraction. A difficulty which has suggested itself in this new method is that the pipes will have a tendency to close, the impact of the stamp throwing pulp up the pipe, unless the stream is of sufficient volume and head to nullify the advantages claimed by the writer.

Amalgamation at the Espiritu Santo Mine.—Ernest R. Woakes, treating the ore of the Espiritu Santo mine, in the Republic of Colombia, which contained 3 to 5% of pyrite, blende and galena, found that the best results here also were obtained by keeping the plate amalgam hard, and feeding a minimum amount to the mortars. Outside lip plates and a splash plate are employed. The two lip plates are removable to aid in cleaning up; below these is a mercury wall, then plates 48 in. wide by 9 ft. long. When the lip plates are cleaned up monthly they are removed and sandwiched in a box with hot sand, which has been heated to a dull red heat, and left for half an hour when the otherwise hard amalgam comes off readily. The upper 4 ft. of the apron plate is then treated with hot sand and scraped after ten minutes. Objections that may be raised to the use of the hot sand is the possibility of salivating the amalgamators. Steaming the plates is generally considered sufficient. The distribution of amalgam from a month's run was as follows: daily clean ups (from dressing plates), 24%; blanket washing, 0·9%; lip plates, 32·28%; apron plates, 18·28%; battery sands and upper mercury traps, 23·36%; lower mercury traps, 1·13%. The amalgam contained 36% of gold, and the resultant bullion ran 932·71 fine.

Trolley for Transport of Amalgam, etc.—In some recently constructed gold mills there is a suspended track in front of the line of batteries, which leads into the clean-up room. On this track a cage is suspended from a trolley, to carry the product of the clean-ups directly from the mortars to the clean-up room. This permits all handling of valuable products being accomplished in one room under the immediate supervision of the mill-man or the superintendent. It dispenses with manual labor in conveying amalgam to the clean-up room, and in removing shoes and dies. It is also useful during repairs.

The Ayton Method of Obviating Settling Tanks in Pan Milling.—At the Bote mine, in Zacatecas, Mexico, the Ayton intermittent thick pulp extractor is employed to deliver thickened pulp to the pans, replacing the large settling tanks formerly necessary. The apparatus is applied to a pointed settling box, 10×10 ft. square at the top, and 8 ft. 6 in. deep with a uniform slope of 60° on each side of the box. A valve is secured to the point of the settling box, the movement of the valve being controlled by a lever held in position by means of a spring and actuated by a cam movement. The extent of the value opening, and the frequency of discharge are regulated by adjusting devices. The pulp is allowed to settle or collect at the point of the box for a short period, when the valve opens, and a portion of the thick settled pulp is allowed to escape, the valve being closed so quickly that the supernatant watery pulp does not reach the level of the discharge opening. This method of intermittent discharge combines the advantages of a comparatively large opening with the limited flow of water, so that the thickness

of the discharged pulp can be maintained, and the danger of obstruction of the pipes, together with the obvious disadvantage of a small opening under the full head of the settling box, attendant upon the use of a siphon discharge, are obviated.

The Bote plant crushes from 7 to 75 tons of ore daily in Chile mills, which require an addition of 85 to 90% by weight of water to effect proper discharge. The pulp is run into eight settling boxes which return to the Chile mills from 78 to 87% of practically clear water, while the pulp extractors deliver a continuous supply of pulp to the pans containing less than 50% water. The general average of the pulp for a month showed 40·5% of water.

Concentration in Absence of Hydraulic Water.—Edw. H. Johnson* describes a rough concentrating appliance, somewhat on the principle of the old labyrinth sluice, which was used in order to effect economy of water, there being insufficient for hydraulic classification as ordinarily practiced. Briefly described the apparatus consists of two similar boxes, in which the pulp from an 80-stamp mill is distributed over a width of 7 ft., causing a reduced flow. Near the entrance of the pulp two weirs are formed by the periodic addition of pieces of wood 3×3 in. square.

These serve to break the force of the flow of pulp, and are always kept a little higher than the bed of accumulating concentrates. The end of the box is similarly built up by 0·5 in. laths as the concentrates accumulate. The boxes are used alternately, the one being discharged while the other is filling, each box holding a day's product. Bottom discharge doors of simple design are fitted to the boxes. The average product for five months equaled 10·5% of the gross quantity of tailings, producing a concentrate of 18·5 dwt. per ton value from tailings of the average value of 4·7 dwt. The tailings leaving the box assayed 3·25 dwt. per ton, thus a saving of 30·8% of the contents in 10·5% of the sands was made.

Hydraulic Method of Squeezing Amalgam.—The Alaska-Treadwell Mining Co. has adopted a hydraulic squeezer for eliminating the surplus mercury from amalgam. The amalgam is placed in a canvas sack, which is inserted in a perforated cylinder, and a ram forced against the bag containing the amalgam. By reversing a four-way valve the ram is backed out of the cylinder, and the amalgam can then be removed and retorted in the ordinary way. The squeeze may be operated either by water or by compressed air.

Manufacture of Pure Gold.—Cabell Whitehead† recommends the following method of making pure gold and silver for assay checks. To make gold proof 10 oz. or thereabouts of fine gold, rolled into a ribbon, is placed in the flask, with HCl and HNO_3 added until the gold is dissolved. The solution is then evaporated until the HNO_3 is entirely removed; this sometimes requires additional HCl. When the HNO_3 is entirely removed, a small quantity of HBr is added, and the solution diluted until it does not contain more than one ounce of gold per gallon. The small amount of silver which the solution contains, due to the fact that argentic chloride is soluble in a concentrated solution of $AuCl_3$ is precipitated by the dilution only partially, but the addition of HBr causes the entire pre-

* *The Journal of the Chemical and Metallurgical Society of South Africa*, 1898.
† *Engineering and Mining Journal*, Dec. 30, 1899.

cipitation, as the bromide is much less soluble than the chloride. After agitation the solution is left to rest until it is clear, and the supernatant liquor is siphoned off without disturbing the precipitate. The filter upon which it is decanted is not washed, and the gold in the filtrate is finally precipitated by SO_2. This allows a separation of the copper from the gold, and unlike when $FeSO_4$ or metallic Al is used, no impurities are introduced. The gold now contains no platinum or copper, and a trace only of silver. It is dissolved as before, HBr acid added again, and the final silver contained allowed to settle as bromide. This operation takes considerable time. The next precipitation of the gold is made with oxalic acid, as this reagent does not precipitate platinum, contains no metal, and is itself volatile at a temperature below the melting point of gold, but, however, it cannot be used, in the presence of copper, as the copper oxalate is only partially soluble. This precipitation occupies 6 to 7 hours. The gold after settling is washed, redissolved, and again allowed to settle. It is then precipitated for the third time. It is now washed with water following with a HNO_3 wash, a second water wash, with HCl acid and finally with water. The precipitated gold after drying is fluxed with one-fifth of its weight of a mixture of three parts of fused borax and one part of pure niter, melted in a porcelain crucible and poured into a chalk lined mould. Adhering impurities are eliminated by careful brushing, and by final boiling in HCl. This process, while tedious, has been found by Whitehead to be the only one absolutely reliable, and to be greatly superior to the method published by Pack,* in which pure aluminum was used as a precipitant, as the aluminum not only contains a certain amount of impurities, but fails to eliminate the silver perfectly.

Quantitative Determination of Gold.—Vanino and Seeman's method is based on the fact that in a solution of gold chloride the gold is reduced to a metallic state by means of hydrogen peroxide and potash or soda lye. The reaction takes place almost instantaneously, and is finished in a few minutes, even in the cold, while the precipitation with iron sulphate, or with oxalic acid takes from 2 to 12 hours. The gold is precipitated from an alkaline solution in the cold, in a very short time, as a black powder which balls together, and assumes a reddish brown color when heated. The reaction takes place according to the following formula:

$$2AuCl_3+3H_2O_2+6KOH=2Au+6O+6KCl+6H_2O.$$

When working with diluted solutions it is advisable to apply heat soon after the precipitation has taken place, then to add some HCl acid to obtain the precipitate in form convenient for filtration. Care has to be taken to add the acid only after heating, for otherwise, in presence of hydrogen peroxide some of the gold may be redissolved by the chlorine evolved from the liquid. This method can also be used for the separation of gold from platinum and from iridium.†

Determination of Gold by Means of Sodium Thiosulphate.—F. A. Gooch and Frederick H. Morley have investigated the possibility of determining small amounts of gold in solution by titration with sodium thiosulphate of the iodine set free by the action of an excess of potassium iodine upon auric chloride.‡

* *Engineering and Mining Journal*, July 9, 1898. † *Zeitschrift für Angewandte Chemie*, Aug. 22, 1899.
‡ *American Journal of Science*, October, 1899, p. 261.

In the investigations it was found that the reduction of the auric salt, and the consequent liberation of the iodine was conditioned by the volume of the solution, the mass of the iodine present and the time of action. With the precautions adopted results were regular.

Assaying Gold Ores.—William Bettel,* in treating on the loss of gold in assaying, states that this may occur through too rapid melting, but if the temperature is too low, some of the lead particles may remain against the side of the crucible, where they become scorified, leaving small particles of gold behind. By too rapid melting spurting may occur, not only causing a loss in that assay, but which may unduly enrich the assays in adjoining crucibles. If too small a crucible is used, and the temperature is too high, boiling over may occur where insufficient attention has been given. A loss may occur through dusting where the charge is added in a paper bag into a hot crucible if sodium bicarbonate is used. If this reagent is employed cold crucibles should be used.

Three conditions must be complied with in the fire assay of solutions or material containing finely divided gold. (1) Intimate mixture of the sample with litharge reducing material. (2) Sintering of charge before melting, *i.e.*, slow fusion. (3) High finishing heat.

The best way to assay solutions is to take two liters of the solution, slightly acidify with oxalic acid, and filter, evaporate the filtrate to a low bulk in a porcelain dish and transfer to a leaden dish, adding litharge in excess, and a reducing agent, or about 10 c.c. of lead acetate solution. When evaporated, add borax, sodium carbonate, litharge, and the ashes from incinerating the filter. Melt in a small pot in the usual way. If the lead button is too large for the cupel, scorify. It is advisable to add the requisite amount of silver for parting, as a second cupellation is undesirable.

He points out that when low grade residues are assayed a single 3 assay ton charge is insufficient. With a balance turning to 0·05 mg. an error of this quantity equals 8 grains per ton. If the original sample has assayed 4 dwt. to the ton, this error would account for a deficiency of 8·33% between the theoretical and actual extraction, but if the original sample had been returned 3 dwt. 16 grains instead of 4 dwt., the returned extraction would become 6·8% higher than the actual. If 12 assay tons are used the errors are lessened by 75%. It therefore follows to get a fairly reliable result, opportunities should be afforded the assayer for taking large quantities of the material which means increased facilities for work, and the employment of white assistants. Under present conditions the increased speed of working means decreased accuracy. If a matte or speiss forms on the lead button a loss is bound to occur. The best way with an ore of this character is to roast the ore and reroast it with a small amount of finely ground coal. A sufficient amount of charcoal is added to the subsequent charge to bring down a button weighing about 40 g. This may then be scorified. The porosity of cupels should be tested by cupelling known weights of lead and pure silver, the temperature being so regulated that feathery litharge is produced. A record of the absorption number from each box or parcel of cupels should be kept.

* *Journal of the Chemical and Metallurgical Society of South Africa*, November, 1898.

PROGRESS IN THE CYANIDE PROCESS IN 1899.

BY LOUIS JANIN, JR.

WHILE there has been nothing strikingly novel in the cyanide treatment of gold ores during the past year, investigations in various parts of the world, notably those of members of the Chemical and Metallurgical Society of South Africa, have added greatly to our technical knowledge.

In New Zealand, dry crushing, which was prevalent some years ago, has generally given place to wet crushing, as the latter has been found more economical. The ore here is hard quartz, and in the dry-crushing stamp mills slightly above a ton per stamp was crushed per day. A crude method of drying was employed, which was found expensive, so that economy has been caused by the elimination of this step. In the United States, dry crushing and subsequent cyaniding has been adopted frequently, but it would seem that while this method has its special advantages on ores well suited to it, and where certain local conditions obtain, it is not a process of general application. Before a new plant is erected ores should be carefully tested by competent men, to determine the exact type of the plant to be erected.

There seems to be some future for the treatment of silver ores by cyanide solutions, but probably its application will not be general until the cost of manufacturing cyanide is reduced much below that at the present time. Not only are the silver ores either raw or roasted, apt to contain compounds prejudicial to economical cyanide treatment, or as they are commonly termed "cyanicides," but the consumption of cyanide in absolute stoichiometrical proportions, which would not be regenerated by present methods of precipitation, would be so great on ores of a high silver tenor as to render this process uneconomical as compared with other methods for treating silver ore. In the treatment of silver ores, and no doubt in cases where the gold of the gold ores is more or less alloyed with silver, a hot solution will be found advantageous, and in the precipitation of silver bearing solutions, the Siemens-Halske method would seem to have a distinct advantage, though comparing unfavorably in economy with the zinc-lead couple for gold precipitation from dilute solutions.

Slime treatment has become almost universal in South Africa, and the methods now employed seem to give indirect as well as direct advantages. Large volumes of extremely dilute solution are employed and agitation and aëration are necessary. Centrifugal pumps are frequently used for this purpose. These are allowed, by means of a valve or other contrivance, to admit a certain amount of air thus supplying the oxygen necessary for the solution of the gold. The Siemens-Halske method to a certain measure has been superseded by the Betty-Carter process, which is nothing more than a zinc-lead couple formed by immersing zinc in lead acetate solution, coupled with strengthening of the solution in cyanide before entering the zinc boxes.

MacArthur and Ellis certainly seem to have a prior claim for the use of the lead coated zinc shavings, and Wm. Bettel as well as other metallurgists on the Rand made extensive experiments in the use of this couple prior to Carter's results. Increasing the strength of the solution before entering the zinc boxes

cannot be claimed as a novelty, as it has been common enough among cyanide workers for years past to make an addition of cyanide to weak solutions before these enter the zinc boxes.

Experiments at Luipaards Vlei Estate.—Franklin White* states that the Rand conglomerate as is well known consists, more or less of fragments or pebbles of quartz cemented together with fine sediment. The quartz itself contains little or no gold, while the gold is generally in the cement or in the iron pyrites which accompany the latter to the extent of 2 to 10%.

The following table shows the results of screening and assaying the material crushed through a sieve of 300 apertures to the square inch. It will be noticed that the finest residues assayed less than the relatively coarse though the original assay was higher.

CHARGE AND RESIDUE OF PYRITIC ORE THROUGH 300 MESH.

Mesh. In.	Charge Assay at 6·5 Dwt.				Residue after 14 days, Assay 1·5 Dwt.				
	Per Cent. Weight.	Assay. Dwt.	Dwt. 100 Tons.	Per Cent. of Contents.	Per Cent. Weight.	Assay. Dwt.	Dwt. in 100 Tons.	Per Cent. of Contents.	Per Cent. of Extract'n
+⅛	2·4	4·0	9·60	1·3	2·9	1·25	3·625	2·4	62·26
+⅛	20·9	3·5	73·15	10·9	23·3	1·75	40·775	28·2	44·30
+⅛	37·2	6·5	241·80	35·7	34·2	1·70	58·140	40·3	75·96
+⅛	12·8	8·5	108·80	16·0	12·4	1·26	14·880	10·3	86·33
+⅛	26·7	9·2	245·64	36·1	27·2	1·00	27·200	18·8	92·33
		6·789	678·99				144·62		77·00

The cost of crushing pyritic ore, including the cost of rock breaking, was as follows: White wages per ton, 10·995d.; native wages, 3·924d.; stores, 8·863d.; lighting, 0·485d.; steam power, 7·856d.; electric power, 0·408d.; maintenance, etc., 7·322d.; rock breakers, 9·747d.; total, 4s. 6d. The cost of cyaniding was 3s. 5·86d. divided as follows: White wages per ton, 11·18d.; native wages, 7·20d.; maintenance, 3·45d.; assay charges, 3·60d.; cyanide (0·40 lb.) 7·70d.; zinc (0·31 lb.) 1·08d.; lime (2·55 lb.) 1·15d.; light and power, 2·30d.; stables, 0·85d.; coal, coke, and sundries, 1·00d.

The preceding figures show the cost of delivering a marketable gold product. The costs in the direct cyanide treatment were 2s. 3·3d. The method used in cyaniding the crushed ore was a preliminary treatment with a 0·25% cyanide solution followed by two others, the last containing 0·15%. Then the ore was transferred to a lower line of vats. The time occupied in this first operation was 66 to 70 hours. The solutions were not allowed to stand but were drained off when the tank was once filled. The extraction during this period was 67% of the original assay value.

The ore was then transferred to the second row of tanks, when a solution not exceeding 0·2% cyanide, and in quantity of 20 to 25 tons was added to the charge which weighed from 165 to 170 tons. It was then drained off and allowed to remain damp for about 96 hours, when weaker solutions (0·15 to 0·10%) were used, followed by washes, making up a total of 75 tons of solution per tank, for this second treatment. About 20 to 30 tons of water were used for washes. The time occupied was 275 hours. The moisture in the discharged

* *Transactions of the Institution of Mining and Metallurgy.*

residues averaged 12·3%, while 4 tons of water were lost for each tank by evaporation. The solutions running from the first row of tanks carried from 13 to 32 dwt. of gold per ton.

As during this treatment of 22,500 tons of ore the solutions had been used over and over again, it was considered advisable to learn if their efficiency had been weakened. Andrew F. Crosse determined oxygen in several solutions. The water used in the first place contained 1·33 mg. oxygen per liter; the strong cyanide solution contained 3·25 mg. per liter, but it left the ore containing no oxygen whatever. It increased to 2·8 mg. per liter at the end of filtering; the weak solution mechanically aërated contained 6·4 mg. per liter, being saturated and in running off contained 6 mg. per liter. The solutions showed that while the first solution that passed off after 1 lb. of caustic lime had been added to the charge, showed an alkaline reaction, it was necessary to add a greater amount than this to insure complete neutralization during the process. Lime, moreover, was found to aid precipitation, although if in great excess a white precipitate occurred in the zinc boxes which was detrimental. The theoretical extraction on the 25,133 tons treated was 83·13%, while the actual extraction was slightly below this.

After coarse crushing the free milling ore from 68·9 to 73% was extracted, and Mr. White believes that if the ore was crushed slightly finer the results would have been brought up to 85%.

Method of Treatment at Mercur, Utah.—The Golden Gate mill at Mercur is one of the most complete and automatic plants in this country for dry crushing, roasting and cyanide extraction. The ore is delivered directly from the skips to grizzlies, the coarse ore is crushed through Gates' No. 6 crushers and goes with the fine material to the proper bin. There are three separate bins, for oxidized, mixed and base ores. The mixed ore is a talcose material which is heated to red heat to drive off water of hydration and to render it readily leachable.

The base ore contains from 2 to 5% sulphur and 0·95 to 2·5% arsenic. From the storage bins, the ore passes by means of belt conveyors over straight-line line dryers, which are used, however, only when the ore contains too much moisture to crush readily by rolls. The conveyor delivers the ore to Berthelot screens, the oversize passing through coarse crushing rolls, thence over a second vertical screen to the finishing rolls. The screens for oxidized material have 0·375 in. mesh, 0·25 in. for mixed, and 0·125 in. for base.

When properly sized the oxidized and mixed products are dumped into bins direct from the Berthelots and finishing rolls. The oversize base material from the screens is passed through a Gates finishing roll, carried to the top of a second storage bin, where it is screened through a trommel with 0·125 in. mesh. The oversize being returned to the fine roll. Six elevators and six Berthelot screens are employed. The ore in the second line of storage bins is kept separate.

From these bins the mixed ore is passed by a belt conveyor to the Holthoff-Wethey calciners, of which there are two, each 100 ft. long and 12 ft. wide. Each treats 145 tons in 24 hours with a consumption of 8 tons of coal. From the calciners the mixed product is elevated and dumped into a general storage and mixing bin above the leaching tanks, together with the roasted base ore.

and raw or free leaching or oxidized ore. From its storage bin the base ore is elevated 50 ft., and distributed to Jackling's modification of the straight-line furnaces. Of these furnaces there are four, roasting 70 tons each, down to 0·1 to 0·15% arsenic, with a consumption of about 7 tons of coal. The calcines are elevated after their passage through the furnace, which takes in the neighborhood of 8 hours, to the general charging bin together with the oxidized and mixed ore. In charging the leaching tank it is found desirable first to spread about 50 tons of the oxidized material evenly on the bottom, making a depth in the tanks of about 10 in. This aids in subsequent filtration. On top of this the general product of the three different classes of ores is charged after having been drawn into side dumping ore cars and trammed by men to the ore tanks. At the present time there are 26 steel leaching tanks, 50×25×5ft., and containing 260 tons at a charge. Three men are able to charge this amount of ore in 8 hours. After the tank is charged, a strong solution containing 0·4% of potassium cyanide is admitted to 10 two-inch openings in the bottom until the mass is thoroughly saturated. This consumes about 8 hours and on completion the solution is allowed to stand for 16 hours further. Then more of the strong solution is fed on top, and the liquor drawn off through the same openings by which it was previously admitted. The first solution is admitted under 14 ft. head, while the subsequent withdrawal of it is accomplished without vacuum or suction. After feeding the strong solution on top for 24 hours, it is followed by a weaker solution containing 0·3% potassium cyanide, at the expiration of which treatment the weak wash of clear water is used. Washing continues from two to three days. Operations are so calculated that the rate of percolation is about 1 in. per hour and the total amount of solution is 2 tons to the ton of ore.

The tank is discharged through the bottom into cars running beneath, after having been sampled two days before and during the final discharge. The solutions flow to storage tanks about 20 ft. in diameter and 12 ft. deep. From these it is pumped to the precipitation tanks of which there are three, each 14 ft. in diameter and 8 ft. deep. Zinc dust is used for precipitation. When the tank is first filled 30 lb. of zinc are put in, 20 lb. are added on the second charge, 10 lb. on the third and thereafter until the clean-up at the end of the month 15 lb. of zinc fume is used for each full charge of liquor.

After the first charge, no zinc is added until the tank is about half full, when one-quarter of the total used by the full tank is put in, the remainder being added as the tank is filled. During the time that the solution is running into the tank air under 20 lb. per sq. in. pressure is forced through a 0·5 in. iron pipe. Each full charge of solution weighs 30 tons. After settlement for one hour the suspended particles are drawn off to Stilwell-Bierce and Smith-Vaile filter presses from which a solution rarely exceeding 20c. per ton in gold contents flows to the sump tank.

The amount of zinc consumed is about 0·4 lb. per ton of ore, which is considered to average in the neighborhood of $7. To maintain the alkalinity of the solution 1 lb. of potassium cyanide is added to each ton before strengthening. It is stated that the increased alkalinity of the solution aids precipitation and

decreases the amount of zinc dust used. The product of the filter presses is treated by sulphuric and nitric acids in the dissolving tank. The product contains 60% gold. This is again filter pressed, dried in a muffle furnace, roughly crushed and melted in a graphite crucible with a flux of one part potassium carbonate, two parts sodium carbonate and two parts borax glass. The resultant bullion is 950 fine; 145 men are employed in the mill so that 4·2 tons of ore are handled for each man employed. It is anticipated that the labor will be further reduced. The Jackling furnace, to which much of the success is due, was described in the *Engineering and Mining Journal,* December 23, 1899. It possesses considerable novelty and would seem to be economical in its working. At present the plant is run by electric power, motors being utilized at various distributing points. The transmission of power system from Provo has proven so unreliable, that a steam generating plant was being put in commission at the end of the year. Electric motors will still be used, the dynamos being driven by two Lane & Bodley tandem compound non-condensing Corliss engines of 400 h. p.

Dry crushing cyaniding has been practiced successfully at the Lisbon-Berlyn mine in the Transvaal; 28,555 tons were crushed at a cost of 1s. 3·5d. and 27,457 tons were cyanided at a cost of 3s. 6d. per ton. The extraction averaged 68·3%. The mining costs were 12s. 6·4d. per ton and tramming to crushers 6·6d., making the total cost 17s. 10·5d. The extraction amounted to 6·32 dwt. per ton.

Cyaniding before or after Amalgamation?—Some writers pronounce in favor of adopting the usual course of cyaniding after amalgamation, instead of cyaniding and then amalgamating as practiced at the Clark mill, Rochester, Mont., where the success is due to fine after crushing, and the resultant saving of galena by the concentrators. One of the first plants built for dry crushing, cyaniding and after amalgamation, was that of the Australia mine of the Associated Gold Mines of Western Australia, on the advice of the writer. The plant consisted of Krupp ball mills, cyanide vats, Huntington mills and plates for amalgamation. This arrangement was to meet the special requirements of the oxidized ore. It was rich and supposed, as specimens indicated, to contain large quantities of coarse gold. The extraction by the cyanide solution was so high, however, frequently running over 95%, that the Huntington mills were not used eventually. The 5-ft. Krupp mill driven at 23 r. p. m. gave the best results. Some trouble was experienced with dust, through imperfect housing and exhaust.

To avoid settling of the pulp when the cyanide solution is turned on to the tank, slow saturation by upward soakage was introduced at this Australian mine. A small pipe was used and 16 hours was taken up in saturation. The ore contained some lime, and when moistened before charging, cemented in the vats, as drying at a high heat drove off the water of combination of the gypsum, leaving the anhydrous sulphate or plaster of Paris. This mill was the pioneer successful plant on Kalgoorlie ores, but the subsequent occurrence of telluride and sulphide which was unanticipated at that time, rendered other treatment necessary.

Notwithstanding the favorable results that have been obtained at the Golden Gate mine, Utah, as well as at several plants, the tendency in New Zealand

has been to abandon dry crushing for wet. This is due partially to the hardness of the ore, about 1·2 tons only being crushed per stamp per 24 hours. The ores here on account of the high percentage of silver they contain require somewhat longer contact with solution than is generally necessary on fine crushed ore, so that increased crushing capacity effects a considerable economy.

New Zealand Practice.—According to John McConnell,* wet crushing with cyanide solution and subsequent leaching has succeeded dry crushing, and direct cyanide treatment in many places in New Zealand. In wet crushing as little solution as possible is used, averaging from 1·5 to 2 tons to the ton of ore. The strength is somewhat high as the time is necessarily shortened. The pulp is first passed over amalgamated plates and then discharged evenly in the leaching vats by means of automatic distributors. The supernatant solution which accumulates is drawn off, the slime which it contains allowed to settle and the clear solution passed through the zinc precipitation boxes. After precipitation the solution is pumped back to the storage vats, made up to the requisite strength, and sent back to the batteries again. When sufficient ore has been charged into the vat it is allowed to settle, the settlement being facilitated by the addition of lime. This takes from an hour to an hour and a half. The clear solution is then run off from the top to within a couple of inches from the top of the ore, and a vacuum applied to draw the remaining solution through the ore. The washings are applied and drawn off in the usual way. It is advantageous to add a little lime to the mortar while the ore is being crushed; this helps the settling of the ore and slimes in the vat, and allows the solution overflowing into the settling vats to pass off comparatively clear. The quantity of cyanide solution used in wet crushing is much greater than when the solution is applied to dry crushed ore, but there is little difference in the actual consumption per ton of ore. Agitation seems necessary to obtain a high extraction in a short time. With agitation, concentrates from the Woodstock mine after being crushed fine enough to pass a 90-mesh screen showed an extraction of 90% of the gold with a relatively strong solution of cyanide in 16 hours against 70% in 70 hours by percolation, and with agitation the extraction of silver rose from 23·4 to 83·8%.

McConnell is of the opinion that certain ores will require dry crushing but that others are treated to best advantage by wet crushing with cyanide solution. Careful tests should be made before determining the method of treatment.

James Park† describes the cyanide practice of New Zealand. Dry crushing which was prevalent a few years ago has been almost entirely replaced by wet crushing. At the Waihi mine the ore was dried previous to crushing in brick-lined kilns excavated in solid rock. They were charged with alternate layers of ore and wood, the wood layers being about 5 ft. apart. Charges are drawn intermittently from the bottom and fresh ore is charged on top. This method of drying is not to be commended, as it is expensive and there is always some danger of unconsumed particles of carbon remaining in the ore and causing precipitation of the gold. Revolving dryers in other mills are operated at one-

* A paper read before the Institution of Mining and Metallurgy, Nov. 28, 1898.

† *Transactions of the American Institute of Mining Engineers*, September, 1899

fourth the cost of fuel as compared with the open kilns. After being broken by Gates' crushers the ore goes to stamps weighing from 900 to 1,000 lb. and dropping from 7 to 8 in. about 105 times a minute. The duty was from 1·5 to 1·7 tons per 24 hours to a 30 to 40-mesh screen. Considerable difficulty was experienced at first with dust, but housing of the battery succeeded in obviating it. It is a remarkable fact that the finer the dust the greater its value. Dust that had settled on the rafters and timbers of the mill building was assayed with the following results: From the floor to 10 ft., $34·00; 10 to 20 ft., $35·50; 20 to 30 ft., $35·75; 30 to 40 ft., $40·00; from 40 to 60 ft., $43·40. The average value of the ore treated was $20·00 per ton. The vats are of concrete in the new mill, and of wood in the old one. The concrete vats hold 150 tons each, but the ore is not charged to a greater depth than 24 in., as excessive depth creates difficulty in filtration. The first solution is applied from below, in the proportion of 1 to 3 tons of ore containing 0·33 to 0·40% of cyanide. When the solution first shows at the surface of the ore the balance is run on top, and that remaining below the filter is pumped on top. Percolation assisted by an artificial vacuum of 20 to 25 in. of mercury takes place for 30 to 40 hours, then weak solutions running from 0·1 to 0·03% of cyanide are used and are followed by water washes. One ton of water is used for each ton of ore in washing. About five days are required for the washing and the time consumed in treating each vat is 6·5 days. The consumption of cyanide varies from 2 to 2·5 lb. per ton of 2,240 lb.

The total cost of treating the ore is $0·96626 per ton, divided into drying the ore, labor and fuel $0·2406, crushing including maintenance, $0·29309, stamping including maintenance, $0·43257. The cost of cyaniding at the new mill per ton of 2,000 lb., was $0·6739 divided as follows: Labor, $0·08510; cyanide, $0·4584; zinc, $0·0384; filter-cloths, $0·0137; clean up and smelting of slimes $0·0783, making a total cost of $1·64016, not including management and general expenses. With the increase in the percentage of sulphides in the lower depths, dry crushing, owing to the partial roasting of the ore in the kiln, has not been successful. Experiments therefore are under way to determine the efficacy of wet crushing and concentration of sulphides followed by cyaniding of sands and slimes.

The Crown Mines Co., in 1897, adopted a method of crushing with cyanide solution, and direct cyanide treatment of mixed sands and slimes, the free gold being afterward saved by sluicing over a large area of amalgamated copper plates. About 0·5 tons of solution are required to the ton of ore in crushing. The whole of the pulp is conducted by a launder to one vat, and allowed to discharge into the center until the vat is half full. It is then conveyed to another vat which is also filled in the same manner.

The mixed sands and slimes in the first vat are allowed to settle for an hour or two, after which the fairly clean top-solution is siphoned off into a collecting tank, whence it is pumped up to two elevated tanks, from which the solution for the stamps is supplied. The pulp is again diverted into the first vat until the charge is complete. After settling, the clear top-solution is again drawn off. In this way three or four vats may be in course of filling at the same time.

The settlement of the slimes is effected without the aid of lime by allowing the solution to percolate from the bottom of the vat during the filling and period of settlement. The downward currents promoted by the draining from below are said to promote the settlement of even the finest material within a reasonable time. The charge is not over 30 in. deep and the treatment which follows is very similar to that of the Waihi.

For ores containing some easily amalgamated gold and fine gold, associated with pyrites and silver sulphides, the treatment used is: (*a*) Crushing with water; (*b*) plate-amalgamation; (*c*) spitzlutte separation of fine slimes, if necessary; (*d*) vanner-concentration of sulphurets; (*e*) cyanide treatment of sands by percolation; (*f*) cyanide-treatment of slimes by agitation and decantation; (*g*) cyanide-treatment of concentrates by agitation.

At the Woodstock mine the ore is chalcedonic and finely crystalline quartz. The value of the bullion varies from 8 to 12s. per ounce by being principally silver. After wet crushing the ore passes over amalgamated plate and is raised by an elevator to a Spitzlutte. The slimes from this pass to a slime tank while the sands containing some slimes and about 1% of concentratable material are run over vanners. The vanner-tailings are distributed through a simple form of automatic distributor to the leaching vat. One vat is filled at a time, the overflow of slimes from this passing to the slimes tank. The slimes are treated by agitation, and when the gold is dissolved lime is added and the clear solution decanted off. The sands and heavy slimes are treated by ordinary filtration. The concentrates which average from £30 to £40 per ton, the larger value being silver sulphide, are treated with a 4% solution of cyanide for 36 hours. Two lb. of lime are added to each ton of concentrates. The agitators hold about 1·5 tons and the recovery is said to vary from 90 to 95% at a cost of 18s. per ton of 2,240 lb. treated.

Dry crushing and cyaniding has been adopted for the ores of the Dorothy Morton mine in British Colombia. It is stated that coarse crushing decreased the time necessary for treatment. The cost, machinery, mining and transportation is given at $6 a ton. The extraction is said to be 90% of the gold and 80% of the silver contents.

The Krupp Ball Mill in Dry Crushing.—At Atacama, Chile, the Guanaco ores are crushed in Grusonwerk mills of nominally 15 tons daily capacity. These mills, however, with an expenditure of 11 h. p. crush about 6·5 tons of quartzose ores in 24 hours through an 80-mesh screen. According to Sidney H. Loram* the mills give little trouble and can be run without skilled labor. The balls and wearing parts are of "Gruson special malleable steel." The screens are of phosphor bronze. The consumption of steel from plates and balls amounts to 5·69 lb. per ton of ore ground, of which 75% is by abrasion, the balance being contained in the worn out plates, etc.

The Griffin Mill.—Thirteen Griffin mills are employed at the De Lamar mines, De Lamar, Nev., for dry crushing ore for the cyanide process. These mills have been in use there for about four years, and have been running con-

* *Transactions of the American Institute of Mining Engineers*, 1899.

stantly. Their capacity is 30 to 35 tons each day. The power consumed is 25 h. p. The cost, in Boston, is $2,250.

Twenty-four of these mills have been shipped to Western Australia to be used there for the same purpose. Ten of these were sold to the Great Boulder Perseverance Gold Mining Co., and 10 to the Great Boulder Proprietary Gold Mining Co., and four to the South Kalgoorlie Gold Mines, Ltd.*

Vats instead of Compartment Boxes.—W. A. Caldecott† describes the use of vats in place of zinc boxes in cyanide practice. The vat is designed with the object that the path of least resistance, which the solution naturally takes, shall be through the mass of zinc, thus assuring proper contact, rather than between the zinc and the sides of the vat. The vats have an internal diameter of 5 ft. and depth of 3 ft., and may be arranged in a series of three; each vat contains about 40 cu. ft. of zinc. The even distribution of the solution in the vat may be readily seen in practice from the uniform rising of the bubbles of gas to all parts of the surface. The method of introduction of the solution, the packing of the zinc between the ring and the side of the vat, the large cross-sectional area of the vat, and the absence of corners all tend to prevent the formation of channels. For convenience in cleaning up, the ring to which the sieving is fastened is made removable. The details of construction of these vats may be considerably modified and a little consideration shows various ways of introducing the solution at the bottom and taking it off at the top. For instance, the inflow pipe may be brought down the side of the vat or introduced from the bottom, and in place of an elbow square down over the center, may terminate in a ring of perforated pipe. Similarly, for convenience in cleaning up, the bottom of the vat may be made slightly conical or sloping to an outlet on one side, which delivers through a launder into a settling vat, or may be connected with the pump of a filter-press. The arrangement of a side pipe-line, enables one vat to be cut out at a time without shutting off the others while cleaning up or dressing the zinc, and an open T-piece allows the vat to be emptied of solution, and likewise prevents any accumulation of air or other gas at the angle. With a precipitation plant composed of small vats as units, the ease of rearrangement or extension to any degree is readily apparent.

At the plant of the Witwatersrand Gold Mining Co., there are two sets of these vats, with three in each series, which treat about 200 tons per day of 0·04% cyanide solution by means of the zinc-lead couple. The rate of flow established was 1 ton of solution per cubic foot of zinc per 24 hours. For a certain period the average value of the solution entering was 3 dwt., which on leaving showed a reduction to 2·5 grains. Special samples showed the following result: solution entering first vat 2 dwt. 20 grains per ton, leaving first vat 10 grains, leaving second vat 5 grains, leaving third vat 4 grains.

The Aurex Sluice and Precipitation by Mercury.‡—This patented device consists of a trough upon the floor of which curved wooden blocks are laid closely together. Amalgamated copper plates, curved to fit the blocks, and slightly smaller in order to admit of easy removal are placed upon them. The contact

* For a description of the Griffin mill, see THE MINERAL INDUSTRY, Vol. VII., p. 562.

† *Journal of the Chemical and Metallurgical Society of South Africa*, Nov. 18, 1899.

‡ *The Engineering and Mining Journal*, Feb. 3, 1900.

between them is caused by mercury being poured into intervening spaces. These plates act as the cathodes, and above them are placed lead plates curved similarly to act as anodes. These dip, however, in the direction of flow so as to cause a contraction of the stream of pulp resulting in the prevention of settling of heavy ore particles. A test plant consisted of 20 ft. of this sluice, in three lengths, forming three sides of a square, the fourth side being taken up by an elevator which lifts the discharge of the last sluice back to the head of the first. In the various experiments the pulp was circulated through the sluice for a period of 45 to 90 minutes, with dilute cyanide solutions. During the passage of the pulp through the sluice the solution dissolves the values in the ore which at the same time are being deposited electrolytically from solution. The gold and silver are eventually recovered in the form of amalgam and treated as in ordinary milling practice.

From the work, which has extended over a space of nearly two years, the following conclusions were arrived at by the inventors:

(1) This form of sluice is a most efficient amalgamator. (2) Fine and float gold is dissolved and reprecipitated during the passage of the ore through the sluice. (3) The time required to thus effect such solution is only a small fraction of that needed for the same purpose in the percolation process. (4) Slimes and clayey ores, unsuited for percolation, offer no difficulty with this apparatus, which is especially designed to treat such material. (5) Owing to the simultaneous cyanidation and amalgamation, the sluice is not only adapted for treating dry crushed ores containing coarse gold, but is especially fitted to handle the outflow from ball mills and other pulverizing machinery, in which ore can be ground with cyanide solution. (6) The principle of this device is correct, but experience has shown the need of substantial modification in the design. With these changes the sluice will be a thoroughly practical instrument, requiring for its operation no more ability than must be the possession of any competent mill man. The lead anodes, which have been used from the beginning, show no signs of wear; doubtless in time, they would have to be renewed, but the cost of this would be little more than that of the plain sheet metal. (7) The extraction of the precious metal from the solution is satisfactory and they are recovered as amalgam, requiring only retorting and melting to fit them for the market.

Use of Sea Water in Cyaniding.—The plant of the Atacama Mineral Co., Taltal, Chile, embodies nothing distinctive except that it is on the beach and sea water is used throughout for making solutions and for washing. Through the whole of this process as well as in the previous amalgamation it answers as well as fresh water. On starting a rather heavy precipitate of magnesium hydrate and carbonate was formed by the caustic soda, both added and contained in the cyanide; but now, as the quantity of sea water added is comparatively small, this precipitate is hardly noticeable. In consumption of cyanide there is practically no difference between the use of sea water and that of fresh water, while the entire absence of fresh water in the district makes it imperative to use salt water where large quantities have to be consumed. The capacity of the plant is 1,500 tons per month, the consumption of cyanide 1 lb.

per ton; the average assay of the tailings treated is $6·37 and the extraction of gold is 80%.

Precipitation from a Cupriferous Cyanide Solution.—A. H. Hartley describes* the precipitation of gold from cyanide solution at the Ulundi mine near Barberton. The ore was highly oxidized and contained a considerable percentage of copper. After amalgamation the tailings assayed from 5 to 8 dwt. of gold per ton, and after cyanide treatment they ran from 16 to 36 grains. The rate of flow of the solution through the boxes averaged from 1 to 1·5 tons per cu. ft. of zinc in 24 hours, or from one-half an hour to three-quarters of an hour's contact.

Solutions carrying from 10 to 39 dwt. showed almost complete precipitation of the gold in passing through the boxes, whether strengthened or not before entering. The strength of the cyanide solution was reduced and good results were still obtained as the following figures indicate: 0·02, 0·05, 0·04, 0·002%, cyanide solutions containing respectively 1, 2·5, 8 and 13 dwt. per ton showed traces only after passing through the zinc boxes. A solution containing 0·008% KCN to which 0·05% had been added at the head of the box showed also traces only.

When the solutions were reduced again to the average 0·009% KCN when leaving the vats, without adding any cyanide at the head of the box, though the zinc was treated with lead acetate, the solution was found to leave the boxes practically free from gold, while they contained from 1 to 20 dwt. per ton on entering. The consumption of cyanide was lowered to 0·5 lb. per ton. According to Hartley this unusually good precipitation was caused by the presence of copper. He considers that copper bearing ores should present no difficulties to the successful treatment of tailings and slimes by the usual methods of precipitation. In the discussion which followed A. McNaughtan stated that in order to get good precipitation, when treating extremely dilute solutions, he found it necessary to run them through the boxes more slowly than the rate mentioned by Hartley.

Zinc dipped in lead acetate solution was successfully employed at the New Kleinfontein mine on the Rand, where also direct filling was employed. Zinc containing 1 to 2% of lead gave excellent results.

Precipitation by Charcoal.—John I. Lowles describes† charcoal precipitation as practiced at the South German mine, Maldon, Victoria. Although it is not likely that this method of precipitation will be employed elsewhere, the description is interesting as, according to deductions from experiments by many previous investigators, charcoal was found to be an inefficient precipitant, doubtless because an insufficient quantity was used. The area of the building containing 198 filter tubs is not less than 1,000 sq. ft. Each tub is 2 ft. 4 in. high; 2 ft. 1 in. in diameter at the top and 1 ft. 9 in. at the bottom. In the center of the bottom of each tub are two small wooden cheats 4 in. apart on which rest a glazed drain pipe 4 in. in diameter with the flanged end downward. The center pipes and tubs are filled with charcoal, coarse for 6 in. from the bottom,

* *Journal of the Chemical and Metallurgical Society of South Africa*, May, 1899.
† *Institution of Mining and Metallurgy*, March, 1899.

and medium size to within 6 in. of the top of the center pipe, with about 3 in. outside and 5 in. inside the tub of coarse charcoal again. On the top of the charcoal in the tub itself, *i.e.*, all round the pipe, is a wooden lid, to keep the charcoal from floating and blocking up the outlet pipe.

The preparation of the charcoal is important. It is ground and sieved to three sizes: coarse (about 0·5 in.), medium and fines, which last are rejected. The other sizes are thrown into separate tanks of water and washed; the compact charcoal sinks, the light, porous stuff of both sizes is skimmed off, and allowed to dry before being used. About 30% of the original quantity of charcoal is lost by this treatment. Soft porous charcoal is found to be the most efficient. In this works the tubs are arranged in two groups, the first of 144 tubs in sets of six each for the strong solution of which about 400 gal. per hour passes through, and in 18 sets of three each for the alkaline wash, and the weak solution containing 0·04% cyanide, of which 300 gal. passes through hourly. The solution passing through a 1-in. branch from a 3-in. main, travels down the center pipe to the top filter in each set, coming up on the outside, over the lid, and through another 1-in. pipe to the second filter, and so on, finally passing away by an iron-lined launder to its own particular sump, whether alkaline, strong or weak. The top of each of the last five filters of a set is placed 3 in. lower than the top of the preceding one. The top filter naturally catches most gold, and, when fairly charged, is removed, and each of the remaining five tubs of the set is moved up one step, bringing second filter into first place, a tub with fresh charcoal being put into sixth position. The filter removed is relieved of some of its solution (to save loss during moving) and emptied into a large box with a filter bottom, when what solution there is in the tub drains away to the strong solution sump, leaving the auriferous charcoal ready to be shoveled out and taken to the furnace room. Every day, eight top strong solution filters are emptied in this way, so that a tub occupies first position during three days. The flow of solution from the intermediary tanks to the filters is regulated by valves in the main, and an ingenious arrangement prevents that flow being exceeded. At the junction of the mains with the distributing pipe a small open glass tube is inserted, in which the solution under the right pressure rises to a certain height. A very slight excess over the proper pressure will cause this tube to overflow, obliging the attendant to pay strict attention to the amount of solution he allows to pass through.

It is stated that solutions down to 0·04% are successfully precipitated and that the solution leaving the vats rarely shows more than 3·5 grains per ton. Tests have shown that the gold precipitated in the top filters is purer than that in the lower ones, there being a gradual increase in the amount of silver deposited so that in the last tub it is nearly pure silver. When a clean-up is made, the charcoal is burned in a reverberatory furnace of which there are four at the South German mine, each 10 ft. long. The charcoal is charged in through side doors, of which there are six to each furnace, each perforated with a number of holes to admit air to assist in burning off the carbon. The furnaces are arranged in pairs, with a stack to each pair. When charged, a wood fire is lighted in the fire-box until the charcoal is well alight, when the dampers are

closed, and it is allowed to burn itself out slowly. This it does not do completely, so it is sieved in a trommel of 900 mesh contained in an iron box; what remains in the trommel being returned to the furnace with the next charge.

The fine stuff in the box is called "ash." Every 12 lb. of ash carries from 9 to 10 oz. of gold. This ash is mixed with borax (in the proportion of 3 parts borax to 1·5 ash, (with 1 of dried and washed tailings, when the ash is alkaline) and smelted in graphite crucibles. The pots are filled and placed in the furnace, the slag skimmed from time to time, and fresh charge of "ash" added, till sufficient gold is accumulated in one pot, or the latter shows signs of wear, when it is poured into a mould. It is certain that some appreciable loss must take place during this operation, owing to the numerous small quantities of valuable ash which are dealt with. Some of the stuff is almost impalpable dust. The total cost of precipitation is said to be 1s. per ounce of bullion 900 fine recovered.

Siemens-Halske Process.—While the zinc-lead couple seems to rival this process and likely to replace it for precipitation from dilute solutions, there are advocates of it who believe that under normal conditions it can do thoroughly good work. The advantages claimed for it have been: it acts under all circumstances, whether the solution is strong or weak in either cyanide or gold, and whether acid or alkaline, and that the resultant bullion, particularly if iron, as in Andreoli's modification of the process is used as the cathode. Sidney Jennings stated that in his experiments at the Crown Deep, the use of the process was a heavy tax on the company using it, outside of the royalty. The formation of Prussian blue and other by-products has always caused some difficulty, but H. T. Durant has developed a method of recovering the gold in the former product by aërating a cyanide solution rendered alkaline, and thus dissolving the gold, which is afterward recovered in the usual manner.

J. R. Williams, in a discussion of Prof. De Wilde's process for precipitation of gold by means of copper sulphide, stated that, while the process is perfect and practically analytical, it would be impossible under the conditions existing at Johannesburg to utilize it with economy, but that in certain cases, as at the Pilgrim's Rest mine, where there is a large amount of soluble copper in the ore, and where the solution leaves the tanks with the free potassium cyanide eliminated, the process might be of advantage.

Betty-Carter Process.—There has been considerable discussion during the year concerning this process of precipitating by a zinc-lead couple.* Wm. Bettel† points out that the methods are not new, and that the clean-up from the electrolytic precipitation is much more easily managed than from the zinc-lead method. M. B. Zerener claims that the Siemens-Halske method did not have a fair trial, as it was furnished with solution which had been contaminated with zinc salts, rendering it unsuitable for electrolytic precipitation.

He thinks it inadvisable to divide the solution between the two methods, and suggests the proper way would be to divide an equal quantity of pulp, which would be treated separately.

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*THE MINERAL INDUSTRY, Vol. VII., p. 331.

† *Journal of the Chemical and Metallurgical Society of South Africa*, January, 1899.

In further discussion of the process, J. R. Williams stated that a trial run at the Nourse Deep resulted in a loss by the Siemens-Halske process of £839 as compared with the zinc-lead couple, which was returned by the Gold Patents Co., owning the Siemens-Halske patents under an agreement. So far as zinc in solution causing the difficulty he contended that while zinc undoubtedly did enter the solution from which the gold was electrolytically precipitated, yet it was unavoidable and at that particular time better work was done than when the solutions were zinc free.

In concluding the discussion Mr. Carter enlarged upon the simplicity of the method, as compared with the Siemens-Halske and endeavored to show that the method had not been anticipated by W. Bettel and others. In this he was not so fortunate.

Slimes Treatment.—The most important paper read in the year on slimes treatment in South Africa, was that of J. R. Williams,* treating on the indirect advantages of the slime plant. He stated that when slimes are treated a larger proportion of gold is obtained by amalgamation; less time is required for leaching the sand; the expense of slimes storage is obviated and there is a pronounced economy in the use of water. Going over this in order, it would seem that the increased amalgamation was due to the coagulation of the crushed particles of ore by the alkaline solution introduced when crushing, thereby bringing the slimes particles in contact with the amalgamated plate. According to Williams there is no disadvantage in using lime in amalgamation, save when hot water is used in the batteries. At the Geldenhuis Deep in 1898, without using lime 49·3% of the gold in the ore was recovered by amalgamation, while in the succeeding year 54·53% was saved, during which time lime was added. In the first year the slimes had a value of 4·21 dwt., whereas when lime was used the value was decreased to 2·93 dwt.

At the Treasury Gold mines similar results were shown on ore of the same grade in the two periods. The tailings or sands being alkaline, they need no preliminary washing, thus a longer time for cyanide treatment is given, which should bring about a higher extraction. The storage of slimes had cost about 6d. per ton, which when slimes were not treated was charged to the mill or a separate slimes dam account. By slimes treatment the other departments are relieved of a charge of 1·5d. per ton of slimes treated.

Moreover, the territory for retaining the slimes has been exceedingly limited, whereas now after treatment they can be distributed in any available place. Before slimes were treated at the Treasury mine, the monthly consumption of water was 5,038,000 gal., after the slimes plant was in operation this fell to 2,533,000 gal. The following figures are stated not to be hypothetical.

Three thousand tons of slimes assaying 2·5 dwt. should yield 75% actual extraction with a cost of treatment of 3s. per ton, but as 1·5 dwt. per ton of slimes was saved in the mill, above what was extracted without the addition of lime, and that the cost of storing 2,000 tons of slimes at 6d. per ton was obviated, a further indirect saving of £650 was made. In addition to this, from 2 to 4% of additional sands were saved in the collecting tanks. Mr. Williams, in

* *Journal of the Chemical and Metallurgical Society*, April, 1899.

response to questions, stated that the greater portion of the amount of gold saved was found at the lower part of the plates and that the alkalinity of the solution varied from 0·005 to 0·01%. The slimes plant was obliged to pay the 5d. per ton for the cost of the lime used, which, however, caused the saving of over 1·5 dwt. of gold per ton.

The discussion of this paper brought forward some interesting if contradictory experiences and views, Mr. Williams' statement that the addition of lime had a beneficial effect in amalgamation being particularly criticised. D. E. Powrie agreed with Mr. Williams that lime discolors the plates, but stated that his experience did not lead him to believe that more amalgam was caught on the lower end of the plates. In tests he had made extending over a period of six months with acid and alkaline battery water he found that 4·8% more gold was caught when the reaction of the water was acid, than when it was alkaline. The value of the slimes moreover was 9% less. A Von Dessauer stated that at the New Heriot no difficulty was experienced in using lime in the battery. A. McNaughtan agreed with Mr. Williams that more gold was recovered when lime was used, disagreeing, however, that it was at the bottom of the plates. Mr. W. Bettel stated that at the New Primrose he had used lime in excess in order to clarify the water, but that he was obliged to discontinue it as the battery manager stated that the amalgamation was imperfect, the amalgam becoming thin, while mercury ran off the plates.

Mr. Bettel was satisfied, however, that there was no diminution in the amount of gold obtained.

A. B. Inglis stated that lime had been used at the New Heriot to neutralize acid water. The immediate effect was disastrous, but by continuing a great improvement occurred. For the first eight months of 1898 the battery recovery was 53·68%, when using two bags of lime per day or about 1 lb. per ton. In the eight months succeeding the recovery improved to 55·1%. In the first period the total recovery, including cyanide treatment, was 79·9%, and for the second 82·49%.

This showed an improvement of about 2·5%. The plates were in good condition and the percentage of sands caught or settled increased from 66 to 76·15%. According to P. S. Tavener the amalgamation of the Bonanza mine increased to 67%, when lime was used and about 5% more sand was caught. The slimes instead of averaging 8 dwt. ran down to 5·5 or 6 dwt.

A. Von Dessauer, in a communication to the *South African Mining Journal,* gives some figures of the New Heriot Gold Mining Co., which confirms Mr. Williams' claim for the indirect advantage of slimes plants. The slimes at the Heriot are separated from the sand by spitzkasten and are pumped to the Jumpers Deep. Four bags of lime are used daily in the mill, costing about 1d. per ton of ore crushed. After a few months' trial the system works with perfection, but with a monthly direct loss of from £30 to £40, though indirect profits were made. Of 8,515 tons milled on the monthly average from January, 1898, to August, 1898, 75·2% equal to 6,282 tons were treated as tailings; 24·8% (equal to 2,232 tons) being stored as slimes. The yield of gold per ton of tailings treated was 6·437 dwt., of which 73·045% was recovered; the

untreated slimes yielded 3·13 dwt. The proportion of gold was, therefore, 85·3% in the tailings to 14·7% in the slimes. The record for March, 1899, is as follows: 9,110 tons milled; 7,200 tons (equal to 79·03%) tailings of a yield of 5·452 dwt. gold per ton, and an actual extraction of 79·8%; 1,798 tons of slimes (equal to 19·74% of the tonnage milled, and of a yield of 2·38 dwt. per ton) were treated by the Jumpers Deep. The proportion of gold in tailings, to gold in slimes is, therefore 89·8 to 10·2%; 1·23% of the ore milled has not been recovered, the same escaping with the overflow of the slimes spitzkasten. A comparison of both records shows, besides a lower yield of the ore treated, a larger percentage of sand retained as tailings for treatment, *viz.*, 79·03% against 75·2% last year; this is due to the erection of two return spitzkasten. Further, it exhibits a largely improved extraction of the tailings, *viz.*, 79·8% against 73·045% last year. The cyanide manager explains this in stating that he has a very much cleaner product to handle almost entirely free from slimes, which, as a result, permits perfect percolation draining to complete dryness—a highly favored condition for a good washing. The slimes have decreased in proportion in quantity as well as in yield, *viz.*, from 24·8% (or rather 24·8% less 1·23% not recovered, and equal to 23·57%) to 19·74% and from 3·13 to 2·38 dwt.

A comparison of results under present conditions and those which would exist were there no slimes plant, shows that under the new system 140·1 oz. of gold will be obtained, valued at £591 18s. 6d.; to this has to be added the difference in the cost of treatment, which is £60 15s. 3d., which gives a grand total of £652 13s. 9d. The net profit is this sum less £40 which appears as a direct loss. In addition the battery manager claims that a larger percentage of gold was recovered than formerly, especially at the foot of the plates. A further economy is effected through the smaller amount of water wasted.

Slimes Treatment at Croydon, Queensland.—At the Croydon Quartz Crushing Co.'s plant a method of treating acid slimes is in use. They are mixed with 20 to 60 lb. of lime to the ton, depending on their acidity. A disintegrator is used for this. The vat has a centrifugal pump over it, with its suction pipe coming within 2 ft. of the bottom of the tank, while the discharge pipe continues down to the floor, terminating by a right angle bend on a swivel joint so that the flow may be directed to any part of the vat. The ore is dumped into the vat after the latter has been filled with solution the pump being started. This action is kept up until the whole charge has been made. Then the pulp is allowed to settle and the supernatant liquor decanted through a swivel jointed pipe. Then the vat is filled with dilute solution and the pumping agitation repeated. Decantation takes place again, followed by dilution of the pulp with water, after which it is run to waste. The slimes averaged 9 dwt., 11 grains of gold and 2 oz. 17 dwt. of silver. The residues averaged 2 dwt. gold and 1 oz. 12 dwt. of silver, showing an apparent extraction of 78·9% gold and 43·9% silver. The cost came to $3·25 a ton, the lime costing $25 a ton.

Slimes Treatment at Yalwal, New South Wales.—Alfred Chiddey describes the method used here. The tailings were formerly passed through spitskasten to separate the slimes, but this was discontinued, as fully 40% of the gold

was carried off in the slimes. Three pounds of lime are used per ton of tailings, and a 0·1% solution of cyanide is applied. Twelve hours later the contents of the vat are leveled and a strong 0·3% solution of cyanide is pumped in. This strong solution is again followed by a weak one and again by wash water. The leaching takes 144 hours. During the first 24 hours 31·5% of the gold is dissolved, and in 48 hours, 50%; 72 hours, 70%; 92 hours, 76%; 120 hours, 79%; 144 hours, 82%. Caustic soda was used at one time in place of lime, but the action in the zinc boxes was unsatisfactory. It was found that the addition of an alkaline solution of lead caused better precipitation in dilute solutions. In cleaning up, the fine zinc is placed in a barrel 4 ft. in diameter and water added, then a thin stream of H_2SO_4 is allowed to run in, its flow being so regulated that the contents do not froth over. It is found that the tendency to froth is greater when caustic soda has been used in the vats than when lime was employed. In about six hours the solution of zinc is complete. The barrel is then filled with water and the contents after stirring are allowed to stand for 24 hours. The slimes are then washed and after the last washing are allowed to stand for 24 hours. They are then removed, dried on an iron plate, and smelted in graphite crucibles with half their weight of a fused flux consisting of two parts of soda and one part of borax. After acid treatment, a partial analysis showed gold and silver 19%, lead 7·0%, copper 12%, silicious matter 22%. A certain amount of matte is formed, approximately of the following composition: Au 2·85%, Pb 28·80%, Cu 40%, S 15%, Ag 9·15%. This was reduced with the same flux as in smelting the slimes with the addition of metallic iron. A bar of metal containing about 8% gold and 26% silver was produced. This bar is refined by means of niter in a clay pot placed inside of a graphite crucible. The slag contains some prills of gold, and is eventually sold to smelting works.

Slimes Treatment in New Zealand.—At the Waitekauri mine the ore is extremely slimy, about 33% of it being fine enough to be so termed. It is crushed with cyanide solution in the mortar and conveyed directly to the sand vat, where it is distributed by means of revolving wooden box launders actuated by a secondary shaft. The slimes are allowed to drain into a collecting vat, through a pipe with a movable elbow joint, which is raised as the pulp accumulates in the vat. The collecting vat is provided with revolving arms which keep the fine slimy pulp from settling.

From this the slimes are pumped into the slime leaching vats where they are treated by agitation and decantation, lime being added with each wash to facilitate settlement. The sands are treated by ordinary percolation. The recovery is said to be 90% at a cost of 5s. 6d. per ton.

Slimes Treatment at Lucknow, New South Wales.—That carbonate of lime is inert in settling slimes is shown by the results of treating them at the Wentworth Proprietary mine, Lucknow. The gangue here is calcite, but much diorite from the formation is crushed. The slimes are so fine that portions will pass through Swedish filter-paper. The mill has been in operation for years, and the tailings have settled so compactly that disintegration is necessary, which is effected by a revolving screen placed over a receiving tank. The slimes which have passed through the trommel are eventually pumped out into

an agitating vat with stirrers and baffle boards. The supernatant water is decanted until sufficient slimes are in the vat. Cyanide solution according to the weight of dry ore and water in the vat, is added to form a solution of the requisite strength. Agitation then takes place for six hours. On agitation being suspended the slimes are allowed to settle. This settling operation, however, takes so long a time that an expensive filter pressing plant is about to be erected. Notwithstanding the basic nature of the gangue and doubtless its alkaline reaction, this is a case where lime should have been added for its coagulating effect. The settling which would have been consequent, would have permitted the several decantations, which are invariably necessary where this method is employed. If a slime disintegrator has been used with addition of lime to the pulp, there would have been no necessity for a filter pressing plant. Another feature not to be commended in this plant is the use of a steam ejector to aid the filtration of the decanted solution. The use of steam in this manner must decompose cyanide and if zinc is used as a precipitant cause excessive consumption in the boxes. Here charcoal is used.

Rand's Continuous Process of Slime Treatment.—Ernest T. Rand describes* a new process for continuous treatment of slimes. The slimes are relieved from the excess of water by settling for three hours in a large inverted conical vat, from which the slimes are withdrawn at the center. They then flow through a V-shaped launder to the dissolving vats. Weak wash solution is run into the launder, its strength being made up to 0·015% by addition of fresh, strong solution at the head of the launder, after the solution of the gold has been accomplished. The dissolving vats are constructed with conical bottoms, their sides for two-thirds their depth are vertical, the remaining one-third inclining to the center. The vats are provided with agitators which are driven, when in motion, with a peripheral speed of 620 ft. a minute. A circulating pump drawing slimes from the bottom is used to circulate the pulp already in the vat and to withdraw fresh slime pulp from the box at the end of the launder from the concentrating vat. This pump delivers the pulp into a pan placed over the center of the dissolving vat provided with radial pipes perforated with a number of small holes.

The vats also have baffle plates to assist the mixture of the pulp. A pipe extending to the box at the end of the launder from the concentrating vat is introduced into the suction pipe of the circulating pump. This pump aërates the solution when the passage in the suction pipe, leading from the dissolving vat is checked, as the air necessarily passes down through the fresh pulp pipe. A pump having its suction pipe with valves placed half way up the slope of the conical bottom draws the pulp away from the dissolving vat at the same rate as it passes into it from the first settling vat.

By this means a constant level is maintained in the dissolving vat. As it takes seven hours to fill these vats the pulp practically receives a 7-hour treatment. The pulp drawn away from the dissolving vat is delivered to an inverted conical vat of such dimensions as to give the pulp three hours' settlement. The overflow solution from this passes through filters and then to precipitation

*Journal of the Chemical and Metallurgical Society of South Africa, May, 1899.

boxes. The pulp drawn from the bottom of this vat passes through a V-shaped launder to a second vat where it is treated with solution from the precipitation boxes. The overflow solution from this vat meets fresh slimes from the concentrating vat at the head of the first launder, where fresh cyanide is added to bring the solution up to working strength. The launders between the concentrating vat and the dissolving vat and between the first and second wash vats are provided with inclined screens in order to assist in mixing the pulp and solution and to break up the lumps. The final residues are run by gravity from the bottom of the second wash vat to the dams. The process has been experimented with at the Simmer and Jack mine with satisfactory results.

In treating pulp assaying 1·8 dwt. per ton, the residues leaving the settling vats assayed 0·3, 0·15, 0·4, 0·5 dwt. on different days. In these experiments a second washing tank was not used, but on treating the residues with clear water they were found to run only 0·05 to 0·2 dwt. per ton. The solution overflowing from the settling vat ran from 6·0 to 9·0 grains per ton.

Sampling Slimes.—Mr. L. Ehrmann* advocates mechanical sampling of the slimes by a device of his own, and in treating slimes the use of milk of lime instead of dry ground lime, as the former has the advantages of simple preparation; the solution can be easily regulated; no waste occurs; the cost of crushing is eliminated and the quality of the solution is more uniform, as slag and unburnt limestone are separated from the lime in the mixing machine.

In assaying a sample of 100 lb. of old slime, samples taken from different parts of the box which contained them were assayed separately. The assays were made on exactly the same principles; new crucibles were used and every precaution was taken. The assays, however, varied from 2 dwt. 14 grains per ton to 6 dwt. In experiments of mixing the slimes thoroughly with water and running through an apparatus provided with amalgamated plates he succeeded in getting fairly consistent samples after passing them six times.

He advocates passing the mixed sample of slime through a filter press, and cutting two opposite triangular parts from each cake; these samples then are remixed with water, and passed through the filter press again so that one cake is obtained. Opposite triangular parts, taken from this are dried and assayed. He recommends substantially the same method for the preparation of a sample of slime after treatment, but in this case washing by the special pump of the apparatus is necessary to eliminate the auriferous solution, confirming Mr. Erhmann. W. A. Caldecott† points out that some of the discrepancies in slime treatment may be caused by erroneous assays. A wet slime residue contained 10 grains of undissolved gold per ton in the solid portion, and sufficient dissolved gold in the moisture to bring the total to 25·5 grains per ton in the whole residue when dried. On assaying this dried material, the result yielded only 15, 17 and 20 grains, according to the fineness and quantity of litharge used in the assay. If previously digested with water, filtered, and the solution evaporated with litharge to secure the soluble gold, and this assayed with the washed residue, the results were 21 and 23 grains, as the quantity of water used was increased. The probable explanation of these

* *Journal of the Chemical and Metallurgical Society of South Africa*, June, 1899. † *Ibid.*, June, 1898.

discrepancies is that the soluble gold deposited in the dried slime is in such a minute state of diversion that even the lead globules from the litharge fail to collect it. Mr. Caldecott states also that in slime treatment 0·003% alkalinity is sufficient to assure their settlement.

J. R. Williams made some experiments in assaying Stourbridge clay to which gold cyanide solution had been added in such proportions that the clay contained 25 grains per ton. In assaying, a large excess of fluxes, particularly litharge, was used, and the button was much larger than that obtained ordinarily, as a large quantity of reducing agent was employed. The result of the four batches assayed showed an average of 24·3 grains returned out of the 25 grains known to exist or 97·2%. These results seem to contradict those of Mr. Caldecott's previously quoted.

Avoidable Loss of Cyanide and Gold.—It being a well known fact that all mine water dams on the Rand and the water used in milling contains from traces up to 12 or more grains of gold per ton, attaining a maximum on mines returning their mill water direct from the slimes spitzkasten to the mill, without mixing with the bulk of water in the main water dam, experiments were made whether the solution was effected by some of the ferric salts in the ore or by cyanide. These clearly prove that cyanide alone was responsible. According to A. Von Gernet the four sources of danger of cyanide contamination are:

(1) The battery; strong cyanide solutions being frequently used in dressing plates. On one mine more cyanide was used during a certain month in the battery than in the cyanide works. Care should be taken to run this solution into a separate launder, and not into the main tailing sluice. (2) Double treatment of the sands. Care has to be taken that the slat gates and pulp outlet doors are tight during the cyanide treatment in the upper tank. After transferring the charge to the lower vat, the filter should be saturated with strong solution and washed out thoroughly before refilling. (3) Slimes plant. In many plants the collecting vat for the condensed spitzkasten pulp is used for cyanide treatment also. In such a case the collecting vat should be washed out thoroughly with water and thus freed from all solution before refilling. However, the better method is to use these collecting vats for collecting only. (4) One of the principal rules in a cyanide works should be to let no solution go to waste.

Effect of Hot Solutions and Organic Matter.—There was considerable discussion in South Africa over the efficacy of a hot solution above a cold one. Mr. Butters, through a communication, stated that particularly where ores were associated with a large amount of silver, better results were obtained by heating the solutions than when they were used in the cold, and that in plants that he was erecting in Mexico he proposed to use a hot solution. The Chemical and Metallurgical Society of South Africa as a body did not agree with Mr. Butters so far as the treatment of South African ores was concerned.

J. R. Williams, in particular, suggesting that the higher extraction in the summer months was due to increased purity of water owing to the greater amount of it in the dams and to the increased chance for settlement of organic matter which it might contain. In this he was supported by F. F. Alexander, who stated that at the New Heriot mine, no higher extraction was obtained

by hot water than by cold. M. B. Zerener spoke of the effect of organic substances, where tailings which contained grass roots were treated. Precipitation is found to be almost impossible owing to the presence of organic matter.

The present writer in treating sands containing Eucalyptus and Mallee scrub, has found the same difficulty in precipitation, but moreover, has found that there is an increased consumption of cyanide in the presence of these leaves. Dr. Loevy stated that while he did not believe that solutions heated to the normal temperature of the atmosphere had greater solvent power, he was of the opinion that when they were heated beyond 35°C. the extraction would be considerably increased. A. Von Dessauer admitted that a much higher extraction was obtained on sun heated tailings at the May Consolidated mine in hot weather than during cold. A. F. Crosse made the statement that heating of tailings and heating of solutions produced a different effect, as the solubility of oxygen was greater in a cold solution than in a warm one, but when the tailings were heated they retained oxygen in sufficient quantity to bring about the necessary reactions.

Analytical Methods.—J. E. Clennel gives an outline of preliminary tests in determining whether an ore is suitable for cyanide treatment.* Two portions of the ore are taken, one of which is agitated with water and the resultant solution tested for soluble salts of iron and copper, acidity, sulphates and chlorides to ascertain if a preliminary wash is advisable. The other portion is treated with a cyanide solution varying from 0·1 to 0·2% in strength. The filtrate from this is examined for soluble sulphides, sulphocyanides and for reducing power. A solution of lead tartrate in an excess of soda is found suitable as a test for the presence of soluble sulphides, especially if cyanides are also present, since ordinary lead salts yield a white precipitate of lead cyanide, which may obscure the black or brown coloration. The reducing power is roughly estimated by a few drops of dilute potassium permanganate solution, which will be effected by ferro- and sulpho-cyanides, sulphides, organic matter, etc. If this reaction be strong, oxidation will probably be necessary before or during the practical cyanide extraction.

After these preliminary tests, experiments with 2 lb. of material are made to obtain information on the following points:

(1) The most economical degree of fineness to which the ore may be crushed for subsequent cyanide treatment; (2) the best strength of cyanide solution to use; (3) the best proportion of this solution to the ore; (4) the length of time necessary for satisfactory results; (5) the necessity for using alkalies; (6) the necessity for oxidation; and (7) the necessity for any preparatory treatment. These experiments are carried out by first crushing portions of the ore to 10, 20, 30, 40 and 60 mesh respectively, and subjecting each to a cyanide solution a trifle above the probable working strength, under exactly similar conditions of quantity, time, etc. The ore of the fineness yielding the best result is then tried with cyanide solution of 0·01, 0·05, 0·1, 0·2 and 0·5% strength. The best strength of solution ascertained, the ore, of suitable mesh, is treated with this solution in the proportions of 1 ton of ore to 1, 1·25, 1·5, 2,

* *Journal of the Chemical and Metallurgical Society of South Africa*, 1898, p. 177.

3 and 4 tons of solution. The necessary amount of solution so obtained, of convenient strength, is added to the ore of suitable mesh, and the treatment stopped after 1, 2, 3, 4, 7 and 14 days' contact respectively. The last two experiments, involving the quantity of solution and the time required for extraction are of great importance, in view of the successful practical working. Guided by the results of these experiments, a more practical trial, on about 1 ton of ore, is then made.

Effect of Sulphides on Solutions of Double Cyanides.—M. Berthelot* gives the result of investigations on the action of hydrogen sulphide and alkaline sulphides on solutions of double cyanides. He found in the case of the double cyanide of silver and potassium that there was no precipitation when a large excess of potassium cyanide was present. On addition of acetic acid a heavy precipitate occurred. The stability of the argento-hydrocyanic acid present increased with the excess of alkaline cyanides. With the double cyanides of zinc and potassium results were obtained which accorded with those on the silver salts.

Dr. J. Loevy† describes a colorometric method of determining the amount of sulphides in cyanide solutions, employing sodium nitro-prusside, which produces a red violet color with them. A solution of 40 g. of $Na_2S,9H_2O$; to the liter to which 0·2 g. of NaHO is added to prevent the decomposition of the sulphide is employed. This should theoretically contain 0·0053 g. of sulphur to each cubic centimeter, but actually a little less. A solution of 44·15 g. $ZnSO_4,7H_2O$ in 1,000 c.c., each c.c. containing 0·01 g. of zinc and a solution of 5 g. of sodium nitro-prusside in 100 c.c. of water to which 4 to 6 drops of a 5% solution of KCN has been added.

The strength of the first solution is volumetrically determined by the second, the indicator consisting of a piece of paper which has been dipped into ferric chloride, and afterward into very dilute ammonia, and which is placed in the zinc solution. Comparisons are then made in cylinders between the cyanide sample and pure cyanide, to each of which 1 c.c. of the third solution is added. The characteristic coloration produced in the sample is equalled and measured in the pure cyanide by adding drop by drop the sulphide solution diluted with water as 1:10. The limits for this method are between 0·0005 and 0·0015% of sulphur in the solution. The coloration within these limits will last from 10 to 30 minutes. If below the limit, it is indistinct and disappears too rapidly, and if above, it is too dark, and the cyanide solution must be diluted accordingly. The sulphide solution does not decompose if the bottle is full and kept in darkness.

Effect of Organic Oxidizing Agents on the Solubility of Gold.—E. Noelting and G. Forel‡ have investigated the influence of a large number of oxidizing substances, chiefly organic, on the solubility of gold in cyanide solutions, and the numerical results are given in the form of tables which cover twelve pages. From these it appears that the presence of nitrosonaphthols, nitrosophenol, nitrobenzine and orthonitrophenol increase the solubility of gold. Potassium

* *Comptes Rendus*, Vol. XXVII., March 20, 1899.

† *Journal of the Chemical and Metallurgical Society of South Africa*, 1899, pp. 54 to 55.

‡ *Bulletin of the Society of Industrie Mulhouse*, 1898, Vol. CXIX., p. 28.

ferricyanide acts in the same way, but much less energetically. Ammonium persulphate is about twice as effective as potassium ferricyanide but only half as effective as nitrobenzine, and its analogues. The action does not appear to be influenced by the presence of sodium chloride.

Determining Protective Alkali.—Andrew F. Crosse has devised a new method for the estimation of protective alkali or alkali available for neutralization of cyanicides present in commercial potassium cyanide or in mill solutions; 500 c.c. of cyanide solution is taken, 1 g. of $KHSO_4$ added and the whole boiled for 45 minutes. The HCN liberated is collected in a flask containing caustic potash, passing the vapor through a Liebig condenser. The reaction which has occurred is expressed by the formula:

$$KHSO_4+KCN=K_2SO_4+HCN.$$

The HCN in the potash solution is estimated in the ordinary way with silver nitrate. The determination of the effective alkali depends upon the absorption of a certain amount of hydrocyanic acid in the solution itself, there then being an amount proportionally less found in the potash solution.

The amount of HCN liberated by 1 g. $KHSO_4$ in presence of excess of cyanide without alkali is known to be 0·1985 g. By subtracting from this the amount liberated in treating the solution under question, the amount of alkali can be determined by multiplying the difference with the factor 1·037. The quotient gives the amount of calcium oxide present, but the result may be calculated for caustic soda or caustic potash if desired. The factor 1·037 is derived from the equation:

$$2HCN+CaO=Ca(CN)_2+H_2O.$$

The operation requires care and practice and the flask in which the solution is boiled has preferably a long, upright tube to prevent drops of cyanide being carried mechanically into the receiver, and in testing working solutions it is advisable to add about 2 g. of chemically pure potassium cyanide to the 500 c.c. treated.

In the discussion which followed several members stated their preference for Clennell's method of titrating the solution with decinormal sulphuric or oxalic acids after titration with silver nitrate. This latter is considered inaccurate for solutions where much zinc is present, though it answers well for Siemens-Halske or dilute solutions.

J. E. Clennell describes* a method of estimating copper in cyanide solution, applicable to liquors which do not contain other metals precipitated by sulphuric acid from their cyanide solutions.

The method depends upon the facts: (1) That cyanide of copper is precipitated from solutions of the double cyanides of copper by the addition of dilute mineral acids. (2) That hydrocyanic and carbonic acids have little or no action on methyl orange. (3) That when an acid is added gradually to a mixture of a double cyanides of copper with free alkali-metal cyanides and caustic or carbonated alkali, no precipitation of the copper occurs until the whole of the alkalies and free cyanides have been neutralized, the first appearance

* *Journal of the Society of Chemical Industry*, Jan. 31, 1900.

of a permanent white precipitate of copper cyanide corresponding precisely with the point at which the solution becomes alkaline to methyl orange.

A measured volume of clear transparent liquor is placed in a 100 c.c. measuring flask and a dilute sulphuric acid is added drop by drop from a burette, with continual shaking, until the turbidity formed ceases to disappear, leaving the liquid slightly milky. This point is generally sharp and definite. The burette is read and a quantity of sulphuric acid more than sufficient to precipitate the copper, is added. The burette is read again and the 100 c.c. glass filled to the mark with distilled water and agitated with its contents; 50 c.c. of the liquor supernatant to the precipitate of copper cyanide, is filtered off and titrated after addition of one drop of a 0·25% methyl orange solution by a solution of sodium carbonate, and the excess of sulphuric acid determined. By difference the quantity consumed in precipitating the cyanide of copper is found.

F. C. Carter* prefers the scorification method for assaying old crucibles, pointing out, however, that if the scorifier is at once placed in the red hot muffle the rapid volatilization of the zinc carries off the gold mechanically. Moreover it is difficult to flux the graphite thoroughly. The method adopted by him after experimenting was to take 6 to 7 g. of the finely powdered material, rub the bottom of a scorifier with silica and place 40 g. of pure litharge in it. Then add the graphite with 4 g. of niter; cover this after mixing with 30 g. of litharge, topping all off with a covering of borax. The temperature of the muffle should remain low for some minutes, then be gradually raised to white heat. For the sake of accuracy three scorification assays should be checked by three crucible ones.

Estimation of Zinc in Zinc Dust.—A method of estimating metallic zinc in zinc dust is described by Horsin Deon.† A Fehling solution prepared according to Soxhlet's method in which the two liquors are kept separate, is used to estimate the zinc, the copper not precipitated, being determined by a solution of glucose; 34·64 g. of copper sulphate contains 8·785 g. of copper, which will be reduced by 9·178 g. of zinc.

A 20-lb. sample of incrustation was taken from the chimney and flue of the slimes melting furnace of the Boatman's cyanide plant at Reefton, New Zealand. It presented the appearance of a yellowish white efflorescence that showed no metallic gold. It was crushed roughly and divided into two portions by means of a 30-mesh screen; 13 lb. passed this screen while 7 lb. remained on it. The fine sample contained 8·416 grains per pound, while the coarse assayed only 3·263 grains per pound, thus the 20-lb. sample contained 5 dwt. 12·5 grains of gold. On panning a yellow film was obtained as a tail. The microscope showed this to be finely divided particles of metallic gold. As only £1,000 in value of slimes had been melted in the furnace the loss was about 0·1%. It is not clear whether the gold was carried off by volatilization or by the draught, although the stack was only 15 ft. high.

Dangers Existing, and Precautions to take, in Cyanide Treatment.—Accord-

* *Engineering and Mining Journal*, Aug. 5, 1899.

† *Bull. de l'Assoc. des Chim. de Sucre et de Distillation*, 1898, Vol. XVI., p. 346.

ing to Dr. J. Loevy the following precautions should be taken in cyanide works: "A quantity of pure hydrogen peroxide, distilled water, and a Pravarz syringe should be kept in every cyanide plant, and all workmen on the plant should be made acquainted with the method of preparing a solution of the required strength and of injecting it subcutaneously. The hydrogen peroxide should be kept in small bottles containing about one-half ounce, with tight fitting glass stoppers and in a dark place."

The fatalities at the North Pole mine in Oregon, have shown that it is necessary to guard against poisoning by the fumes of arseniuretted hydrogen in plants where arsenical ores are treated. The probabilities are that the arsenic which enters the solution as an arsenite is reduced by the zinc to the metallic state and when this is dissolved by an acid, arseniuretted hydrogen is evolved. Similar fatalities occurred at the Mercur mine where the ore contains a small percentage of arsenic. In treating slimes where arsenic is known to exist in the ore, the solution of the zinc should take place beneath a hood to which a chimney with sufficient draught to carry off the fumes is connected.

It has been the custom at many mines to dispose of cyanide tailings by running them into the mine for filling purposes. Accidents have been infrequent, though in some localities the miners have complained of the cyanogen heating fumes causing nausea and lethargy. At the Try It mine, Paddington, Broad Arrow, West Australia, three men were filling a stope with tailings freshly discharged from the cyanide agitators. The chute choked, shutting them off from fresh air, with the result that one man died and his two companions were resuscitated with difficulty. A clause has been introduced in the Mines Regulation Act forbidding this use of tailings. It was, however, hotly contested by the mine managers who explained the cost under which they would suffer if other material had to be substituted. It was eventually left to action by the Minister for Mines in conjunction with the Chamber of Mines who would jointly draft a bill protecting the miner and the industry at the same time.

GOLD DREDGING IN THE UNITED STATES.

BY A. GROTHE.

THE use of dredges for the manipulation of gold-bearing gravels is of comparatively recent introduction. The first dredge was operated on Clutha River, New Zealand, in 1864, and consisted of a small barge provided with a leather spoon worked by hand windlasses. In 1881, the first steam dredge was built in the same locality and its success lead to the construction of numerous others, each being more powerful than its predecessor. From New Zealand the system gradually found its way to Australia, California, British Columbia and even to the Siberian placers. At present there are hundreds of dredge plants successfully at work in all parts of the world, and their number is rapidly increasing.

In the following pages the salient features of the system will be considered, together with the modifications which in special cases were necessary to overcome the adverse local influences.

The first dredges were constructed to work the gravel from river bottoms and low bars, and the majority are now used for this purpose. Their utility, however, is not restricted to this practice and it is probable that they will find more extensive application for working placers removed from navigable rivers. A number of extensive and often very rich placer deposits exist which cannot be worked by the usual hydraulic method, either because of an insufficient water supply or because a suitable dumping ground for the tailings is lacking. The operation of dredges is not confined by altitude or by any particular configuration of the ground; they use a moderate quantity of water only and dispose of the débris in the immediate vicinity and thus avoid the pollution of rivers or damage to agricultural lands; and as they handle the gravel at a very low cost, their use will doubtless be considerably extended.

Omitting further reference to the earlier efforts by hand and water power, which are now of historical interest only, we may say that the present dredge is a powerful plant, driven by steam or electricity, and capable of handling large quantities of material at a low cost. The apparatus is placed on suitable barges or scows and consists of the following parts:

(1) The dredging appliances for excavating the material and conveying it to the receiving hoppers on the barge. Sometimes the dredging plant is on a separate barge.

(2) The means for discharging the tailings at a convenient distance from the dredge.

(3) A set of winches which operate cables for moving the boat or holding it in position. In most cases supplementary vertical spuds are added.

(4) The plant for separating the gold from the gravel, producing concentrates of more or less value, according to the richness of the original material and the degree of concentration of which it is capable.

(5) Appliances to separate the gold from the concentrates, by amalgamation or other means.

Incidental to these are pumps for raising water and an electric light plant.

A boat of this description is suitable for both river and inland work. If the

latter is intended a reservoir is first excavated or dammed in and filled with sufficient water to float the dredge. The machinery is erected after launching. The mooring cables are attached to anchors or "dead-men" and dredging can then begin, the tailings being discharged at the stern. A certain area is worked out by swinging the boat by the mooring cables, the tailings being piled up behind at a height sufficient to insure a proper size and depth of the flotation basin. When the limit of area has been reached, the spud is raised and the barge moved forward or sideways. The working is not confined to level ground, as it is only necessary to allow more water to come into the basin and thereby raise the boat in accordance with the slope of the surface. In this manner the dredges at Warm Springs, Idaho City, Id., work their way gradually upward through the sloping ground on the banks of Moore Creek.

When the object in view is the removal of the gravel from the bedrock so that the latter can be thoroughly cleaned by hand, the tailings have to be carried to a place beyond the starting point, so as to be certain that all the rich contact soil is obtained. When a sufficient area has been excavated the dredge is removed and the water drained off.

DREDGING APPLIANCES.—(*a*) *Suction Pumps.*—Powerful centrifugal pumps with 12 in. suction hoses reaching to the bottom of the river are sometimes used. The end of the hose is slung in such a manner that it can be manipulated from the barge. Mechanical devices, as rotary cutters and strong water jets, have been proposed to loosen the material at the inlet, but generally the strong water current caused by the pump suffices to disintegrate the material and to elevate all sand and gravel, and even rocks approximating the diameter of the hose in size.

While this system is cheap as regards first cost and takes up but little deck-room, it has obvious disadvantages. It is very difficult to regulate the relative quantity of gravel and water brought up, and should the ground be too free, the hose buries itself and becomes choked. Furthermore, a natural selection as to size takes place; the larger rocks being less readily moved than the gravel and sand, soon form a layer which effectually stops the supply of material. The quantity of water is at all times excessive (an important consideration in the gold-saving operations) and the power consumed is out of proportion to the weight of the gravel raised. The pump-linings wear very rapidly and require frequent renewal, which adds to the working expenses and causes delays.

After many trials this method of dredging has been almost entirely abandoned in Idaho and British Columbia, with the notable exception of Sweetser and Burroughs' plant, which has for several years been working very successfully on Snake River, near Minedoka, Id., and a second plant of the following dimensions is now under construction: The boat is 105 ft. long and 20 ft. wide. The rotary pump is driven by a 75 h. p. engine and the suction pipe has a diameter of 12 in. and a length of 30 ft. A scraper is used to move the gravel, which is delivered to a sluice 6 ft. above the deck. The pump delivers 200 cu. yd. per hour.

The statement of capacity is largely a matter of conjecture, for under the conditions of working it is hardly possible to make even an approximate estimate

of the quantity treated. In the case cited above the quantity is probably too large. The particulars obtained from dredge work where the material is dumped into barges and accurately measured do not show such good results. The performance of a 12-in. suction dredge at Lowestoft, England, is given at 100 tons of gravel per hour and the quantity of water necessary to move this is five times the weight of the gravel. As the weight of 1 cu. yd. of gravel *in situ* is nearly 2 tons, the solid material brought up by the dredge was only 50 cu. yd. per hour. The consumption of coal was 1 ton for every 500 cu. yd. of gravel raised, which is far in excess of what is used on dipper or ladder dredges. As the suction dredge delivers the material to the hopper or sluice at a very high velocity, the apparatus has necessarily a low efficiency.

The following notes on the work of a suction pump under the direction of A. J. Wiley on the south banks of the Snake River in Owyhee Co., Idaho, are of interest.

The dredging apparatus, consisting of a 15-in. centrifugal pump and a directly connected 18-in. turbine, was in this case mounted on a movable timber frame. (Fig. 1.) The suction and discharge pipes constructed of No. 12 riveted steel had a diameter of 12 in.

The end of the suction was placed in a shallow pit dug by hand and worked its way down to the required depth by breaking down and sucking up the gravel at the sides and bottom of the pit. The surrounding material was then sluiced into this pit by means of a hose under pressure. The first pit was 200 ft. square by 20 ft. deep and the contents of this after passing through the sluices were discharged into the river. The dredging apparatus was then moved to the back and a second pit was excavated, the tailings being now discharged into the first pit.

The gravel and water together were elevated to a total height of 25 ft., of which 20 ft. was suction. The total volume of water passed through the pump was 10 cu. ft. per second.

The working capacity for a period of 8 hours averaged 100 cu. yd. of gravel per hour measured in bank, but owing to frequent interruptions the average over a longer period was much lower. The material handled ranged from fine sand to boulders 8 in. in diameter.

While the suction and discharge pipes did not wear rapidly the impeller and casing of the pump could not be kept in good working order for any length of time. By the use of an encased impeller and the introduction of a half-inch steel lining with a layer of rubber riveted between, these difficulties were mitigated to some extent, but on the whole the experience gained confirms that obtained in other places and is not favorable to this method of dredging.

The cost per cubic yard under the most favorable circumstances is computed at 2c. water being used as the motive power. If steam were used it would doubtless be much higher.

It should also be noted that the suction dredge is available for river work only, and cannot be used for inland dredging. The success of the Minedoka dredge is attributable to other causes, chiefly the extreme mobility of the plant, which can be shifted from one rich bar to another over a long stretch of river. No

FIG. 1.—SUCTION PUMP ON THE BANKS OF THE SNAKE RIVER, IDAHO.

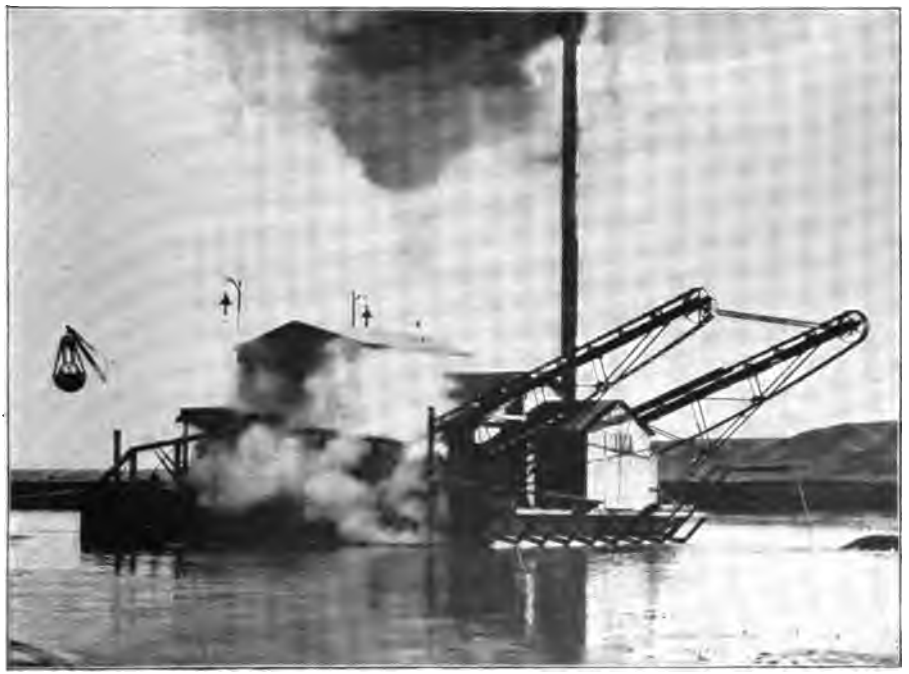


FIG. 2.—CLAMSHELL BUCKET DREDGE ON SNAKE RIVER, IDAHO, READY TO BREAK INTO THE BANK.

FIG. 3.—LADDER BUCKET DREDGE AT NYSSA, ON THE SNAKE RIVER, OREGON.

FIG. 4.—THE FRONT PART OF A LADDER BUCKET DREDGE, SHOWING THE MANNER OF OPERATING.

other boats on Snake River, or on the Fraser or Quesnelle Rivers in British Columbia, have been able to do satisfactory work with suction dredges.

(*b*) *Clamshell Bucket Dredges.*—These dredges are operated by a swinging crane at the front of a scow. They have a few commendable features, as they deliver the material moderately dry to the hoppers and afford great facilities in digging a channel without moving the boat. There is also immunity from the caving of high banks, which sometimes interferes with the working of dredges of the suction or the ladder bucket type. The consumption of fuel also is lower than in the suction dredge.

Still these advantages are more than counterbalanced by drawbacks of various kinds, and a careful trial extending over several months has convinced the writer that dredges of this type can only be used successfully under special circumstances. Fig. 2 shows a dredge of this kind on the Oregon side of Snake River ready to break into the bank. The bucket has a capacity of 1·125 cu. yd. and weighs about 4,000 lb. The crane swings all round at a radius of 25 ft. and the gravel is delivered 14 ft. above the level of the deck. There is a receiving hopper on each side of the deck into which the discharge is made alternately. The sluices and the tailing stackers of the two sides are entirely separate. The speed is 1·5 minute per complete trip or 40 buckets per hour, but frequently a stone will lodge between the jaws of the bucket and prevent its complete closing; in these cases the buckets reach the hoppers on the average only half full and the daily capacity of the dredge is reduced to 400 cu. yd. The crane requires the constant attendance of one man and as the work is arduous three shifts are necessary. The cost per cu. yd. is above 5c., an amount about five times that of a good ladder dredge.

(*c*) *Steam Shovel Dredge.*—A dredge of this type has been used on Fraser River, British Columbia, by the Beatty Dredge Co., of Welland, Ontario, in 1896-1897. At first a suction dredge was tried, but the peculiar formation in which the gold occurs in the cement surrounding large boulders and not in the finer gravel and sand, necessitated a change of plans. The present plant consists of two barges moored side by side, one containing the steam dredging machinery and the other the appliances for washing the gravel. The dredge boat is 75 ft. long, 30 ft. wide and 6 ft. deep and draws 4 ft. It is provided with a 75 h. p. boiler and the double hoisting and swinging engine is of 50 h. p. The spoon or bucket is 4 ft. 6 in. deep and of the same diameter and is provided with three steel teeth and a hinged bottom. Its maximum speed is said to be three trips every two minutes and it raises 1·5 cu. yd. every trip. Boulders up to 5 or 6 tons can be handled with ease. The weight of the bucket is 2 tons.

While the shovel dredge may have a limited application under special circumstances, its use for ordinary dredging operations cannot be recommended, the weak points being analogous to those of the clamshell bucket.

(*d*) *Ladder Bucket Dredge.*—This is undoubtedly the most economical and efficient machine for this class of work. It consists of a series of steel buckets connected by means of double links, the whole forming an endless chain which moves over drums or tumblers fixed at the ends of a ladder or frame. The

higher end of this ladder is placed above the receiving hopper, the lower end is supported by tackle which allows it to be raised or lowered according to the depth at which the work has to be done. The motive power is applied at the upper end by means of friction gear, so that any undue strain will not rupture the machinery.

The prevailing type is that used on the rivers in New Zealand. Recent improvements consist in details of pins and other minor parts, and above all in the increasing size and capacity of the machines, which allow more economical work. Fig. 3 shows a Hammond dredge of this type operating at Nyssa, on the Snake River, Oregon. The dredge built by the Risdon Iron Works for Swan River, Colorado, is provided with buckets, each elevating 5·5 cu. ft. of material.

The dimensions of the chain depend on the size of the buckets and on the nature of the material to be excavated. Where large rocks have to be dislodged or snags are encountered, the links must be much stronger than when loose gravel is dredged; if bed-rock has to be scraped, or hardpan broken up, further allowance must be made for the additional strain on the chain. In the Hammond dredge, built for use on Snake River, where the material consists of small round gravel not exceeding 12 in. in size, the weight of buckets of 4·5 cu. ft. capacity and links is only 350 lb.

The New Zealand type of dredge has the discharging end of the ladder placed well aft and the dredging end does not extend far over the bow. For dredges working on coarse gold this arrangement answers well because the gold saving apparatus, placed between the receiving hopper and the tailings discharge, requires but little room. When very fine and flaky gold is found, it is necessary to use longer sluices, and the discharge of the dredge must be placed farther forward. Fig. 4 shows the front part of a dredge, constructed for use on the Snake River, which can work from 10 ft. above to 20 ft. below water level.

The rate of discharge is from 12 to 16 per minute and a dredge with 4·5 cu. ft. buckets should therefore deliver a minimum of 120 cu. yd. per hour. As the buckets are not completely filled the average work of the dredge should not be estimated above 1,500 cu. yd. per 24 hours; furthermore, this may be materially decreased by the loss of time caused in shifting the boat to other positions. Allowing for these deductions from the theoretical capacity, the cost of dredging proper does not exceed 1c. per cu. yd. in a plant of medium size, a result unobtainable with other dredging appliances.

Disposal of Tailings.—For work on river bottoms the tailings are discharged by a chute overhanging the stern of the boat. For inland work, however, where the pond or dock in which the dredge works must be maintained at a convenient size and depth, this matter requires serious consideration. Much trouble has resulted from neglect of this important detail.

Rubber belt conveyors answer the purpose, but are rather expensive. The usual arrangement consists of steel pans fixed on endless chains which run on tumblers, or on sprocket-wheels supported on the end of beams or girders overhanging the stern of the boat. The free end is held by tie-rods connected with the general bracing system of the barge. The motive power is supplied to the

overhanging end, either directly by cables or from the lower extremity by means of sprocket chains. The speed is about 100 ft. per minute and the size of the pans depends on the maximum quantity of material which can be passed through the apparatus at any time.

Great care should be taken to discharge the tailings at a distance from the boat, leaving as much open water at the stern as possible. They should be dumped at least 30 ft. away. The height of the tailing heap depends on various circumstances. When fine and coarse tailings are discharged together, the increase in bulk of the tailings is about 30% above that of the gravel *in situ*, but when these tailings are discharged separately, as is generally the case, this increase amounts to from 50 to 60%. A dredge working from 10 ft. above to 20 ft. below water will therefore have to provide for the disposal of a mass of waste material from 45 to 50 ft. in height. Deducting the 20 ft. of water, the height of the tailing heap will reach to 30 ft. above water and to 20 ft. above the original level of the ground.

APPLIANCES FOR MOORING AND MOVING THE DREDGE.—These consist of four steel cables leading through sheaves at the corners of the boat and provided with anchors. Each cable is operated by a separate winch with friction gear; a spud also is placed near the stern, on which the boat can pivot.

In the New Zealand dredges, the winches for operating the mooring cables are placed on the deck of the barge near the driving engine and are manipulated by the engineer. This arrangement has serious objections, as the man in charge cannot overlook his work and often has to act on orders received from other parts of the boat; and the lower deck, already encumbered by the long well for the ladder dredge which separates the two sides of the boat, becomes still more crowded.

The plan of arrangement on the Hammond boats shows considerable improvement. The winches are placed on an upper deck and under direct control of the dredge-master in the pilot house, who can handle the entire apparatus without leaving his post of observation.

GOLD-SAVING APPLIANCES.—When the gold is not extremely fine a high percentage can be easily saved. The typical arrangement for this purpose consists of a grizzly (usually of rotary type) which passes the finer gravel and sand and discharges the larger material to the tailing stacker. The screened material together with a large quantity of water is led to sluices, in which a further concentration takes place. The sluices are covered with cocoa matting, on which is laid a sheet of expanded metal, consisting of a thin sheet of steel provided with a large number of crossway incisions and then drawn out in the direction of its length. The raised edges form a series of very effective riffles and the underlying fiber of the cocoa matting further assists in catching the gold, most of which is generally found within 18 in. of the tops of the sluices. A supplementary set of sluices is used to gather any heavy material which may have escaped the first set. This system, however, is quite ineffectual for saving very fine and flaky gold such as occurs in the gravels of Snake River and in the black sands of the Pacific Coast.

The gold-bearing gravels of Snake River in Idaho and Oregon, are interesting

on account of their large area and the peculiar character of the gold contained. The lowlands on both sides of the river are from a half to 10 miles wide, forming a broad and level valley between lava-covered bluffs about 80 ft. high. The soil consists of a fertile volcanic earth sometimes overlying to a depth of several feet, enormous beds of gravel which often crop out at the surface and contain from a trace of gold up to a value of $2 per cu. yd. The distribution is uneven and the pay-streak varies from a few inches to several feet in thickness.

The gold occurs in flakes and although the colors appear large, their extreme thinness makes them very light and from 900 to 1,000 colors are required to produce one cent in value. The fineness of the bullion recovered varies from 900 in the upper reaches of the river to about 800 in the lower parts of the valley. The difference is due to the silver and other metals brought down by the Boise and Payette Rivers and incidentally recovered with the gold. The composition of the gold flakes themselves does not vary and all are evidently of the same origin.

The lava beds through which the river runs extend over the whole of southern Idaho and cover several thousand square miles. They contain everywhere a trace of finely divided gold, and the gold in the gravel has probably been derived from this source. Under high magnifying power the colors appear to consist of very small particles which may well have been obtained from the millions of tons of disintegrated lava and which, after settling in favorable nooks and eddies, became welded together by the pressure of the overlying sands subsequently deposited on them. The total aggregate of gold in this field is enormous and with proper methods a large part can be profitably extracted. It is probable that in the near future this region will add materially to the gold output of the country, and it may therefore be of interest to record some of the methods employed for its recovery, the more so as they differ materially from those which give good results on coarser gold.

On a small scale, screening under water affords an easy means of concentration. One ton of gravel treated in this manner through a 30-mesh screen produced 115 lb. of concentrates containing $3·20 of gold per ton, practically all that was contained in the gravel. Large revolving screens of progressive fineness were then tried, but had to be abandoned, as it was impossible to keep them from clogging.

The best method of screening is through long sluices having a false bottom of perforated steel plates over which the gravel is washed by a powerful current of water. The dredge shown in Fig. 3 is constructed on this principle and does satisfactory work. The elevated material is dumped into a hopper where a strong stream of water breaks it up and at the same time divides it between two sluices placed one on each side of the boat and uniting near the stern. The sluices are 3 ft. wide and 18 in. deep; the fall varies from 1 to 2 in. per ft. and gradually increases toward the stern to facilitate the removal of the coarse waste to the tailing stacker, otherwise there would not be enough water, as a large proportion passes through the perforated plates with the fine screenings. It frequently occurs that when the buckets come up well filled, the rocks pile up in the sluices and the dredge has to be stopped until they are removed. It

is obvious that the large quantity of water required for moving the material produces a violent current, which prevents all of the fine sand and gold from passing through the perforations in the plates; under these conditions losses are unavoidable.

The further concentration of the fine sand screened in the sluices is then carried out by means of several tables about 30 in. wide and from 6 to 8 ft. long. They have a fall of from 0·75 to 1·5 in. per ft., depending on the supply of sand and water. The tables are covered with burlap, which retains the gold and heavy sand while the lighter tailings flow off at the bottom and are collected and discharged through a sluice at the stern of the boat.

The next operation is the clean-up which takes place once or twice in 24 hours. The dredge is stopped and clean water is supplied to the tables to carry off the lighter sands. The burlap is then removed and thoroughly washed in the collecting tank, after which it is replaced on the tables. As it takes a considerable time to clean the tables, even with the assistance of all hands, the operation is not done often enough and while the burlap is very efficient when clean, the saving is less when the cloth becomes clogged with heavy iron sand over which the gold flakes slide and are carried off in the tailings. The efficiency decreases very rapidly and the writer's experience shows that during the latter part of a four-hour's run considerable loss takes place.

Brussels carpet has been used as a substitute for burlap with good results. They wear better, and being of closer texture the sand cannot penetrate to the underside of the cloth and cause inequalities of surface. Canton flannel also is employed, either alone or with a covering of burlap.

The writer obtained best results with white cedar boards sawn in diagonal cuts from the logs. These boards have a rough surface and are covered with elastic fibers, to which the flakes of gold cling most tenaciously. They also retain a part of the heaviest sands. When the cavities between the fibers become filled with these the efficiency of the boards ceases.

The boards are half an inch thick, from 6 to 12 in. wide and 5 to 6 ft. long. They are placed on the bottom of sluices and completely cover the surface, the direction of the fiber being against the current. A heavy wire screen of 8 or 10 mesh to the inch is put over the boards, or perforated steel plates may be used for this purpose.

A very effective arrangement of this kind is built as follows: The rough unfastened boards cover the entire bottom of the sluice and are held in position by the weight of the screens, aided by a small, horizontal ridge at the lower end of the sluice which prevents them from sliding downward. The perforated plates are of No. 12 steel and have 0·125-in. holes, 0·333 in. apart. They are 2 ft. long and the full width of the sluice, which should not be more than 3 ft. in order to insure uniform distribution of the material. A strip (0·25×1 in. section) is riveted to each side at the upper end of the screens and bent half round, forming a hinge by means of which the plate rests on pins fixed in the sides of the sluice; a second pin keeps the hinge in place. The lower end of each plate rests on the hinges immediately below, so that a series of steps is produced, leaving a space of about 0·75 in. between the end of one plate and the top of

the next. Each plate can be readily removed and replaced by itself, and as the boards are of corresponding lengths they also can be easily handled. To lay the boards bare the screens are tilted, beginning at the top, and rested against stops in the sides of the sluice, leaving a clear space of 1·5 in. between the edge of the screen and the surface of the boards.

A convenient arrangement of the plant has two sluices 40 ft. long and 3 ft. wide, placed side by side in the center of the barge. The fall is 2·5 in. to the foot, and is heavy enough to move the gravel downward without the use of water in excess, only sufficient being supplied to keep the sluices clear and to wash the material thoroughly without setting up violent currents. The gravel is kept at a depth of about 3 in. and has to move at a velocity of 30 ft. per minute to overtake a supply of 100 cu. yd. per hour. The pressure of the water is 10 lb. per sq. in. supplied through two jets at the upper end of the receiving hopper; an additional spray formed by a perforated pipe is placed lengthwise over the sluices. One hundred tons of gravel require 500 gal. of water per minute, an amount only one-fifth of that considered necessary in ordinary sluicing.

The water and motion combined, separate the fine sand from the coarse material and the action is appreciably assisted by the short drops at the end of each screen. The heavier parts, consisting of the iron sand and gold, settle in the eddies and slack waters behind the larger gravel and in moving over the screens have abundant opportunity to drop through and reach the surface of the rough boards, where they are retained, while the lighter sands are carried forward by the current. Before the gravel reaches the chute leading to the tailing-conveyor, all the gold has come to a safe anchorage on the boards and none can be found in the waste, provided the boards have not yet been covered with black sand. If the operation is continued beyond this point, the iron sand will keep moving downward and the concentrates will become richer but, at the same time, a considerable loss of gold will take place. In one experiment, 160 tons of gravel were concentrated to 26 lb. of the value of $800 per ton, but with a loss of at least 40%. The safe way is to give the boards a partial cleaning hourly, removing as much of the black sand and gold as possible, and reserving the thorough cleaning of the boards, by removal and washing, for occasions when the machinery has to be idle for some other reason. The partial clean-up causes but a few minutes' delay, as it is only necessary to tilt the screen plates, beginning at the top, and wash down with a hose the material which has accumulated on the boards; during this time the current is diverted from the chute of tailing stacker to one leading to a storage tank for concentrates placed below the deck. Working in this manner on gravel containing 10c. per cu. yd. about 3 tons of concentrates of $50 per ton are produced every 24 hours.

An experimental plant, having a sluice 18 in. wide and 15 ft. long, treated 1,250 cu. yd. and produced bullion 822 fine of a value of $137·69 so that 11c. per cu. yd. was recovered.

APPLIANCES FOR SEPARATING THE GOLD FROM THE CONCENTRATES.—Very little attention has been devoted to this subject in the published reports and descriptions of dredge boats, for the reason that in all cases the gold is so coarse

and heavy (if small) that it can easily be recovered by ordinary means. When the gold is flaky, as on Snake River, the question becomes of great importance.

The Snake River gold is not only extremely light but is slightly coated and does not readily combine with mercury; the amalgamation is superficial in character, the particles of gold being held together by a thin coating. If amalgam several weeks old is treated with nitric acid and the mercury removed, the identical flakes of gold from which it was made will appear. This condition makes it very difficult to save the gold on plates, as the adhesion between the gold and plate is so easily broken that, when the little ridges attain a certain size, the black sand will carry them off to the tailings. Numerous experiments with all kinds of plates, stationary, shaken and rotary, have proven conclusively that all fail to recover this light gold.

If the concentrates, together with a little potassium cyanide and a globule of mercury are shaken in a bottle, they lose all the gold in about 15 minutes. The tests to ascertain the richness of the gravel are generally made in this manner and the results agree very well with the fire assay of the same material. In amalgamating it seems necessary to throw the particles of gold with some force against a comparatively large body of mercury and to penetrate it in order

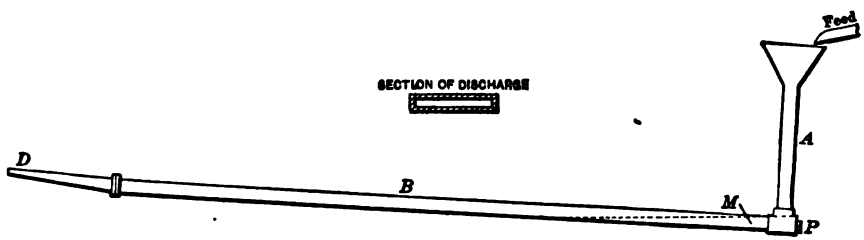


FIG. 5. WILLIAMS AMALGAMATOR.

to insure a firm hold. This effect can be produced on a large scale by means of a rotating barrel, but the separation of the amalgam from the tailings is a tedious process and uncertain in its results. It is better to use a small pan similar to those usually employed as clean-up pans in mills, and agitate the concentrates with mercury until a sample indicates that the action is complete. The speed of rotation is then reduced, more water is added and the lighter tailings are drawn off. When the bulk of the sand has been removed, a fresh charge is added and in this manner a pan of 42 in. diameter can easily treat all the concentrates produced on a 1,500 cu. yd. dredge.

Where a continuous feed is obtainable the William's amalgamator gives good results. It consists of two pipes of 3 in. diameter, fixed at right angles to each other, Fig. 5. The upright limb *A*, is about 3 ft. long and is widened out at the mouth to receive the feed. The pipe *B*, is about 18 ft. long and flattened out at the discharge end which is raised 18 in. above the lowest part of the apparatus. The frame in which the pipe is fixed, receives a duplex shaking motion; the end *A*, is moved up and down at the rate of 200 motions a minute while the end *D* is shaken in a horizontal plane at a speed of 300 per minute. Enough mercury is charged to close the communication between the vertical and horizontal part of the machine when it is at rest. A plug at *P* serves for draw-

ing off the amalgam. In this manner the pulp is violently shaken up with the mercury, and the tailings are carried on with the feed water and are ejected at the discharge end *D*, while the broken mercury is collected by the transverse shake of the apparatus and flows back to the lowest point at *M*. In a trial run on 700 lb. of concentrates, the writer found the efficiency of this apparatus to be fully 96%.

COST OF DREDGING PLANT AND WORKING EXPENSES.—The first cost of a dredging plant compares favorably with that of the appliances in use for hydraulic mining, as the large expenditure for ditches and sluices is avoided. In cases where electric power has to be generated and conveyed over long distances, the cost of construction is naturally much greater than where the motive power is steam, but works of this kind are undertaken only when the saving in current expenses will justify the increased outlay of capital. The saving in cost by electric installation is considerable, as inferred from the returns of two dredges in Montana, where the cost per cu. yd. of the steam dredge "A. E. Grater" was double that of the dredge "F. L. Graves," operated by electricity.

The first cost of a steam dredge varies between $15,000 and $60,000, according to size and locality. The latter price is an extreme one in localities where freight rates are high. The first dredge at Warm Springs, with buckets of 4·5 cu. ft. capacity, cost about $40,000; in this case the transportation of the machinery was very costly.

A dredge of the Hammond type, with buckets of 4·5 cu. yd. and a daily performance of 1,500 cu. yd., with a dredge ladder of 45 ft., can be built within a few miles of the railroad for $20,000, including electric light plant and all tools and appliances.

The following detailed estimate is given for the cost of a ladder bucket dredge having a capacity of 1,500 cu. yd., the dredge being constructed a few miles distant from a railroad siding, on the Snake River, Idaho.

Boat, including building an l launching	$3,800	Dredge ladder complete	$4,500
Main & gallows frames & hog chains or tierods	400	Tailing stacker	620
House over the boat	380	Shafting, pulleys and belts	410
Horizontal 11x15 in. steam engine	650	Pumps and water pipes, valves, etc	670
Steel sluices	540	Boiler (70 h. p. marine or similar type) and fittings	900
Two sets of cedar boards	150	Electric light plant and auxiliary engine	830
Two sets of perforated steel plates	290	Wages for erection and preliminary runs	2,500
A 4-ft. amalgamation pan and fittings	320		
Winches, cables, guides, sheaves and anchors	380	Total	$17,340

The daily expenses of a boat of the Hammond type capable of dredging 100 cu. yd. per hour based on an actual performance of 1,500 cu. yd. per day, were:

Wages.—Two dredging masters, at $4·00	$8·00	Fuel.—Two tons of coal delivered on board, at $6·00	$12·00
Two engineers, at $3·00	6·00	Stores	1·00
Two deckhands, at $2·00	4·00	Mercury and chemicals	2·00
One amalgamator and assayer	5·00	Repairs and renewals, and general expenses	20·00
One assistant	2·00	Total expense per day	$60·00

The cost per cu. yd. calculated on an output of 1,500 cu. yd. per 24 hours is therefore 4c.

There is enough ground of 10c. per cu. yd. value in gold on Snake River to keep many dredges occupied for years to come.

NOTES ON THE MOEBIUS ELECTROLYTIC PROCESS FOR PARTING GOLD AND SILVER.*

By MALVERN W. ILES.

THESE notes cover special points of construction and working, which were developed at the electrolytic refinery of the Globe smelter, Denver, Colo., and possess particular value, as they embrace many important details obtainable only in actual practice. Valuable construction data, special points in working, and costs are freely given.

In starting up the electrolytic parting plant at the Globe smelter, as installed by the late Bernard Moebius, radical changes were made which greatly improved this interesting and satisfactory method of parting and increased the profits as well.

In the "gold room," where the very finely divided gold is boiled with nitric acid, changes were made which brought this part of the process to a very high degree of efficiency, and the writer wishes to acknowledge the aid of his assistants, Mr. Gustave Nebel, Mr. M. S. Fallis, Mr. Thos. Maclear, and others at the Globe works, whose cheerful help greatly aided in the preparation of this article.

1. *The Amount of Nitric Acid Used.*—It was found that the amount of nitric acid used could be lessened very materially, and the fineness of the gold improved by straining the black gold mud through a perforated wooden sieve, to remove the numerous flat particles of doré silver invisibly mixed with it in the anode trays. This sieve is best constructed of poplar wood 0·625 to 0·75 in. thick, with a raised rim of 1 to 2 in. around the edge, and the bottom thoroughly studded with circular holes 0·25 to 0·375 in. in diameter. This sieve fits with reasonable tightness into the washing tank, and is given several coats of P. & B. paint (a specially prepared asphaltum paint readily obtainable in the American markets) for protection against the acid. In use, this sieve is held beneath the surface of the wash water in the large gold-receiving tank at a sufficient depth so that the anode tray may be easily and quickly washed or freed from its gold mud; any visible pieces of doré silver are removed by hand.

2. *The Construction of the Gold Room.*—The floor should receive special care in its construction, and the following arrangement is recommended: First, a filling of finely divided blast-furnace slag 1 ft. or more in thickness, properly leveled and well pounded down; upon this slag filling *A*, Fig. 2, 6 in. of cement concrete *B* is laid, using well screened crushed stone and the most approved Portland cement.

After a sufficient time, 2 in. of asphalt concrete *C*, is laid upon the concrete cement, and several days later a finishing course of 1 in. of Trinidad asphalt *D*, is laid, to guard against cracks. It is advisable to face all the walls in the gold room to a height of 6 ft. with two heavy coats of asphalt, and above this cover all the inner walls and ceiling with one or more good coats of P. & B. paint for protection against the strongly corrosive action of nitric acid and its vapors. This calls for great care in the design of the room so that sufficient

* For a description of the Moebius process see THE MINERAL INDUSTRY, notably Vols. II., IV. and V.

light will be obtained. The red brick drain *E*, is very carefully constructed of specially hard brick, laid in gray lime cement mortar, and its inner walls are coated with soft Trinidad asphalt to prevent cracks. In this brick drain *E*, a closely fitting wood-fiber pipe is laid, which rises through the floor in the corner of the room to a height of about 18 in., and has a 12-in. crockery funnel loosely set in it to receive the wash water, acid solutions, silver solutions, etc.

For some time after the first starting up of the gold room it is well to use a

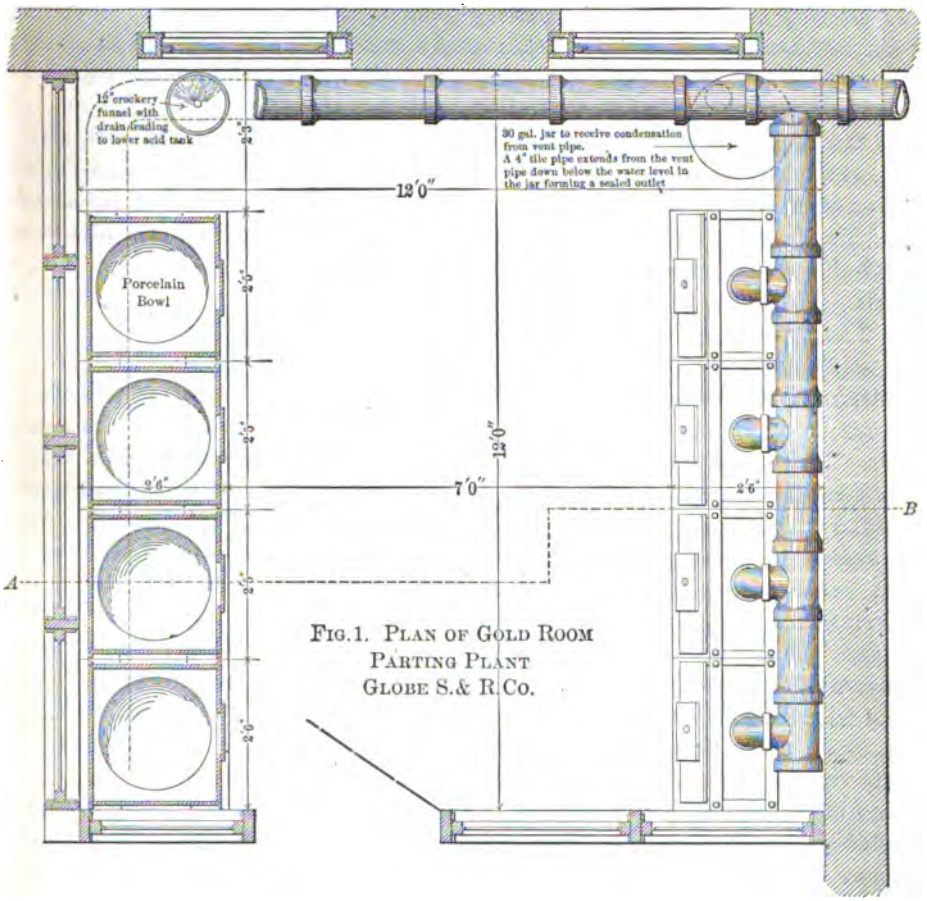

12" crockery funnel with drain leading to lower acid tank
30 gal. jar to receive condensation from vent pipe. A 4" tile pipe extends from the vent pipe down below the water level in the jar forming a sealed outlet
12'0"
Porcelain Bowl
2'6"
2'6"
12'0"
7'0"
2'6"
A
B

FIG. 1. PLAN OF GOLD ROOM PARTING PLANT GLOBE S.& R. CO.

covering of 1-in. pine boards upon the floor to preserve its smooth and level surface, and to prevent losses of gold and silver, which otherwise would adhere to the bottom of the shoes of the workmen. Even with the utmost care in the original construction, there will generally be found some pipe-joint, valve, steam jacket, or ventilator pipe which will leak. After all the leaks have been stopped, this board floor is burned up in a furnace with a bag-house attachment or other suitable means for preventing loss by volatilization, and the gold and silver recovered from the ashes.

In the gold room are eight porcelain bowls, each supported and heated by a steam jacket which conforms to the bottom of the bowl.

A hood of Vermont slate is placed over each bowl, the construction of which is accurately shown in Figs. 1 and 2. It is better to have the hand holes at the top of the hood circular in form instead of square, as experience with many materials subject to cracks—cast-iron, glass, slate, etc.—shows a weak place at

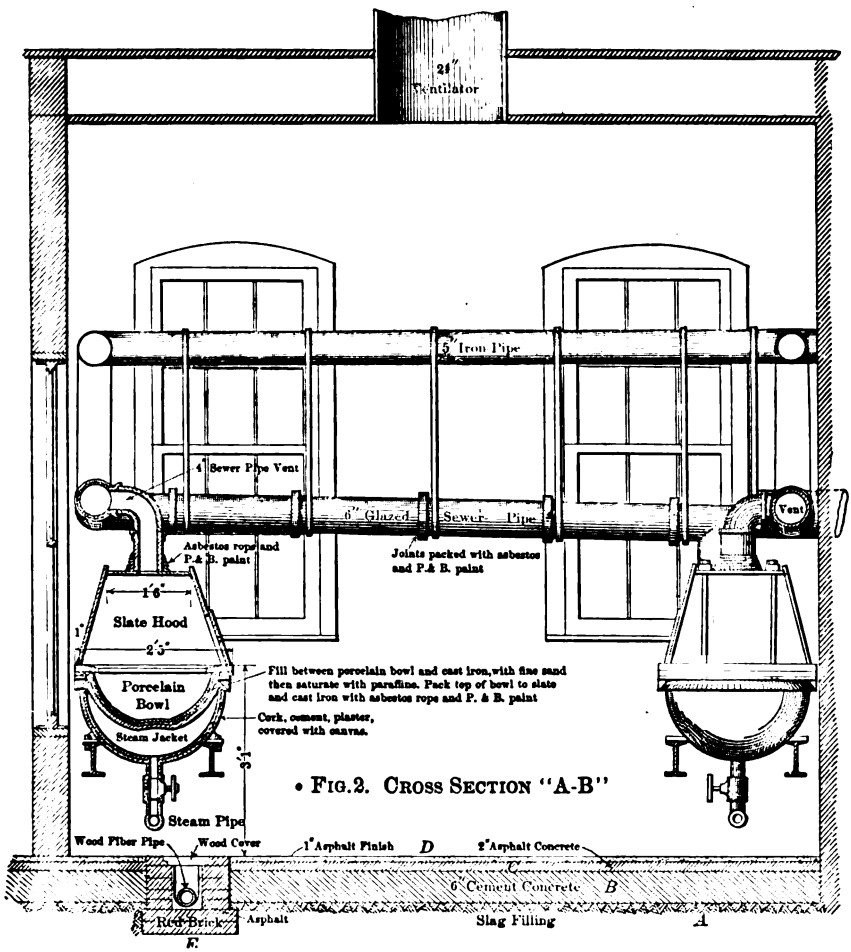


FIG. 2. CROSS SECTION "A-B"

the corners from which the cracks usually originate. The steam jackets and the superposed structure are supported upon either 8 or 12-in. I-beams, according to the number of bowls and to the span. Generally, it is best to have the space below the steam jackets as open and free as possible. Each steam jacket is provided with live steam having its own steam valve, and a small 0·25 in. pet-cock is screwed into the front of each jacket (not shown in Figs. 1 and 2); this pet-cock has a specially constructed cap to prevent leakage.

The steam jacket is encased entirely with a heavy coating of cork cement

plaster, and a cover of heavy canvas, painted with several coats of P. & B. paint. The space between the jacket and the bowl is filled with fine sand saturated with paraffin, to produce the proper temperature for good results. The steam is at least 80 lb. pressure, and an accurate steam gauge is placed conveniently upon the wall of the gold room to regulate it—a range of pressure of 10 lb. up or down is allowed. The joint between the top of the bowl and the slate and cast-iron jacket is packed with asbestos rope and P. & B. paint. The steam jackets are tested at the foundry to withstand a pressure of 150 lb., and none but perfect jackets used, as the failure of an imperfect one might cause a considerable loss of gold or silver. A broken bowl is immediately replaced. The sand and paraffin, particularly the latter, afford considerable protection to the cast-iron against corrosion by nitric acid, should any get between the jacket and the bowl.

Both the 4-in. and the 6-in. glazed ventilation sewer pipe are shown in the illustrations, and the 6-in. pipe has a fall or dip so that from every point the acid steam water, invariably carrying some silver in solution as a nitrate, will run freely into a 30-gal. stone jar placed in one corner to collect the water from all of the eight porcelain dishes. A 4-in. tile pipe extends downward from the vent pipe to below the water level in the jar, and forms a sealed outlet. The water thus condensed is found to have not only an appreciable amount of silver, but is really a diluted nitric acid of sufficient strength for use again to commercial advantage. A properly constructed exhaust fan could be used having interposed a simple and cheap mode of condensation.

The 6-in. glazed ventilation sewer pipe is carefully packed with asbestos and covered with P. & B. paint. While this particular style of construction gives fairly satisfactory results, the writer is far from enthusiastic in recommending it, and suggests a pipe of porcelain or glass, preferably the former. The 5-in. iron pipe shown in Fig. 2, serves to support the 6-in. glazed ventilation pipe by means of the bands of $0{\cdot}375 \times 3$ in. wrought iron.

A steam trap of the most approved style is placed on the outside of the gold room, and another on the outside of the parting plant. Much difficulty has been experienced in getting a good and satisfactory steam trap for this purpose; indeed the writer thinks that there is none to be had in the market, and deems it advisable to have a trap especially made, in which the ground joints are of some material not affected by dilute solutions of nitric acid or traces of silver nitrate.

The pipes from all the steam jackets should be merged into one pipe before the exhaust steam passes through the trap, and even with these precautions, it will be found advisable to allow all the water of condensation to flow into a large cast-iron kettle, of a size used at the refinery for the desilverization of the lead bullion. This kettle should be arranged so that the steam from the traps will enter an 8-in. pipe carried at least 10 ft. above the kettle, with its lower end dipping into the kettle to within 8 to 12 in. of the bottom.

To avoid any disturbance of sediment or precipitate in this large kettle the 8-in. exhaust pipe is bored with 0·25-in. holes, and the water of condensation seeks its level by quietly trickling through them. Under all circumstances

the water which has accumulated in the large exterior cast-iron kettle, should be tested for silver. This water is drawn off once or twice a week by one person only, to whom this part of the work is relegated, and the attachment for drawing off the water should be kept under lock and key to guard against loss by others tampering with it.

The trap and cast-iron settler are not shown in the drawings, yet they are quite necessary for safety. All exposed cast-iron, wrought-iron or metallic substances should be protected against the corrosive action of nitric acid and its fumes by a thorough coating of P. & B. paint. The illustrations show an excellent plan of arrangement when the problem is to part 30,000 oz. of doré silver per day of 24 hours.

The following suggestions for practice are important: All the waste products are submitted to a careful scrutiny daily; the attendants must remain within the locked doors under all reasonable circumstances, and for their greater convenience a larger space for the gold room is recommended, making it 11 ft. wide in place of 7 ft., and increasing its length at least 10 ft.

The gold room should be properly ventilated with a fan, which will deliver a stream of at least 6 in. of pure air per second. An exhaust fan might also be used.

At the top of the room is a 24-in. ventilator, extending at least 12 to 15 ft. above the roof, and painted inside and out with P. & B. paint.

It is difficult to see how loss of gold is possible through the ventilator pipe, yet the writer found it quite an appreciable one; hence the ventilator is connected with the bag-house, as are also the pipes which conduct the acid fumes of the porcelain bowls. Everything which goes to or comes from the gold room is tested for gold, silver, chlorine, copper and lead.

The wash water is tested daily for chlorine, and errors and accidents are prevented by invariably testing the nitric acid for chlorine before use. The men who work about the gold room are not allowed inside. This precaution is not only a safety measure, but prevents loss of gold that would be carried off by the shoe soles of the men. The room is carefully watched to avoid losses by theft, and losses from chemical or mechanical action.

The presence of lead in the gold, even in minute traces, is a great detriment for coinage purposes, and much thought and experimentation should be devoted to the subject of its removal from the gold before it is allowed to pass from the porcelain dishes.

The lead which contaminates the gold is believed to be a black oxide not readily soluble in nitric acid; if so, this oxide must be changed to some other oxide, salt, or compound of lead, which is soluble in nitric acid. There is room for good work in this direction.

The arrangement and construction of the gold room, as described in this paper, is believed to possess many novel and meritorious features.

3. *Inventory of Gold and Silver.*—The following records show the manner of accounting for the gold and silver produced as metal and the amount of each carried off in the various by-products; dust, fines, cupel bottoms, etc., together with final disposition of each.

INVENTORY OF FINE GOLD ON HAND IN PARTING PLANT, JAN. 1, 1899.
(On hand at close of "clean-up.")

Description.	Ounces, Troy.
In anodes, cupellations Nos. 570 to 575 inclusive	2,554·842
In anodes from scraps in tanks	262·455
In samples in office	4·000
RECORD OF BALANCE.	2,821·297
In gold bars to Mint, Nos. 317 to 326 inclusive	6,137·381
In 1,365 lb. parting plant flue-dust to bag-house	11·739
In 697 lb. fines from melting furnace flue, to blast furnaces	·781
In 445 lb. cupel bottom and skims, to blast furnaces	·768
In 326 lb. fines of No. 2 sweeps and ashes, to blast furnaces	68·193
In 4 bars (3,752 oz.) from metallics from No. 2 sweeps, and skims, to cupel furnace refinery	17·147
In 12 bars from residues, to cupel furnace refinery	9·020
In 2 base gold bars (1,070·27 oz.), to cupel furnace refinery	951·042
	10,017·368

INVENTORY OF FINE SILVER ON HAND IN PARTING PLANT, JAN. 1, 1899.
(On hand at close of "clean-up.")

Description.	Ounces, Troy.
In absorption*	20,000·00
In solution on hand	4,196·00
In anodes, cupellations No. 570 to 575 inclusive	96,448·42
In anodes from scraps in tanks	15,105·08
In samples in office	270·00
RECORD OF BALANCE.	136,019·45
In fine silver shipped for account of	
Globe, Nos. 252, 253 and part of No. 255	65,558·80
Pueblo, Nos. 175 and 176	60,063·35
Everett, Nos. 63, 64, 65 and 66	38,808·25
In 1,365 P.P. flue-dust, to bag-house	3,249·93
In 180 gal. spent acid (Nos. 33 to 38-inch), to blast furnaces	2,931·12
In 572 gal. solution, to blast furnaces	1,423·93
In 697 lb. fines from melting furnace flue, to blast furnaces	6,919·15
In 445 lb. cupel bottom and skims, to blast furnaces	1,620·20
In 139 lb. residues from upper tank and suction barrel, to blast furnaces	1,000·00
In 326 lb. fines of No. 2 sweeps and ashes, to blast furnaces	2,213·80
In 12 bars from residues, to cupel furnace, refinery	12,396·59
In 4 bars from metallics from No. 2 sweeps and skims, to cupel furnace, refinery	3,610·55
	335,815·12

* By the term "absorption" is meant the silver and gold actually absorbed by the trays, tanks, and the melting and other furnaces, as well as the particles of both gold and silver in the dust chambers, and the flues and passages connected thereto. Practically all of these absorption values will be ultimately recovered, but with new plants the absorption is often quite large, and sometimes productive of considerable anxiety.

4. *The Action of Nitric Acid on the Silver Belts of the Moebius Process.*—The late Mr. Moebius shared with others the opinion that when the silver belt is at rest, no wasting away occurs from the chemical action of the nitric acid solution, as that contains so small an amount of free acid (from 0·2 to 1%), and, in addition, the surface of the belt is protected by a thin film of oil. In actual practice, however, a considerable wear of the belts has been noticed; they rapidly became thinner and were frequently perforated without apparent cause. The various explanations offered failed to show that this wear was due to mechanical or electrical action, such as short circuiting, etc. It seemed wrong from a chemical point of view, to allow the silver belts to rest in the dilute nitric acid solution when the electric current was turned off, and a series of tests were devised to settle this question.

These experiments were carried out by Mr. Charles F. Shelby, and showed the daily and total losses, from August 15, 1898, to September 9, 1898, on

three pieces of silver belt of 10 g. weight each, immersed in different glass jars, each of which contained 150 c.c. of solution similar to that used for parting. These three solutions were of different strengths of free nitric acid. The weakest contained 0·2% of free acid, the strongest, 1% of free acid, and the intermediate solution contained 0·32% of free acid. This last corresponded to the strength of the solution in use on August 15, 1898.

Each piece of silver belt was carefully taken out of its solution daily, thoroughly washed, accurately weighed, the loss recorded, and then replaced in the solution from which it had been taken.

The table given below showed the daily loss in weight sustained by the piece of silver belt and proved conclusively that there was a gradual dissolving of the silver, although the percentage of free acid is quite small.

TABLE SHOWING THE SOLUTION OF SILVER BELTS IN NITRIC ACID.

Date—1898.	Solution No. 1, 0·20% Free Nitric Acid.			Solution No. 2, 0·32% Free Nitric Acid.			Solution No. 3, 1% Free Nitric Acid.		
	Weight of Silver Grams.	Total Loss.	Daily Loss.	Weight of Silver Grams.	Total Loss.	Daily Loss.	Weight of Silver Grams.	Total Loss.	Daily Loss.
August 15	10·000	.....	.....	10·000	.....	.....	10·000	.....	.....
" 16	9·977	0·023	0·023	9·963	0·037	0·037	9·828	0·172	0·172
" 17	9·959	0·041	0·018	9·916	0·084	0·047	8·885	1·115	0·943
" 18	9·946	0·054	0·013	9·884	0·116	0·032	8·150	1·850	0·735
" 19	9·932	0·068	0·014	9·850	0·150	0·034	7·846	2·154	0·304
" 20	9·911	0·089	0·021	9·796	0·202	0·052	7·700	2·300	0·146
" 21	9·893	0·106	0·017	9·752	0·248	0·046	7·642	2·358	0·058
" 22	9·876	0·124	0·018	9·707	0·293	0·045	7·585	2·415	0·057
" 23	9·860	0·140	0·016	9·665	0·335	0·042	7·554	2·446	0·031
" 24	9·845	0·155	0·015	9·628	0·372	0·037	7·534	2·466	0·020
" 25	9·831	0·169	0·014	9·590	0·410	0·038	7·520	2·480	0·014
" 26	9·818	0·182	0·013	9·555	0·445	0·035	7·505	2·495	0·015
" 27	9·800	0·200	0·018	9·520	0·480	0·035	7·495	2·505	0·010
" 28	9·782	0·218	0·018	9·482	0·518	0·038	7·487	2·513	0·008
" 29	9·765	0·235	0·017	9·445	0·555	0·037	7·480	2·520	0·007
" 30	9·750	0·250	0·015	9 415	0·585	0·030	7·474	2·526	0·006
" 31	9·730	0·270	0·020	9·390	0·610	0·025	7·468	2·532	0·006
September 1	9·713	0·287	0·017	9·370	0·630	0·020	7·462	2·538	0·006
" 2	9·696	0·304	0·018	9·356	0·644	0·014	7·458	2·542	0·004
" 3	9·680	0·320	0·016	9·345	0·655	0·011	7·455	2·545	0·003
" 4	9·667	0·333	0·013	9·334	0·666	0·011	7·452	2·548	0 003
" 5	9·655	0·345	0·012	9·323	0·677	0·011	7·450	2·550	0·002
" 6	9·650	0·350	0·005	9·318	0·682	0·005	7·450	2·550	0·000
" 7	9·645	0·355	0·005	9·313	0·687	0·005	7·450	2·550	0·000
" 8	9·642	0·358	0·003	9·311	0·689	0·002	7·450	2·550	0·000
" 9	9·640	0·360	0·002	9·310	0·690	0·001	7·450	2·550	0·000

It will be seen that in the experiment with the weakest solution, 25 days were required for the silver to arrive at a constant weight, and 20 days with the strongest solution.

In experiment No. 1, with 0·2% of free nitric acid in the solution, the silver lost in weight 0·23 g. the first day, and 0·360 g. or 3·6% of its original weight in 25 days.

In experiment No. 2, with 0·32% of free nitric acid in the solution, the silver lost in weight 0·037 g. the first day, and 0·690 g. or 6·9% of its original weight in 25 days.

In experiment No. 3, with 1% acid solution, the loss was 0·172 g. the first day, 2·415 g. in six days, and 2·55 g. or 25% of its original weight in 20 days.

These simple tests prove the error of allowing the silver belts to remain in the solution for a single shift or even less time.

In these experiments no oil was used on the silver, and therefore the conditions were not directly comparable with practice; yet in practice the under side of the silver belts generally has no oil, and furthermore, the belts upon standing, present spots upon the top surface free from oil. It is at these spots that corrosion takes place. In one case observed, a perfectly new belt, free from an oil film, had a circular hole cut through it, which, upon investigation, proved to be due to a leaky stop-cock immediately above the hole, that allowed single drops to fall at intervals upon the belt.

A short circuit will rapidly wear away the belt, and is chiefly due to the location of the belt, which travels too closely to the bottom of the anode trays. This fault is remedied by the use of larger slate disks which depress the belt below the surface of the solution, and cause it to travel at a greater distance from the bottom of the trays.

The entire belt is easily and completely removed from the acid solution when not in actual service, by a simple mechanism, and when so removed, the belt is supported in some inexpensive manner. Lengths of pine wood 1·5 in. sq., coated with P. & B. paint, placed across the top of the tank so that they rest upon the tank sides, furnish a satisfactory support.

5. *The Testing of the Electrolytic Parting Plant Solutions.*—Some doubt arose as to the accuracy of Gay Lussac's volumetric method of silver determination when applied to the Moebius electrolytic parting plant solutions, and ten experiments were made to prove its value.

The quantity of silver present in one liter of the parting plant solution was determined in two ways: one by the wet method of Gay Lussac, and one by the following method, devised as a fair and reasonable check, namely: To 100 c.c. of parting plant solution, hydrochloric acid was added to slight excess and the silver precipitated as silver chloride, which was filtered, washed thoroughly, and the dried filter paper and contents assayed for silver by the regular fire assay.

I then had Mr. Charles Shelby make the same experiments with 50 c.c. of spent acid solution, also with a standard solution of silver nitrate prepared by dissolving 24 g. of chemically pure silver in nitric acid, and this, diluted with water to one liter, furnished a solution of exactly 24 g. of silver to the liter.

The results obtained by the fire assay were lower than those by the Gay-Lussac volumetric method: 0·72 g. lower with the parting plant solution, and 0·70 g. lower with the standard silver nitrate solution.

The complex character of the parting plant solution does not materially interfere with the accuracy of the determination by the wet method.

The low results in the fire assay are due to slight losses in filtration, chemical and mechanical losses in converting the silver into a metallic state, and cupellation losses from volatilization and cupel absorption.

Taking these facts into consideration, it is of interest to note the agreement between the two methods of determination.

The results of the experiment were tabulated as follows, viz.:

GRAMS OF SILVER PER LITER.

	Wet Assay.	Fire Assay.	Difference in Weights.	Percentage of Difference in Terms of Wet Assay Weights.
Standard Silver Nitrate Solution containing 24 Grams of Silver per Liter	24·00	23·30	0·7	2·9
Spent Acid Solution	135·00	128·00	7·0	5·2
Parting Plant Solution:				
September 12, 1898	20·00	19·30	0·7	3·5
" 13 "	22·00	21·20	0·8	3·6
" 14 "	20·00	19·20	0·8	4·0
" 15 "	20·00	19·80	0·2	1·0
" 16 "	19·50	18·20	1·3	6·6
" 17 "	19·00	18·10	0·9	4·7
" 19 "	17·00	16·50	0·5	2·9
" 20 "	16·50	15·80	0·7	4·2
" 21 "	15·80	15·20	0·6	3·8
" 22 "	15·00	14·30	0·7	4·7

6. *Improved Contact Points in the Moebius Electrolytic Process.*—At the electrolytic parting plants of the Guggenheim smelter, Perth Amboy, N. J., and of the Globe works, Denver, Colo., the electric contact point consisted of looped platinum wire which rested on the top of the anode bar of doré silver. These platinum points rapidly dissolved and required replacement with consequent cost and loss of time.

The life of the contact wire was materially lengthened when it was of larger section; this, however, involved a much greater cost from the larger quantity of platinum required.

I induced Mr. Gustave Nebel, a co-worker with Mr. Moebius, to make a silver contact point, circular in section and 3 in. long, with a taper from 0·437 in. at one end to 0·312 in. at the other. The smaller end rested directly on the anode bar and the larger end was attached to the 0·25-in. copper conducting wire of the anode tray.

These first silver contact points were so satisfactory that split moulds were made, similar to an old-fashioned bullet mould, and by their use these silver "points," or "pencils," were easily cast direct from the silver melting furnace.

The mould was first made to contain 22 points, the total number required to equip one tank. In practice, however, a mould of half this size was found more satisfactory. Many of the points were defective when cast, but as these imperfect ones were put back at once into the melting furnace, this objection is not a serious one. To demonstrate the practical value of these points, a tank was fitted with them and put into operation November 17, 1898. The voltmeter measurement showed a saving of half a volt and when calculated for all the tanks, 48, the total saving was 24 volts, equivalent to 8 h.p.

This experimental result was afterward checked by a six months' run of the process with all the tanks equipped with silver points. The voltage on each tank was frequently ascertained and the results showed an actual saving of 25 volts at the end of the run.

From calculations made by Messrs. T. W. Maclear and Gen. Irving Hall, the cost of one horse power per year at the Globe smelting works was $36, and with a complete tank equipment of silver contact points instead of the usual platinum points, the total saving per year in power cost represented $36 × 8 = $288.

The cost of equipping a plant to part 30,000 oz. doré silver in 24 hours with silver contact points was figured as follows:

1,056 silver points (48 tanks of 22 points each), weight, 864 oz., at 60c. per oz., cost $518·40. To this amount was added the labor cost in the preparation of these 1,056 points—24 days at $2·50 per day, $60—gave $578·40 as the total cost.

The cost of equipment with the larger sized platinum points was as follows: 1,056 points at 80c. each, $844·80 (cost of platinum assumed to be $17 per oz.). The labor involved in this case was two days at $2·50=$5, which, added to the cost of the platinum, gave $849·80 as the total cost.

The total weight of the platinum tips required to fit up the 1,056 trays was 50 oz., and as these tips wore away the platinum fell in part to the anode trays and was eventually weighed up as gold.

The value of gold was considered at $20·67 per oz., and platinum as $18 per oz., the difference, $2·67, represented a credit to the company for each ounce of platinum which went with the gold, and with the generous assumption that none of the platinum was dissolved and none remained attached to the copper conducting wire, *i.e.*, all the platinum went with the gold, the total weight of the platinum, 50 oz., represented a value of ($50 × $2·67) $133·50. This amount deducted from the cost of the platinum ($849·80—$133·50) gave $716·30 as the actual cost of equipment with heavier platinum points.

The total cost of the silver point equipment was $578·40, and contrasting this with the cost of the platinum point equipment, there was a balance of $137·90 in favor of the use of silver in place of platinum.

To this amount, $137·90, was added the saving in power per year of $288, which gave $425·90 as the total sum per year saved by the use of the silver points in place of the platinum ones. The use of the silver points offered other advantages.

Silver has an electric conductivity of 100, while that of platinum is 10·5 and when a short circuit occurred, the platinum being the poorer conductor became heated to a red heat, while the greater conductivity of the silver prevented its heating to any such extent.

It required constant care and attention to prevent the heating of the platinum, and this was one of its objectionable features. The silver points when properly made lasted as long as platinum points.

7. *Cost of Parting by the Moebius Process.*—The cost of parting doré bars at

SUPPLIES.	
Oil	$56·20
Nitric acid, 1,698 lb. @ 7·5c.	127·35
Cotton waste, 113 lb. @ 9·5c.	10·73
Coal, 25·47 tons @ $2·25	57·31
Coke, 1,543 lb., @ $9·50 per ton	7·32
Cupels for silver melting	6·75
Crucibles, 2, No. 40 @ $2	4·00
Sundry supplies	21·12
	$290·78
LABOR.	
Assistant superintendent	$160·00
Five men, 31 days	379·75
Superintendent, half time	200·00
	$739·75

INTEREST.	
$200,000 for 1 month, @ 10%	$1,666·66

RECAPITULATION.		Cost per 100 oz.
Supplies	$290·78	0·0427
Labor	739·75	0·1087
Interest	1,666·66	0·2450
	$2,697·19	0·3964
Royalty		0·1000
Total cost		0·4964

the refinery of the Globe smelter, Denver, Colo., for May, 1898, is given in the preceding statement, and during this time, 680,133 gross ounces of doré bullion were treated.

This statistical data presents several interesting points for discussion:

SUPPLIES.—(*a*) *Oil.*—$56·20.—The oil was mainly used to prevent the close adherence of the electrolytically deposited silver crystals to the belt; a valuable fact discovered and successfully applied by Mr. Nebel.

If the belt were not covered with this oily film, the deposited silver would cling firmly to it, eventually increasing the thickness until it becomes no longer sufficiently flexible to pass the slate rolls with ease.

(*b*) *Nitric Acid.*—1,698 lb. @ 7·5c.; $127·35.—Attention is called to the high price—7·5c. a lb.—of the acid used. There are many places in the United States where acid of the same quality and strength can be procured for 3c. a lb. The amount of acid then used, 1,698 lb., was excessive, but was greatly reduced later by the use of the filtering sieves or trays for the gold mud already mentioned under Section 1 of this article.

(*c*) *Cotton Waste.*—113 lb. @9·5c.; $10·73.—This requires no special mention as the amount is small and the price regular.

(*d*) *Coal.*—25.47 tons @ $2·25; $57·31.—The amount and price are reasonable and call for no special discussion.

(*e*) *Coke.*—1,543 lb. @ $9·50 per ton, $7·32.—The amount of coke is small, but of better quality than that used for the blast furnace.

The items for cupels, crucibles and sundries are sufficiently small in comparison with the total cost to dismiss them without further comment.

Labor.—$739·75.—The item of labor might be regarded as high, yet, when one considers the usefulness of specially trained scientific men and artisans, together with the fact that all the men, laborers and attendants as well, must be selected with special reference to their honesty, this item of labor cost is not unreasonable.

Interest.—$200,000 for one month @ 10%, $1,666·66.—Many mercantile industries both in Europe and America would consider their business a good one if they made 10% interest.

Royalty.—The royalty charge was 0·1c. per oz.

The total cost of parting 100 oz. of doré silver under the above conditions of high interest charge was practically 50c., and of this, the interest charge of 24·5c., nearly doubled the cost of supplies and labor, which was 15c. If the assumption is made that money could be obtained at 5%, the interest charge would then be reduced to 12·5c., and the total cost of parting doré silver would become 37·5c. per 100 oz.

At most places, with careful management, the writer is of the opinion that the cost of parting doré silver by the Moebius electrolytic process could be reduced to at least 30c. per 100 oz.

GRAPHITE.

THE production of graphite in the United States in 1899 was 3,632,608 lb. ($145,304) against 1,647,679 lb. ($148,291) in 1898, an increase in production that more than doubles the 1898 output.

The mines at Ticonderoga, N. Y., and in Chester County, Pa., continued the production of crystalline graphite while the amorphous variety was produced mainly at the Rhode Island mines. The carbonaceous mineral from Baraga County, Mich., the so-called "Baraga" graphite, is simply a carbonaceous schist. This substance is not included by us in the statistics of graphite production in the United States, but is classed and reported as slate pigment.

The localities wherein there has been special activity in 1899 are in northeastern New York, eastern Pennsylvania, Massachusetts, and Alabama.

For many years the graphite industry has been dominated by the Ceylon mineral, and the recent increased demand for it has served to stimulate activity both in the production of and prospecting for graphite in the United States. The few producing mines have increased their output materially and some abandoned mines have been reopened. Several new graphite companies have been formed, and while some of these were "wild-cat" schemes based on very poor deposits and lack of business ability, the trade will be benefited ultimately by the activity and investigations of their operations, even though they were failures. These unfortunate conditions will demonstrate that the simple possession of a graphite mine will not tend to success unless backed by proper business methods and general knowledge of the trade.

It is worthy of note that the increased production of American graphite has made a distinct impression on some of the old traditions of the business. For many years certain grades of Ceylon graphite, by reason of these traditions, have had almost a monopoly of a good part of the trade, but during 1899 the field for American graphite became greatly enlarged and it was proved that graphite of a certain standard of purity and formation is suitable for certain uses, irrespective of the locality of its production.

The present uses of graphite include the manufacture of pencils, crucibles, stove polish, foundry facing, paint, motor and dynamo brushes, anti-friction compounds, electrodes for electro-metallurgical work, conducting surfaces in electrotyping, and covering the surfaces of powder grains. For most of these purposes it is used in the natural impure state, while for others it is necessary to render it quite pure and free from grit. The general method of purification is accomplished by a method worked out by Brodie, which consists in first grind-

THE GRAPHITE INDUSTRY IN THE UNITED STATES.

Year.	Production. (a)		Imports.		Consumption.	
	Pounds.	Value. (b)	Pounds.	Value.	Pounds.	Value.
1895	377,450	$17,286	20,124,160	$260,111	20,516,168	$277,751
1896	405,006	18,225	33,824,000	437,189	34,229,006	455,414
1897	993,138	44,691	19,113,920	270,952	20,107,058	315,643
1898	1,647,679	82,385	30,199,680	743,820	31,847,359	826,205
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953

(a) Production of refined graphite only; of amorphous graphite there were produced 840 short tons ($4,700) in 1895, 574 short tons ($3,850) in 1896, 1,200 tons ($11,400) in 1897, 1,900 tons ($11,400) in 1898, and 1,080 tons ($8,240) in 1899. (b) Nominal.

ing, or otherwise reducing the graphite to a state of fine subdivision, washing out the heavier impurities, mixing 14 parts with 1 part of potassium chlorate and 2 parts of concentrated sulphuric acid, heating on a water-bath for some hours, washing thoroughly and afterward roasting at a red heat. If silica is present a treatment with hydrofluoric acid is added to the process.

Market Conditions.—The activity of the graphite industry was not confined to its mining and prospecting, but extended to its manufacturing interests as well. The continued activity in all metal trades, particularly those involving the use of crucibles—copper alloys for electrical and other supplies, crucible steel, etc.—has served to maintain an increased consumption so great that it was barely met by the increased production. As a natural result, the price followed the increase and during 1899 averaged at least 50% higher than in 1898.

The scanty supply in the market has afforded an opportunity for the sale of stocks of inferior grade. Goods which could find no market at the low prices ruling in 1894-95 have been unloaded at high prices during 1898-99. Hence while the general average of prices has been higher in these latter years, the quality has been inferior.

Alabama.—Very good samples of graphite have been derived from this State and the deposits are reported to be extensive. The Alabama Graphite Co., operating mines in Clay County, produced, it is stated, from 50 to 60 short tons in 1899. Deposits occur also in Cleburne County, but these are not being worked at present.

Massachusetts.—It is reported that the graphite mine near Southbridge is to be reopened and worked. This mine was operated for a short time by the Tudor Co., of Boston, some 70 years ago, since when good sized trees have grown up in the old open cut workings. The graphite is crystalline and granular and is associated with a feldspathic rock. The vein is about 4 ft. wide and extends for several hundred feet.

New York.—For many years the only producing graphite mine of the United States was the Hooper property in Hague, Warren County. It is still the only successful and profitable graphite mine in the State. This property was described in detail in THE MINERAL INDUSTRY, Vol. VII. The mine has been owned and operated for years by the Joseph Dixon Crucible Co., of Jersey City, N. J. The ore is of low grade and difficult to treat. The output in 1899 was largely increased, and the company is now extending its plant. Con-

siderable excitement has been shown in graphite prospecting in the region of the Dixon mine in Essex and Warren counties. Mills have been built and others projected. Many reports have been circulated, but as yet no evidence of production has been forthcoming.

Pennsylvania.—(By T. C. Hopkins.)—This State showed a strong activity in 1899 and produced five times the output of the previous year. The Pickering Valley region has increased its output and other properties have been taken up. Contemplated developments, if carried out, will make this region of much importance in 1899.

The Pennsylvania Graphite Co., in 1897, opened up on a large scale, the Riley mine, three-quarters of a mile south of Mertztown, Berks County, that has been inoperative for many years past. This company extended the works, and in July, 1899, was producing at the rate of 2 tons of graphite per day. Three-fourths of the product was flake graphite; the remaining fourth was of a lower grade, and was utilized in the manufacture of graphite paint.

The graphite occurs in what might be classed as a fine-grained conglomerate or a coarse-grained sandstone, which in the vicinity of the graphite mine is weathered to a considerable depth. The deposit is lens-shaped, and stands nearly vertical. The maximum thickness of the "vein" is said to be 39 ft., thinning out in both directions. Two shafts, one to a depth of 90 ft., have been sunk into the graphite, and galleries have been extended from the shafts. The graphite contains more or less rock, running as low as 25%; other specimens contain as high as 80%. The average yield of graphite from the mine is 28% of the material handled. The company claims that with their present facilities 10% ore can be cleaned with profit. The by-product is quartz sand, which is sold to the foundrymen for moulding purposes.

The Boyertown Graphite Co. reopened the old graphite mine, a mile west of Boyertown, Berks County, and erected a mill for the separation of the graphite from the gangue material. The graphite occurs in a mica schist, and is mined through a shaft 54 ft. in depth. The rock containing the mineral is crushed, washed, dried and separated. The refined graphite is sacked and hauled by wagon to Boyertown, where it is shipped to the purchaser.

Rhode Island.—The higher price of Ceylon graphite has made an opportunity for an increased production of the so-called graphitic-anthracite of this State. This term hardly describes the material, and while it is undoubtedly carbon, more or less graphitized it differs in many respects from the crystalline grades. It is valuable as a facing for foundry purposes and in some polishes.

The Rhode Island Graphite Co. reported that its production in 1899 was about the same as in 1898, and gave the prices of $8 per ton for lump and $33 per ton for manufactured f. o. b. at Providence.

South Dakota and Wyoming.—The graphite reported in these sections is amorphous. It would probably give good results in foundry facings. We have no figures of production in 1899.

Austria.—Workable deposits of graphite occur in Styria between the valleys of the Enn and the Lemmering Rivers. The country rock is a graphitic schist of varied character, to which Dr. Weinschenk gives the name "graphite-phyllite."

WORLD'S PRODUCTION OF GRAPHITE. (IN METRIC TONS.)

Year.	Austria	Canada	Ceylon.	Germany.	India.	Italy.	Japan.	Russia.	Spain.	Sweden	United States. (b)	Totals. (e)
1894.....	24,121	63	10,718	3,133	1,623	1,575	1,091	(a)	10	107	350	42,683
1895.....	26,443	199	13,711	3,751	(a)	2,657	77	(a)	*Nil.*	7	171	49,050
1896.....	35,972	126	10,463	5,248	(a)	3,148	215	(a)	*Nil.*	14	184	55,356
1897.....	38,504	396	d19,275	3,861	61	5,650	204	(a)	(a)	99	450	69,311
1898.....	33,062	1,107	d78,509	4,593	22	6,435	(c)	98	10	50	824	124,660

(a) Not reported in the government statistics. (b) Crystalline graphite. (c) Statistics not yet published. (d) Exports. (e) The United Kingdom produced 41 tons in 1895, which is included in the total for that year.

Within this schist are found seams of rich graphite ore, which have been worked at numerous points, but particularly around Mantern and Kaisersberg. The graphite is of two distinct and separate types. The first has all the appearance of anthracite and is of massive habit but chemically and physically it behaves like graphite. Analyses show as high as 90% C. Dr. Weinschenk considers it to be a metamorphic product of coal. The other variety is soft and possesses the characteristics of amorphous graphite. It is probable that the latter at one time had the same coal structure, but was later subjected to further metamorphism. This theory is supported by the geological conditions surrounding the deposits.

Canada.—The Canadian graphite is crystalline and of good quality. It has been reported as occurring in extensive deposits, and while operations have been carried on for a number of years, but comparatively small quantities have appeared in the market. The production in 1899 was 1,220 short tons valued at $16,179.

Mexico.—The United States Graphite Co., of East Saginaw, Mich., has for several years been operating a graphite mine near Yarres, in Old Mexico. The graphite is very finely crystalline, in fact, almost amorphous. The ore occasionally runs as high as 96% C. The poorer grades are utilized for foundry facings while the better quality of material is carefully sorted and ground for special uses. The output is about 1,200 metric tons per annum, the more part of which is exported.

Artificial Graphite.—(By E. G. Acheson.)—The Acheson Graphite Co. was organized in January, 1899, and took over all of the patents and business that had until then been operated through the courtesy of the Carborundum Co., whose plants and furnaces had been used in conducting the work. The use of graphitized anodes and electrodes has shown a remarkable increase for the trade, which amounted to 200,000 lb. in 1898, reached 405,870 lb. in 1899, and the year closed with indications that 1900 will double the large output of 1899. The manufacture of specific grades of graphite in bulk, for use in the manufacture of crucibles, paint, foundry facings, etc., has made progress in the perfection of the process, although the actual amount of graphite produced and marketed was almost negligible.

The most important development of the year was the advancement of the manufacture of graphite from amorphous carbon and carbonaceous materials. In a paper read before the Franklin Institute* I briefly outlined the theory

* *Journal of the Franklin Institute.* March 15, 1899.

upon which I was working for the formation of graphite, a summary of which is as follows:

"(1) Graphite is the form that carbon assumes when freed from its chemical associations, under the conditions of low pressure and protection from chemical influence. (2) Diamond is the form that carbon assumes when freed from its chemical associations under the conditions of high pressure and protection from chemical influence, and, by inference, (3) amorphous carbon is the form that carbon assumes when freed from its chemical associations under the conditions of high or low pressure and exposure to chemical influence.

"This theory must be supplemented by the hypothesis that catalytic action occurs under certain conditions, during the transformation of amorphous into graphitic carbon—probably the progressive formation and dissolution of carbides."

Later in the year my work developed into a recognition of a distinct class of natural carbonaceous materials, exemplified in anthracite coal—which was found possible of conversion into high grade graphite* with many of the qualities necessary for the manufacture of crucibles, paints, lubricants, and the various uses to which the several grades of natural graphite had been put. I have made some 12 or 14 tons of graphite from anthracite—(anthracite culm may be used for this purpose)—a part of this has been made up into crucibles for steel refining, and is now being tested in the several steel works—the remainder of the product has been sold for manufacture into paint. This particular lot showed on analysis an average purity of 93·5% graphite.

The Acheson Graphite Co. contemplates extending its plant which now consists of a building 100 ft. square, and has ordered an electrical equipment of 1,000 h. p. It has also contracted with the Niagara Falls Power Co. for 1,000 electrical h. p., for the furnaces to be erected in this new building for the conversion of anthracite coal into graphite. With this equipment an output of 6 to 7 tons per day of high grade material is anticipated. The electrical furnaces to be used are of special construction,† in which the material to be subjected to the heat of the electric arc is placed in a mass or pile, the sides of which have the natural slope due to gravity. The zone of highest temperature is maintained by the electrodes at a point between the top and the bottom of this pile. The treated material is withdrawn from the base and fresh material is added at the apex from time to time, in this way the operation may be continuous. The material does not come in contact with the walls of the furnace until it has passed out of the zone of highest temperature, an important point with regard to furnace repairs. In special instances the material is cooled by water-cooled pipes which extend through the bottom of the pile. The screen conveyor for removing the treated product may also be water cooled.

In the use of a furnace of this type for graphitizing coke, a current of 12 volts and 37,500 ampères was required to give a product of 500 lb. per hour; the electrodes in this case were approximately 20 in. apart, and had a cross-section of 1,250 sq. in.

* United States Patent No. 645,285. † United States Patent No. 645,284.

GYPSUM.

THE production of gypsum in the United States for 1899 showed an increase, and as in the previous year it is principally confined to the Western States, but at the same time imports increased, owing mainly to the Eastern States being dependent upon Nova Scotia and New Brunswick for their supply of gypsum and plaster of Paris. The total production of crude gypsum in the United States in 1899 was 372,285 tons, against 314,865 in 1898. Prices were well maintained during the year principally in the Western States, where increased values stimulated production. Kansas shows the most remarkable production, having increased from 49,720 short tons in 1898 to 82,016 in 1899, which surpassed its record production of 1897. Texas shows an increase of 50%, and Iowa likewise showed considerable increase in production.

PRODUCTION OF GYPSUM IN THE UNITED STATES. (*a*)
(In tons of 2,000 lb.)

States.	1897.	1898.	1899.	States.	1897.	1898.	1899.
California	2,750	3,875	3,663	Oregon	*Nil.*	150	500
Colorado	1,190	1,570	*e* 1,600	South Dakota	6,000	3,750	600
Ind. Ter. and Oklahoma	11,120	15,229	20,750	Texas	17,161	24,417	34,214
Iowa	24,478	38,338	51,958	Utah	2,500	*e* 3,000	1,700
Kansas	62,556	49,720	82,016	Virginia	7,150	8,125	12,773
Michigan	95,040	93,181	*e* 95,000	Wyoming	3,875	3,633	2,817
Montana	425	400	304				
New York	47,164	46,477	39,390	Total	300,369	314,865	372,285
Ohio	*e* 19,000	*e* 23,000	25,000				

(*a*) Statistics reporting the amount quarried. (*e*) Estimated.

GYPSUM IMPORTED INTO THE UNITED STATES. (IN METRIC TONS.)

Year.	Ground or Calcined.			Unground.			Value of Manufactured Plaster of Paris.	Total.
	Quantity.	Value.	Per Metric Ton.	Quantity.	Value.	Per Metric Ton.		
1895	3,348	$21,526	6·42	196,680	$215,706	1·10	$10,352	247,583
1896	3,348	22,058	6·58	183,165	193,549	1·06	11,722	227,329
1897	2,707	17,028	6·29	165,812	178,686	1·08	16,715	212,429
1898	3,021	18,500	6·12	168,723	181,364	1·09	40,979	240,843
1899	3,317	19,250	5·80	199,724	220,603	1·10	58,073	297,926

PRODUCTION OF GYPSUM IN THE PRINCIPAL COUNTRIES. (*a*) (IN METRIC TONS.)

Year.	Algeria. (*b*)	Canada.	France. (*b*)	Germany. (*c*)		Greece.	India.	United Kingdom	United States.
				Baden	Bavaria.				
1894	23,900	202,877	.....	40,652	25,267	85	3,220	155,905	273,553
1895	45,474	205,187	2,456,150	29,925	21,778	113	6,816	180,738	237,399
1896	37,512	187,818	2,051,124	32,801	28,799	120	7,605	196,404	201,305
1897	36,750	217,392	2,004,839	40,702	26,153	51	8,187	184,287	272,493
1898	37,337	198,908	2,115,261	28,037	25,688	83	8,390	198,164	216,326

(*a*) From official reports of the respective countries, except the statistics for the United States. (*b*) A part of the product is reported as plaster of Paris. In converting this to crude gypsum it has been assumed that the loss by calcination is 20%. (*c*) Prussia is a large producer of gypsum, but there are no complete statistics available.

IRON AND STEEL.

THE iron and steel trades during 1899 were pushed to their utmost, and resulted in the greatest production on record, despite the fact that the manufacturers were not always able to obtain raw materials, and while active plants were extended and idle plants reopened, the large demand was not fully met at once. Steel works and foundries, machine and railroad shops were at times inactive from a lack of raw material, and the car shortage which developed in the fall seriously handicapped several manufacturing establishments by the consequent scarcity of coal and coke.

The general prosperity of business enabled the railroads to make renewals and improvements, and the large increase in traffic called for increased motive power and rolling stock as well. During 1899 the rail mills were fully occupied, and had sufficient contracts to keep them busy well into 1900. The locomotive and car builders were well supplied with orders, as exemplified by the Baldwin Locomotive Works in Philadelphia, which produced, during the year, 970 locomotives, of which 358 were exported; some going to England and France. Rails, bridges, cars and other railroad materials were freely ordered from abroad, notably the great Atbara Bridge on the Soudan Railroad in Egypt, the contract for which was taken by the Pencoyd Works, near Philadelphia, against competition with English builders. The Maryland Steel Co. has made large shipments of rails to Yokohama, for the Trans-Siberian Railroad, and to Australia for the Victoria Railroad. The general tendency in the iron and steel trade during 1899 has been to concentrate the industry in very large units and to control either directly or by affiliated interests all stages of the manufacture from the ore and coal to the finished products, and the transportation from mine to furnace. Additional consolidations of iron and steel interests were effected, and large corporations formed in 1898, extended their influence by absorbing smaller companies. Notable in this respect was the extension of the American Steel and Wire Co. Its present capitalization comprises $50,000,000 common, and $40,000,000 preferred 7% stock. Early in the year, 29 sheet metal companies, representing 160 mills, consolidated to form the American Sheet Metal Co., with a capitalization of $54,000,000; one-half of stock is preferred, carrying 7% dividends, and one-half is common. Consolidation in other lines of steel manufacture was effected. The National Tube Co.

was organized with a capitalization of $80,000,000, divided into $40,000,000 7% cumulative preferred stock, and $40,000,000 common stock. The bar mills effected a consolidation of 34 plants, resulting in the Republic Iron and Steel Co., organized with a capital of $19,000,000 common stock, and $14,000,000 preferred, with a 7% guaranteed dividend on the latter. This consolidation comprises seven or eight of the more important hoop mills and cotton tie mills. Attempts were made to reorganize the Carnegie Steel Co., H. C. Frick Coke Co., and various allied interests, and to form one large concern with $100,000,000 bonds, $250,000,000 preferred stock, and $275,000,000 common stock, but the efforts for consolidation did not meet with success.

Southern iron masters have realized the advantage that can be gained by selling their product in the advanced form of steel, rather than in the crude state of pig iron. The Ensley plant, owned by the Alabama Steel and Shipbuilding Co., and controlled by the Tennessee Coal, Iron and Railroad Co., is to be supplemented by rolling mills for plates, structural shapes and other forms of finished basic steel. Other steel works are contemplated in connection with the Birmingham and Sheffield furnaces. The $1,000,000 steel plant of the Alabama Steel and Shipbuilding Co., at Ensley, made its first run of steel on November 30, 1899. When completed there will be ten Wellman open-hearth furnaces, capable of producing 1,000 tons of steel a day. Many new companies have been formed to develop the iron and steel trade, and the prospects for 1900 are very bright.

The advantageous location of Birmingham in respect to freight rates to foreign countries, has claimed the bulk of the export trade of the United States, and pig iron or finished articles were exported to Germany, Great Britain, Italy, Japan, Holland, and other countries.

Iron Ore.—The production and consumption of iron ore in 1899 was enormous. The notable feature of the year has been the extraordinary development of the Lake Superior region. The mines of Michigan, Minnesota and Wisconsin furnished the ores from which were made nearly 80% of the total amount of pig iron produced in the United States.

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.

District.	1897.	1898.	1899.
Lake Superior, shipments to furnaces	11,500,667	14,099,683	18,801,000
Southern States, shipments to furnaces	4,283,700	4,960,000	4,800,000
Other States, shipments to furnaces	1,587,600	1,678,500	2,240,000
Add increase in stocks at Lake Erie docks	995,000	None.	None.
Total mined in United States	18,316,967	20,688,183	25,341,000
Add decrease in stocks at Lake Erie docks	.....	787,348	.....
Add importations	289,970	187,093	674,098
Total	18,606,937	21,662,624	26,015,098
Deduct increase in stocks at Lake Erie docks	995,000	.....	750,000
Deduct exportation	7,583	31,579	40,690
Total consumption	17,804,354	21,631,045	25,224,408

The imports of iron ore in 1899 amounted to 674,098 long tons, against 187,093 in 1898. This ore went to a few furnaces on and near the seaboard.

A large part of it came from Cuba, where the iron mines of the Santiago district were reopened almost immediately after the conclusion of the Spanish war. A few cargoes were received from Spain. A few cargoes were also received from Elba, and one or two from Greece and Algeria. Some Newfoundland ore has been offered in the seaboard markets, and some large contracts for its delivery during 1900 have been made.

Pig Iron.—The production of pig iron in the United States in 1899 and in previous years is given in the subjoined table. In these tables we have omitted the earlier division according to fuel used, which has now but little meaning. The great bulk of the pig iron is made with coke; the so-called anthracite furnaces use a mixture of anthracite coal and coke, and the tendency is continually to diminish the proportion of anthracite. Two or three furnaces only use anthracite alone.

PRODUCTION OF PIG IRON IN THE UNITED STATES. (*g*) (IN LONG TONS.)
(Includes Spiegeleisen and Ferromanganese.)

According to Fuel Used.	1896.	1897.	1898.	1899.
Anthracite	1,146,412	932,777	1,203,273	1,599,552
Coke	7,166,471	8,464,692	10,273,911	11,736,385
Charcoal	310,244	255,211	296,750	284,766
Totals	8,623,127	9,652,680	11,773,934	13,620,703

Kind of Iron.	1897.		1898.		1899.	
	Tons.	%	Tons.	%	Tons.	%
Foundry and forge iron	3,127,010	32·4	3,437,337	29·2	4,213,124	30·9
Bessemer pig	5,795,584	60·0	7,337,384	62·3	8,202,778	60·2
Basic pig	556,391	5·8	785,444	6·7	985,033	7·3
Spiegeleisen and ferromanganese	173,695	1·8	213,769	1·8	219,768	1·6
Totals	9,652,680	100·0	11,773,934	100·0	13,620,703	100·0

States.	1894.	1895.	1896.	1897.	1898.	1899.
Alabama	592,392	854,667	922,170	947,881	1,033,676	1,083,905 *a*
Colorado	73,669	58,508	45,104	6,589	*a* 141,010	138,880
Connecticut	7,416	5,615	10,187	8,336	6,336	10,129
Georgia	40,268	31,084	15,593	17,092	13,762	*b* 17,835
Illinois	604,795	1,006,091	925,239	1,117,239	1,365,898	1,442,012
Kentucky	33,854	63,780	70,660	85,499	100,724	119,019
Maryland	5,600	10,916	79,472	193,702	190,974	234,477
Massachusetts	156	4,710	1,873	3,284	3,661	2,476
Michigan	95,171	91,222	149,511	132,578	147,640	134,443
Minnesota	(*f*)	(*f*)	(*f*)	(*f*)	(*f*)	(*f*)
Missouri	6,522	27,518	12,548	23,863	(*c*)	(*c*)
New Jersey	63,273	55,502	59,153	95,696	101,681	127,596
New York	175,185	181,702	206,075	243,304	228,011	264,846
North Carolina		323	2,151			(*e*)
Ohio	900,029	1,468,789	1,196,326	1,372,889	1,986,358	2,378,212
Oregon	1,000					
Pennsylvania	3,370,152	4,701,163	4,024,166	4,631,634	5,537,832	6,558,878
Tennessee	212,773	248,129	248,338	272,180	263,439	346,166
Texas	4,671	4,682	1,221	6,175	5,178	5,803
Virginia	298,086	346,589	336,277	307,610	283,274	365,491
West Virginia	80,781	141,968	108,569	132,907	192,699	187,858
Wisconsin	91,595	148,400	*d* 158,484	*d* 103,909	*d* 172,781	*d* 203,175
Totals	6,657,388	9,446,308	8,623,127	9,652,680	11,773,934	13,620,703

(*a*) Production of Missouri included. (*b*) Production of North Carolina included. (*c*) Production included in Colorado figures. (*d*) Production of Minnesota included. (*e*) Production included in Georgia figures. (*f*) Production included in Wisconsin figurec. (*g*) Statistics furnished by James M. Swank, of the American Iron and Steel Association.

The increase of 1,846,769 tons, or 15·7% was quite evenly distributed. The larger part of the product was intended for the manufacture of steel, but there was also a considerable demand for foundry and forge iron.

Blast Furnace Output.—On January 1, 1899, there were 200 furnaces in blast

in the United States of a weekly capacity of 243,516 long tons, and a corresponding annual capacity of 12,000,000 tons. On January 1, 1900, there were 280 furnaces in blast, of a weekly capacity of 294,186, and an annual capacity of 15,000,000 tons, showing a net increase of 3,000,000 tons annual capacity during the year. The monthly progress of production during 1899 has been more or less regular through the entire year, and this phenomenal output bids fair to be surpassed in 1900, as many of the large steel companies are balancing their plants by providing ample means for the supply of pig iron. The American Steel and Wire Co. are building six new furnaces; two in the Cleveland district, and four near Pittsburg, Pa. The Carnegie Steel Co. have four new furnaces under way, two at Rankin and two at Duquesne, Pa. The Ohio Steel Co., at Youngstown, and the Illinois Steel Co., at South Chicago, are each adding two new furnaces to their plants; Jones & Laughlin are constructing two furnaces, and contemplate two additional ones in 1900; the National Steel Co. are building three new furnaces, one each at Youngstown, Mingo and Newcastle; the National Tube Co. have one under construction at Brentwood, W. Va.; the Sharon Steel Co., one at Sharon, Pa.; and the Republic Iron and Steel Co. will add one to their Birmingham plant. The aggregate yearly capacity of these modern furnaces, 22 in number, will surpass 3,000,000 tons of pig iron.

Three new furnaces are being constructed in the Eastern district, one at Port Oram, N. J., one for the Thomas Iron Co., and one at Pottstown for the Warwick Iron Co.

PRODUCTION OF STEEL IN THE UNITED STATES. (IN TONS OF 2,240 LB.)

Kinds.	1894.	1895.	1896.	1897.	1898.	1899
Bessemer	3,571,313	4,909,128	3,919,906	5,475,315	6,609,017	7,586,354
Open-hearth	784,936	1,137,183	1,296,700	1,631,843	2,230,292	2,947,316
Crucible	51,702	67,666	62,500	65,600	89,747	126,500
Miscellaneous	4,031	858	900	1,750	3,801	
Total tons	4,412,082	6,114,834	5,582,606	7,174,508	8,932,857	10,662,170
Total metric tons	4,482,592	6,212,671	5,671,928	7,289,300	9,075,788	10,832,765

The production of Bessemer steel in the United States in 1899 was 7,586,354 long tons against 6,609,017 in 1898, showing an increase of 977,337 tons. Of the production in 1899 Pennsylvania made 3,968,779 tons, Ohio, 1,679,237, and Illinois, 1,211,246 tons, these three States furnishing 90·4% of the total amount. Practically the entire Bessemer steel output in 1899 was made by the acid process, although the basic plant at Pottstown, Pa., was started up toward the close of the year. The basic steel produced was made for the most part by the open-hearth process. The production of Bessemer steel rails by the producers of Bessemer steel ingots was 2,240,767 long tons in 1899, against 1,955,427 in 1898, showing a gain of 285,340 tons. Contrasting the production with that of 1898 shows the continued increase in the use of heavier rail sections. Rails of 110 lb. and over have been rolled during the past year, and experiments on the one railroad with 115 lb. rails have proven their advantage. The large increase in the production of light rails is due to the development of mining and lumbering enterprises.

The division of steel rails according to the weights per yard is separately

reported by the American Iron & Steel Association in the following table, the weights per yard of less than 45 lb., between 45 and 85 lb., and above 85 lb., forming the three classes.

	1898.	1899.
	%	%
Heavy rails, over 85 lb. per yd	23·2	25·9
Medium rails, between 45 lb. and 85 lb. per yd	70·7	68·3
Light rails, less than 45 lb. per yd	6·1	5·8
Total	100·0	100·0

Pennsylvania continues to be the chief producer of rails. There were no open-hearth steel rails made during 1899

The subjoined table shows the production of rails as compared with that of Bessemer steel ingots, for the past six years.

Year.	Tons Ingots	Tons Rails.	Per Cent.	Year.	Tons Ingots	Tons Rails.	Per Cent.
1894	3,571,313	904,020	25·3	1897	5,475,315	1,614,399	29·4
1895	4,909,128	1,266,081	25·8	1898	6,609,017	1,955,427	29·6
1896	3,919,906	1,102,892	28·1	1899	7,586,354	a 2,240,767	29·5

(a) Not including rails made from purchased blooms or re-rolled rails.

The 2,947,316 tons of open-hearth steel for 1899 exceeds the production of 2,044,819 tons in 1887, which was the maximum yearly output recorded up to the present year, and consisted of 29·4% acid steel, and 70·6% basic steel; contrasting these amounts with the output for 1898, 70·1% basic and 29·9% acid, shows a slight increase in the production of basic steel.

The production of direct steel castings (included in the table) amounted to 169,729 tons, of which 39,687 tons were basic steel, and 130,040 tons were acid steel; the corresponding figures for 1898 were 110,787 tons of basic steel and 18,460 tons acid steel, showing an increase of production during 1899.

The open-hearth steel produced in 1898 was made by 77 works in 14 States. Pennsylvania leads with a production of 2,393,811 tons.

Summarizing the total steel productions in the United States in 1899 gives Bessemer steel, 71·1%, open-hearth steel, 27·7%, and crucible and special steels, 1·2%.

PRODUCTION OF FINISHED TIN PLATE, (a) FISCAL YEAR ENDING JUNE 30. (IN LBS.)

1891-92.....13,646,719	1893-94.....139,223,467	1896-97.....446,982,063	1897-98.....681,674,028
1892-93.....99,819,202	1894-95.....193,801,073	1895-96.....307,228,621	1898-99.....791,371,486

(a) Statistics furnished by the United States Treasury Department.

Exports of Iron and Steel.—Full details of both domestic and foreign exportation are given in the tables at the end of this volume.

Changes in Iron and Steel Manufacture.—The advancements made during the year 1899 have been chiefly mechanical, and have resulted in a quicker and better handling of material with consequent increased output of furnace and mill. Prof. Henry M. Howe has specially detailed the progress in the metallurgy of iron and steel in his review, to which reference should be made further on in this caption.

The Iron Market in 1899.

There was a continuous rise of prices of all descriptions of iron and steel during 1899, which was particularly marked in the second half of the year. The prices covered a range greater than has ever before been shown in a single year. All grades of pig irons and varieties of finished steel sold in December for more than double the prices quoted in January. The result of these high prices had but little effect on the production of 1899, as a large part of the output was delivered under contracts made early in the year, when prices were comparatively low. The manufacturers have secured enormous profits, considerably in excess of the returns for many years past.

Birmingham, Ala.—While the production of pig iron during 1899 was but little in excess of that in 1898 the industry has been greatly benefited by the high prices during the past year. The accumulation of stocks due to the depressed prices of recent years has been cleared off, and at the present day there is hardly a week's supply on hand. The outlook for 1900 is very encouraging.

The following figures contrast the prices of January, 1899, with the closing prices of December, 1899:

	December, 1899.	January, 1899.		December, 1899.	January, 1899.
No. 1 Foundry.....	$18·50 to $19·00	$8·00	Gray Forge	$16·00 to $16·25	$6·50 to $6·75
No. 2 Foundry.....	17·75 to 18·50	7·75 to $8·00	No. 1, soft.....	18·50 to 19·00	8·00
No. 3 Foundry.....	16·75 to 17·50	7·50 to $7·75	No. 2 soft.....	17·75 to 18·50	7·75 to $8·00

Chicago, Ill.—The demand for iron during 1899 was the greatest ever experienced, and in the early months every line of industry had sold its capacity for the entire year. By July practically no contracts were taken for immediate delivery, all being dated three, six, nine, and even twelve months ahead. The year opened with a brisk demand for pig iron which increased so rapidly that the prices rose on an average of $1 per month. The demand for iron could not be met by the local furnaces, and other centers were drawn upon in the latter part of the year. With all the great prosperity of the iron trade, foundries and other consumers had great difficulty in obtaining supplies of pig iron during the latter half of the year, and were seriously handicapped by the lack of raw materials, the car shortage that occurred in the autumn leading up to a famine in coal and coke, which further handicapped the manufacturers; yet withal, the year may be considered a record year, although prices did not reach the high position they occupied in 1879-1880.

The progress of the increased price of pig iron is set forth in the subjoined table:

MONTHLY AVERAGE PRICES OF PIG IRON IN CHICAGO.

Month.	Northern No. 2.	Southern No. 2.	Month.	Northern No. 2.	Southern No. 2.
January.....	$11·50	$11·25	August.....	$20·25	$19·75
February.....	12·25	12·00	September.....	22·25	21·75
March.....	14·00	13·75	October.....	23·50	23·50
April.....	15·25	15·00	November.....	23·50	22·50
May.....	16·25	16·00	December.....	23·50	22·50
June.....	17·50	17·00	Av. 1899.....	18·25	17·70
July.....	19·25	18·75	Av. 1898.....	10·90	10·46

The steel rail market showed a greater conservativeness than other branches; a heavy business was done in rails less than 45 lb. per yd., due to the great activity in the mining and lumber regions. The following average monthly prices per ton (2,240 lb.) were quoted for large lots:

January	$20·00	May	$26·00	September	$32·75
February	22·75	June	27·50	October	33·22
March	23·75	July	30·00	November	35·00
April	25·75	August	31·00	December	35·00

Average price for 1899, $28·50. Average price for 1898, $19·00.

The fluctuation in the prices of billets during 1899 was as follows (monthly average price per ton):

January	$18·25	May	$28·00	September	$41·00
February	20·00	June	33·00	October	40·00
March	24·25	July	38·00	November	39·00
April	25·50	August	38·00	December	37·00

Average price for 1899, $31·85.

Cleveland, Ohio.—More business was done in the season of navigation just brought to a close than in any previous year. The total ore shipments from the Lake Superior region are detailed in the following table:

IRON ORE SHIPMENTS FROM UPPER LAKE PORTS.

Ports.	1897.	1898.	1899.	Ports.	1897.	1898.	1899.
Escanaba	2,302,121	2,803,513	3,720,218	Duluth	2,376,064	2,635,462	3,509,965
Marquette	1,945,519	2,245,965	2,733,596				
Ashland	2,067,637	2,391,088	2,709,447	Total by Lake	12,215,645	13,655,433	17,901,358
Two Harbors	2,661,466	2,693,246	3,973,733	Rail shipments	253,993	374,250	400,000
Gladstone	341,014	335,955	381,457				
Superior	531,825	550,403	878,942	Total shipments	12,469,638	14,029,683	18,301,358

SHIPMENTS OF LAKE SUPERIOR IRON ORE BY RANGES FOR FIVE YEARS.

Year.	Marquette.	Menominee	Gogebic.	Vermilion.	Mesabi.	Totals.
1895	2,097,838	1,923,798	2,547,976	1,077 838	2,781,587	10,429,037
1896	2,603,889	1,560,467	1,799,971	1,088,090	2,882,079	9,934,446
1897	2,715.035	1,937,913	2,258,236	1.278,481	4,280,873	12,469,638
1898	3,195,039	2,527,274	2,498,462	1,265,142	4,613,766	14,029,683
1899	3,757.010	3,301,052	2,795,856	1,771,502	6,626,384	18,251,804
Totals from opening of mines	56,205,271	30,759,767	28,341,341	13,585,360	23,596,606	152,369,667

IRON ORE RECEIPTS AND STOCKS AT LAKE ERIE PORTS.

Ports.	Receipts at Ports.				Stock on Docks Dec. 1.			
	1896.	1897.	1898.	1899.	1896.	1897.	1898.	1899.
Toledo	301,794	416,438	414,012	792,348	151,959	194,644	146,568	186,422
Sandusky	58,667	79,792	136,200	87,499	59,491	84,786	48,500	23,184
Huron	226,515	198,231	126,755	263,600	200,075	230,029	139.982	164,480
Lorain	191.445	355,188	536,086	1,112.946	231,288	317,509	324,034	337,822
Cleveland	2,313,170	2,456.704	2,645,318	3,223,582	1.419,311	1,478,355	1,175,970	1,200,806
Fairport	941,446	1,008,340	912,879	1,241,013	773,905	895,312	719.794	692,147
Ashtabula	2,272,892	3,001,914	2,684,563	3,341,526	1,441,666	1,835,694	1,732,671	1.902.598
Conneaut	327,623	495,327	1,404,169	2,320,696	275,800	360,895	288,101	468,808
Erie	847,849	1,311,526	1,092,364	1,309,961	355,222	484,871	439,167	361,335
Buffalo, Tonawanda	545,101	797,446	1,075,975	1,530,016	82,267	111,660	121,630	192,681
Totals	8,026,432	10,120,906	11,028,321	15,222,187	4,954,984	5,923,755	5,136,407	5,530,283

Pittsburg, Pa.—The remarkable iron trade of 1898, both in production and consumption of iron and steel, was eclipsed by the phenomenal record for 1899. Pittsburg maintained her supremacy and general conditions of large production, and high prices prevailed in all branches of the trade. Prices of all raw and finished materials more than doubled during 1899, and remained firm throughout with the exception of sheets, which declined in the latter part of the year. Bar iron made a greater proportionate gain than probably any other line. The price was around 1c. in the early months, and rose as high as 2·75c. before the close of the year.

Bessemer pig iron sold in January at $10·75, and it was not until late in February that prices began to advance. In May the price was $16·50, and $1 higher was quoted on June 1st. On July 1st, Bessemer pig iron was selling at $20, and $2 was soon added to the price. In September the rate ran from $22·25 to $23·50, and October opened with pig iron at $24. There was little left to sell for the balance of the year and several lots were disposed of at $25 and higher. Foundry irons kept pace with Bessemer. No. 2, which sold in January at $10·25, advanced to $23·50 before December 1st. Proportionate advances were made in other grades.

Structural material experienced a decided boom. Prices have advanced about 75% since the beginning of the year, when beams and channels from 3 to 15-in. were quoted at 1·30c.; 18, 20 and 24-in. at 1·40c.; angles, 1·15c.; zees, 1·25c.; and tees, 1·30c. The average prices ruling during the last quarter were: Beams and channels, from 3 to 15-in., 2·25c.; 18, 20 and 24-in., 2·35c.; angles, 2·25c.; zees, 2·25c.; tees, 2·30c. Prices for steel plates more than doubled during the year. On January 1st, tank plate was quoted at 1·25c. and flanges at 1·35c. During December, tank was quoted at 2·75c. and flange at 2·85c.

The price of steel billets more than doubled during the year, opening at $16·25 and closing at $37, and during the past three months $40, and higher, was paid. While these high prices prevailed, steel rail manufacturers were filling contracts for rails taken at $16·50 a ton. Rails were advanced to $37 for new business by an agreement of the makers, but a heavy joint order for delivery in 1900 was taken at $33.

The sheet market opened and closed the year in a very unsatisfactory condition. On January 1st, No. 27 gauge was quoted at the abnormally low figure of 1·85c., No. 28 at 1·90c. and galvanized at 80 and 10% off. During December No. 27 was quoted at 2·90c., No. 28 at 2·95@3·05c., and galvanized at 75% off. The sheet market improved in February, when prices advanced $2 a ton, and steadily increased until October 1st, during which month No. 27 averaged 3·20c., No. 28, 3·30c. and galvanized 70 and 10% off. A slump occurred on November 1st, when the options on the sheet mills of the country for the proposed consolidation expired and the deal was declared off.

The advance in wire nails eclipsed all records. The price at the opening of the year (in car load lots) was $1·35, and cut nails at $1·10, but before the close of the third quarter the former was quoted at $2·85 and the latter at $2·65. These rates were maintained during the balance of the year.

DECEMBER PRICES FOR IRON AT PITTSBURG IN SIX YEARS.

	1894.	1895.	1896.	1897.	1898.	1899.
Bessemer	$10·25	$12·75	$11·15	$10·25	$10·80	$25·00
No. 1 Foundry	11·40	14·25	12·10	10·00	11·10	24·00
No. 2 Foundry	10·40	13·50	11·25	10·75	10·75	23·25
Mill Iron	9·25	11·75	9·80	10·25	10·65	21·50
White Iron	9·00	9·00	9·10	9·00	9·25	20·00
Mottled Iron	9·00	9·00	9·10	9·25	9·30	20·50
Silvery, No. 1	13·60	14·80	14·50	13·00	13·00	27·00
Charcoal, No. 1 Foundry	17·50	17·80	17·50	15·50	16·00	27·00
No. 2 Foundry	16·25	17·00	16·00	15·25	15·50	26·00
Cold Blast	23·50	23·00	23·25	21·50	21·50	28·00
Warm Blast	16·50	16·00	16·10	15·00	15·25	27·00
Muck Bar	18·40	21·00	19·75	18·75	18·85	33·00
Billets	15·15	16·50	16·00	15·50	16·50	37·00
Billet Ends	10·45	14·00	13·00	10·00	11·00	24·00
No. 1 Scrap	16·00	14·00	12·75	11·00	12·00	24·00
Steel Rails	32·00	28·00	25·00	18·50	17·50	37·00
Bar Iron	1·15	1·25	1·30	1·15	1·18	2·15
Iron Nails	1·00	1·25	1·60	1·30	1·30	2·50
Steel Nails	1·00	1·30	1·60	1·30	1·25	2·60
Wire Nails	1·00	1·30	1·55	1·45	1·50	2·95
Coke at Ovens	1·00	2·00	2·00	1·75	1·50	3·00
Ferromanganese, 80 per cent	49·50	54·00	48·25	47·00	50·00	100·00
Old Iron Rails	12·50	15·00	13·00	14·00	14·00	30·00
Old Steel Rails	10·00	13·00	12·75	10·50	10·00	23·00

PRODUCTION OF PIG IRON AND STEEL IN THE WORLD.

The total production of pig iron in 1899 in the principal countries of the world shows an increase of 3,757,531 metric tons, or 10% over the previous year, while for steel the increase was 2,755,554 tons, or 11·3%. The proportion of pig iron converted into steel was somewhat less than in 1898. A comparison of the relative production of pig iron and steel in the past five years shows that the pig iron product is being more and more converted into steel, a condition due to the superiority of steel as a structural material, and to the continued cheapening of its manufacture. In other lines also steel is replacing cast iron, more particularly in the construction of large engines, locomotives and cars. The continued industrial activity is naturally reflected in the iron and steel trades, and even the enormous production in 1899 will doubtless be further increased in 1900. The statistical diagram which is given later in this caption shows very clearly the relative positions occupied by the chief producers of iron and steel during the past 25 years. Of the world's production of pig iron in 1899 the United States made 37·7%, the United Kingdom 25·7%, and Germany 21·1%, a total of 84·5% of the entire output for the year. The steel production and percentages in 1899 were: United States 44·1%, the United Kingdom 20·2%, and Germany 25·8%, a total of 90·1% of the world's supply. During the past decade the United States and Germany have greatly increased their production of steel while the United Kingdom, occupying the third position on the list of producers, has not materially advanced. Assuming an average yield of 50% of pig iron from the ore charged into the furnace the total production of pig iron involves the mining and handling of at least 80,000,000 tons of iron ore, exceeding the corresponding figure for 1898 by 10,000,000 tons.

Belgium.—The total production of pig iron in Belgium in 1899 was 1,036,185 metric tons against 982,748 in 1898, showing an increase of 53,437 tons, or 5·4%. This includes foundry, forge and steel pig. The total production of

PRODUCTION OF PIG IRON AND STEEL IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	Austria-Hungary		Belgium.		Canada.	France.		Germany.	
	Pig Iron.	Steel.	Pig Iron.	Steel.	Pig Iron.	Pig Iron	Steel.	Pig Iron	Steel.
1895	1,075,000	*a*495,000	929,135	455,550	47,586	2,005,889	899,676	5,788,798	*a*3,941,300
1896	1,130,000	*a*520,000	982,780	598,755	47,635	2,333,702	1,159,970	6,360,962	4,297,447
1897	1,205,000	553,000	1,024,666	616,604	49,497	2,472,143	1,281,595	6,888,087	5,091,294
1898	1,250,000	605,500	962,748	653,130	69,248	2,525,075	1,438,717	7,402,717	5,734,307
1899	*a*1,300,000	*a*660,000	1,036,185	729,820	*b*92,869	2,567,388	1,529,182	8,029,305	6,290,434

Year.	Italy.		Russia.		Spain.		Sweden.	
	Pig Iron.	Steel	Pig Iron	Steel.	Pig Iron.	Steel.	Pig Iron.	Steel.
1895	*a*10,500	50,314	1,454,298	574,112	206,430	65,000	455,200	281,900
1896	11,280	59,500	1,629,810	*a*625,000	246,326	104,577	466,400	250,600
1897	12,500	57,250	1,857,000	821,000	282,171	121,100	533,800	268,300
1898	12,950	58,750	2,228,850	1,095,000	261,799	112,605	492,700	263,700
1899	*a* 13,250	*a* 60,250	*a* 2,600,000	*a* 1,400,000	295,840	122,954	534,000	258,500

Year.	United Kingdom.		United States.		All Other Countries.		Totals.	
	Pig Iron.	Steel.	Pig Iron.	Steel.	Pig Iron	Steel.	Pig Iron.	Steel.
1895	8,022,006	3,444,201	9,597,449	6,212,671	*a*375,000	*a*275,000	29,887,591	16,659,005
1896	8,700,220	4,306,211	8,761,197	5,366,518	395,000	290,000	31,015,302	17,581,131
1897	8,930,086	4,559,786	9,807,123	7,289,300	450,000	310,000	33,513,073	20,979,179
1898	8,819,968	4,639,042	11,962,317	9,075,783	545,000	355,000	36,658,272	24,365,502
1899	9,454,204	4,988,010	13,898,634	*c*10,736,806	*a*650,000	*a*400,000	40,401,675	27,121,056

(*a*) Estimated. (*b*) Includes an estimate of 5,897 metric tons for the Province of Quebec. (*c*) Included in the total is an estimate of 130,556 metric tons for crucible steel.

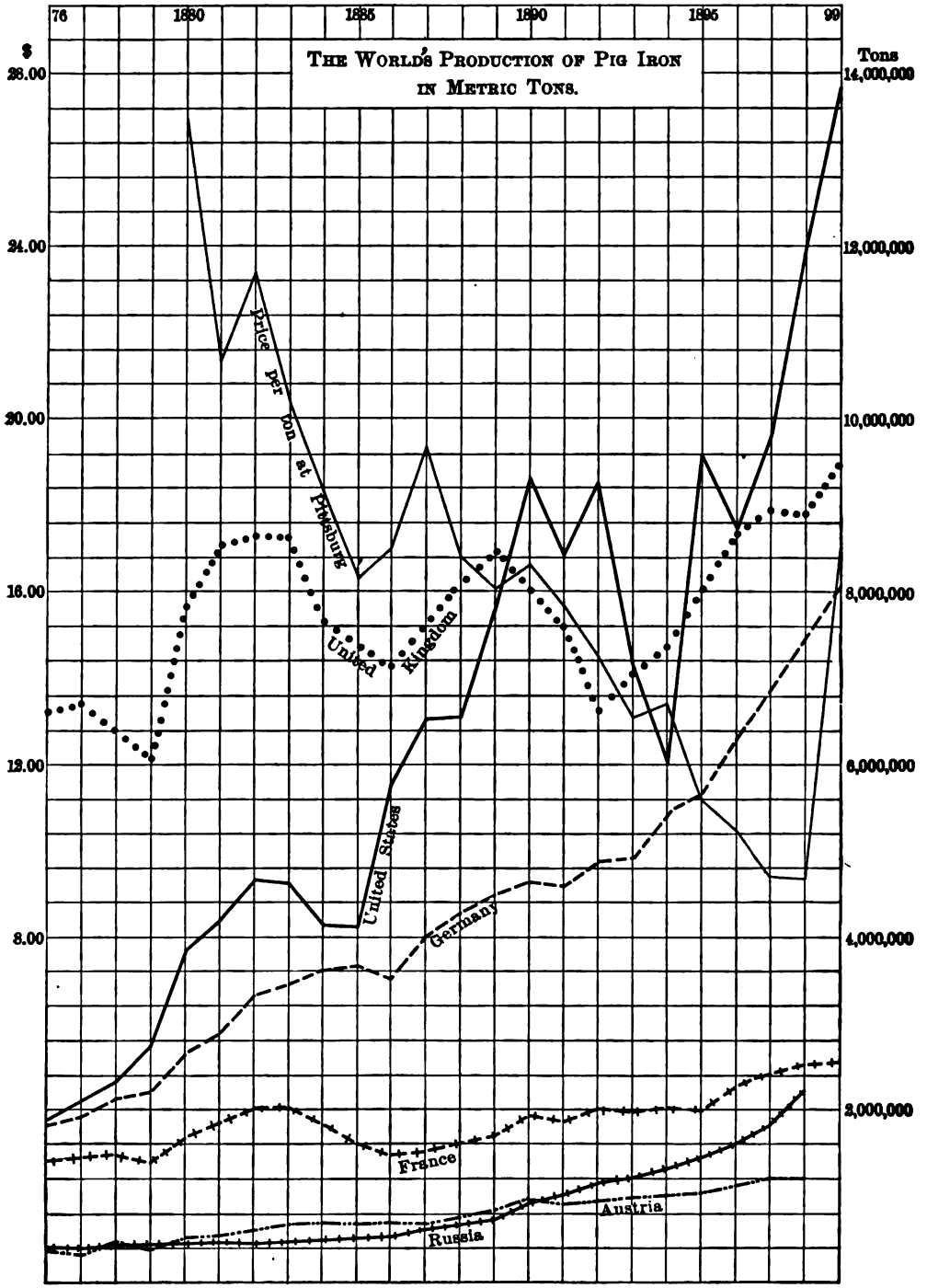
steel in 1899 was 729,820 tons, against 653,130 in 1898, an increase of 76,790 tons, or 10·2% over 1898.

France.—The pig iron production in 1899 was 2,567,388 metric tons, an increase of 42,313 tons, or 1·7% over the production in 1898. The production of steel, including both converter and open-hearth steel ingots, was 1,529,182 tons, an increase of 95,465 tons, or 6·5% over 1898. The output of finished steel was 1,253,701 tons against 1,138,639 in 1898, an increase of 115,062, or 10%. The production of wrought or puddled iron, including all forms, was 852,755 tons, an increase of 41,212 tons, or 5% over the production in 1898.

Germany.—The production of pig iron in Germany, including Luxemburg, is reported by the Association of German Iron and Steel Makers in the following table:

Description.	1897.		1898.		1899.		Changes.
	Metric Tons.	%	Metric Tons.	%	Metric Tons.	%	Metric Tons.
Foundry iron	1,126,408	16·4	1,301,768	17·6	1,424,732	17·7	Inc. 122,964
Forge iron	1,619,556	23·5	1,564,149	21·1	1,663,571	20·7	Inc. 99,422
Bessemer pig	577,947	8·4	534,674	7·2	516,950	6·5	Dec. 17,724
Thomas (basic) pig	3,565,156	51·7	4,002,126	54·1	4,424,052	55·1	Inc. 421,926
Totals	6,889,067	100·0	7,402,717	100·0	8,029,305	100·0	Inc. 626,588

The total increase in 1899 of 8·5% over the product in 1898 shows the steady growth of the industry. During the past seven years there has been no retro-


76
1880
1885
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99
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Tons
THE WORLD'S PRODUCTION OF PIG IRON IN METRIC TONS.
28.00
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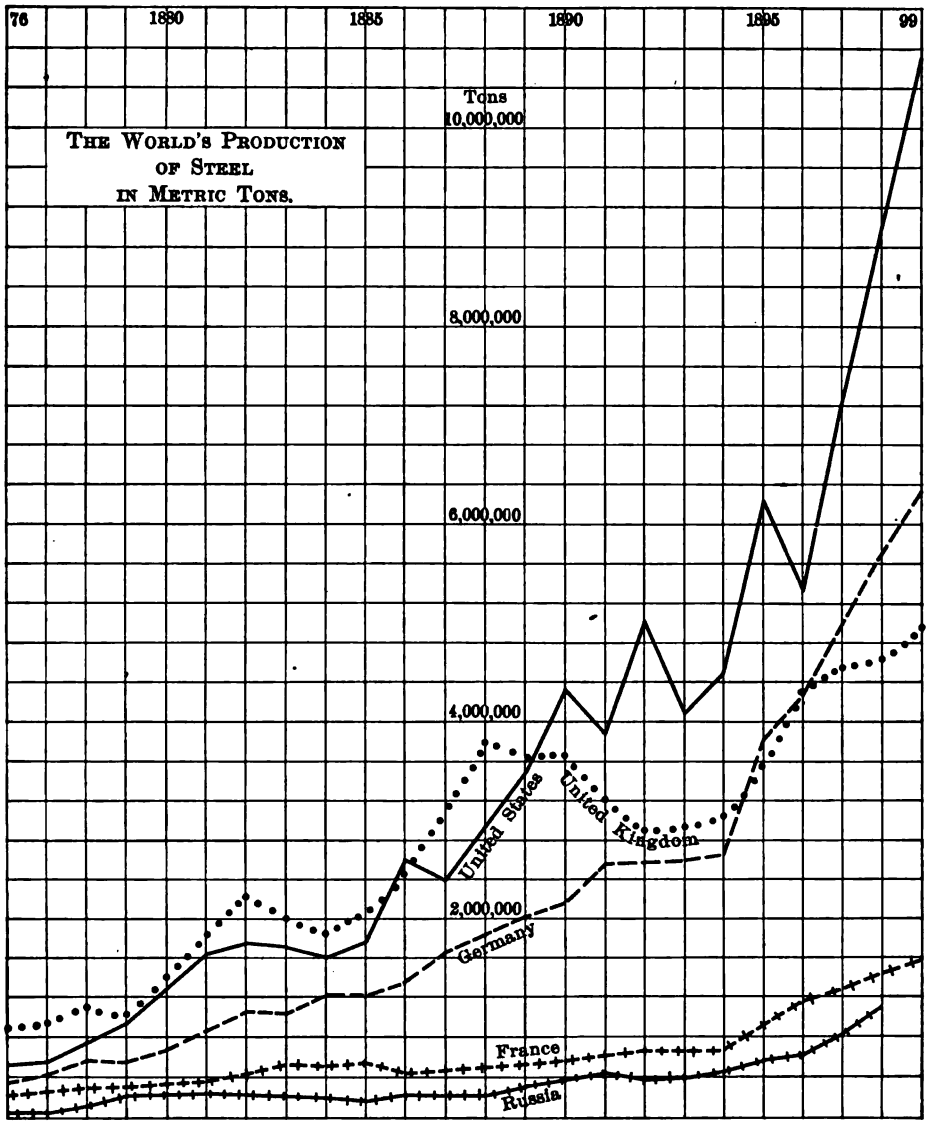
Price Per Ton at Pittsburg
United Kingdom
United States
Germany
France
Austria
Russia



grade movement; each year showing a gain over the preceding one. There were no important changes in the divisions of the output, although it is worthy of note that there was an increase of 10·5% in the production of basic pig, and an increase of 3·3% in the Bessemer pig output. As in previous years, the greatest proportionate increase was in the production of basic pig, which is a natural result of the high phosphorous content of nine-tenths and more of the native ores.

The production of steel in Germany, as reported by the Association of German Iron and Steel Makers, is given in the subjoined table:

	1897.		1898.		1899.	
	Metric Tons.	%	Metric Tons.	%	Metric Tons.	%
Basic converter	2,234,214	63·5	3,606,737	62·9	3,978,225	63·2
Basic open-hearth	1,304,423	25·6	1,459,159	25·4	1,608,825	26·9
Total basic	4,538,637	89·1	5,065,896	88·3	5,667,050	90·1
Bessemer	552,657	10·9	668,411	11·7	623,384	9·9
Totals	5,091,294	100·	5,734,307	100·	6,290,434	100·

The increase in the production of total basic steel was 601,154 metric tons, corresponding to 11·9% over the output in the previous year; the individual progress was most marked in the basic open-hearth process, which showed an increase of 16·1%, while that of the basic converter process was 10·2%. The acid Bessemer process yielded 623,384 metric tons against 668,411 in 1898, showing a decrease of 45,027 tons, or 6·7%. Contrasting the conditions of pig iron and steel production for the past three years in Germany and in the United Kingdom, shows that while the production of pig iron has been less than in the former country the steel output has been greater. This was due to the increasing tendency in Germany to convert the pig iron output into steel and the continued use of wrought iron in the United Kingdom.

Russia.—The statistics for the production of iron and steel in Russia in 1899 are not yet available. The output in 1898 was 2,228,850 metric tons of pig iron and 1,095,000 tons of steel. The production in 1899 will doubtless exceed these figures.

Spain.—The total production of iron ore in 1899, reported by the *Revista Minera,* was 9,234,302 metric tons, against 7,197,047 tons in 1898; an increase of 2,037,255 tons, or 28·3%. Two-thirds of this production—6,146,542 tons—was from the mines in the Province of Vizcaya, while 1,285,440 tons were mined in Santander. The domestic consumption of iron ore was 621,165 tons, or 6·7% of the total ore mined, the remainder, 93·3%, being exported. The products of iron and steel are given in the following table:

	1897.	1898.	1899.	Changes.
Pig iron	282,171	262,497	295,840	Inc. 33,343
Wrought, or puddled iron	.....	.....	66,568	.....
Bessemer steel ingots	68,500	54,500	68,300	Dec. 13,800
Open-hearth ingots	52,600	58,105	54,654	Dec. 3,451
Finished iron and steel	163,500	154,910	173,569	Inc. 18,656

Of the pig iron production but a small proportion, 19,000 tons, was made with charcoal, the principal fuel being coke. The exports of pig iron in 1899

Destination.	1897.	1898.	1899.	Changes.
Great Britain	5,091,057	4,748,557	6,224,229	Inc. 1,475,672
Germany	1,058,694	1,193,224	1,544,449	Inc. 350,525
France	435,972	399,424	443,818	Inc. 44,394
Belgium	224,776	201,693	254,860	Inc. 53,167
United States	59,243	5,792	132,422	Inc. 126,630
Other countries	14,876	8,670	13,359	Inc. 4,689
Totals	6,884,588	6,558,060	8,613,137	Inc. 2,055,077

were 40,919 tons, a decrease of 5,186 tons from 1898. The amount in metric tons and the destination of the exports of iron ore for the past three years are given in the preceding table.

Sweden.—The production of pig iron in 1899 was 524,000 metric tons, an increase of 31,300 tons, or 6·3% above the production in 1898. The Bessemer and steel ingots produced were 102,200 tons, being 10,400 tons, or 11·3% greater than 1898. There was a decrease of 9%, or 15,600 tons in the amount of open-hearth steel ingots produced, the figures for 1899 and 1898 being respectively 156,300 and 171,900 tons. The production of blooms and billets was 198,900 tons, an increase of 3,600 tons, or 1·8% above the production in 1898. The most marked change in the iron and steel industry of this country is the gradual replacement of the open-hearth product by the cheaper Bessemer steel.

United Kingdom.—The production of pig iron in 1899, as reported by the British Iron and Steel Trade Association, was 9,454,204 metric tons (9,305,-319 long tons), against 8,819,968 metric tons in 1898, a gain of 7·2%. The 1899 output was divided into forge and foundry iron, 4,258,956 long tons (45·8%); Bessemer pig, 3,990,282 long tons (42·9%); basic pig, 860,672 long tons (9·2%); spiegeleisen and ferromanganese, 195,409 long tons (2·1%). The gain was entirely in steel pig, the foundry and forge iron showing a decrease of 3·4%. In 1898 steel pig of various classes constituted 49·2% of the total make, while in 1899 they formed 54·2%. This is a much lower proportion than in the United States, where in 1899 the proportion of steel pig was 69·1% of the total; or Germany, where it was 61·6%. The lower proportion in the United Kingdom is due to the greater adherence of the iron makers to wrought or puddled iron. The total stocks on hand at the end of the year was 720,717 long tons, showing a reduction of 190,000 tons during the year. The exports in 1899 were 1,379,296 tons, while 171,416 tons were imported, giving a net export of 1,207,880 tons. The approximate consumption of pig iron in the United Kingdom in 1899 was 8,287,439 long tons. The production of steel in the United Kingdom is detailed in the following table:

	1897.		1898.		1899.		Changes.
Open-hearth:	Long Tons.	Per cent.	Long Tons.	Per cent.	Long Tons.	Per cent.	Tons.
Acid steel	2,393,718	92·0	2,590,512	92·3	2,785,563	90·3	Inc. 145,051
Basic steel	208,088	8·0	216,088	7·7	294,688	9·7	Inc. 78 600
Total open-hearth	2,601,806	100.0	2,806,600	100.0	3,080,251	100·0	Inc. 223,651
Bessemer:							
Acid	1,374,339	73·0	1,255,252	71·3	1,307,696	71·6	Inc. 52.444
Basic	509,816	27·0	504,134	28·7	517,378	28·4	Inc. 13,244
Total Bessemer	1,884,155	100·0	1,759,386	100·0	1,825,074	100·0	Inc. 65.688
Total steel	4,485,961	.....	4,565,986	.....	4,855,325	.....	.....

The production of steel in the United Kingdom in 1899 was 4,855,325 long tons, an increase of 279,339 long tons, or 6·1% over 1898. Of this total 3,030,-251 tons (62·4%) were open-hearth, and 1,825,074 (37·6%) tons were Bessemer. Nearly two-thirds of the total output of steel was made by the acid processes, of which the open-hearth process continued to yield the greater part of the open-hearth steel output. The increase in the production of basic steel

is very slow, and is mainly confined to the basic open-hearth process. Of the total steel produced in 1899, 4,043,259 long tons, or 83·3% were acid, and 812,066 tons, or 16·7% were basic steel. During 1899, 65 of the 76 converters were in operation, 24 making basic and 41 making acid steel. The average output per converter for the year was 28,078 tons. Of the 438 open-hearth furnaces the average number at work was 381, giving an average output for the year of 7,953 tons. The Bessemer steel output is largely used for rails, 879,155 tons being consumed for this purpose, while the open-hearth steel has the preference for other uses. Wrought iron still continues to be used in the United Kingdom to a much greater extent than either the United States or Germany. The total make of wrought or puddled iron in 1899 was 1,201,606 long tons, an increase of 85,907 tons over 1898. There was a decrease in the production of both bar iron and plates, amounting to 31,610 and 28,280 tons respectively. Of the 1,320 puddling furnaces in the United Kingdom the average number that worked during the year was 1,149, showing but 171 idle, a very small proportion.

Notes on the Progress of Iron and Steel Metallurgy in 1899.

By Henry M. Howe.*

1. *The Blast Furnace.*—The important advance in blast furnace practice in 1899 was the further development in the use of blast-furnace gas-engines, including the use of gas for driving the blowing engines of the blast furnace itself. As the use of electric motors for driving rolling mills promises to be successful, there is probably at hand ample use for all the gas which a blast furnace can make.

In the use of multiple tuyeres, blast furnace men appear to have generally settled down to the use of about 12 tuyeres, where eight were formerly used. The advantage of a large number of tuyeres is that the conditions are more uniform throughout a given horizontal section of the hearth. But of course every additional tuyere means additional expense for maintenance; and the general opinion appears to be that the extra cost of additional tuyeres over and above the number of 12 is not compensated for by further increase in regularity of work.

In the use of fine Mesabi ore, while it has been found possible to work with a much larger proportion, yet the usual practice appears to be to use not more than 33%.

No diminution in the great outputs from blast furnaces has occurred or is looked for. The two things which have been necessary to do in order to succeed in making these great outputs are: (1) to provide the furnaces with very large crucibles, since we cannot expect to burn more than a certain quantity of coke per square foot of hearth per second; and (2) to provide powerful engines so as to secure penetration and to overcome incipient scaffolds, as well as to provide the enormous quantity of blast required by these great outputs.

The use of these powerful engines fits in very well with the use of fine Mesabi ore, in that it tends to provide for the very difficulties which the use of this ore

*For the sake of uniformity with other parts of the book symbols, such as Si, Mn, etc., are used in this article instead of writing in full the names silicon, manganese, etc., which would be in conformity with Prof. Howe's usual practice.—Editor Mineral Industry.

occasions, scaffolds. A powerful blast is an efficient weapon with which to attack incipient scaffolds.

2. *Talbot's Continuous Open-Hearth Process.*—What promises to be the most important, and is certainly one of the most interesting modifications of the open-hearth process which has ever been proposed, is the continuous procedure, devised and developed by Mr. Benjamin Talbot of the Pencoyd iron works.

The common procedure in the open-hearth process is to treat each charge as a distinct entity, charging it as a whole, bringing it as a whole to the proper final temperature and composition, and then tapping and casting it as a whole. This we may call the intermittent process. Mr. Talbot's process consists essentially in working the furnace quasi-continuously instead of intermittently, tapping at short intervals say one-fourth of the charge, immediately charging an equivalent quantity of pig iron to be treated, and again tapping another fourth. Thus the furnace is worked like the tank of a glass furnace; throughout the week it is full or nearly full of metal, and every four hours about one-fourth of the charge is replaced by an equivalent fresh charge, which loses its individuality at once in the residual bath of metal into which it is introduced.

At first one is a little incredulous about the advantages of this system. The work of the open-hearth process is essentially to melt pig iron with either scrap as a diluent, or iron ore as an agent to oxidize its carbon, silicon and phosphorus; and further to oxidize those elements by additions of iron oxide and by exposure to the oxidizing atmosphere of the furnace.

One at first hardly sees why this operation should go on faster or better in continuous than in intermittent work. Indeed, the reasons for the merits of the process do not lie on the surface, and they are of a kind not easily made clear, except to those pretty familiar with the open-hearth process.

Perhaps a simile may make the rationale of the process easier to grasp. We may distinguish two extreme types of feeding the human body. One is that which as boys, we were taught that the American Indians followed, *viz.:* the intermittent way, in which the stomach was gorged at considerable intervals, and received no further food perhaps for days, until the body had been practically drained of the products of the gorging meal. The other is the quasi-continuous method of feeding which we commonly pursue, *viz.:* to eat in moderate quantities three times a day, the food at each meal becoming mixed up in the system with that of the preceding meals. In the case of invalids we go still further and feed every two hours. It is needless to say that the long interval way of feeding is very much more trying to the human digestion than the intermittent or civilized method; and that the bi-hourly method used for invalids, while inconvenient, throws even less burden on the digestive organs. The gorging of the long interval Indian mode of feeding chokes the body with the intermediate products of digestion. In the continuous method each organ has a moderate amount of work to do at all times, and this work it discharges more easily and better.

Now very much the same thing seems to be true of the open-hearth process. If we introduce the charge as a whole into the furnace, that furnace is strained by the melting down of so great a charge, and it is a long time before we reach the

conditions of temperature and composition of the bath favorable to the most rapid removal of carbon and phosphorus. In Mr. Talbot's continuous process the temperature and the slag and bath composition vary only very slightly from period to period, so that after the introduction of any given charge, we very quickly reach and easily maintain the conditions most favorable to the rapid oxidation and removal of carbon and phosphorus.

Perhaps all this may be made clearer by a description of the process itself.

The Furnace.—The furnace which Mr. Talbot now has at Pencoyd has a capacity of about 75 tons; it is a Wellman tipping furnace, 30 ft. between the neck blocks and 9 ft. wide inside, with a bath about 25 ft. long, 8 ft. wide, and 2 ft. 6 in. deep at its deepest point. Its regenerators are about 22 ft. long and 6 ft. high; the air regenerator is about 10 ft., and the gas about 7 ft. wide. In addition to these there are slag pockets about 7 ft. wide, 14 ft. long and 8 ft. high. The general procedure is the pig and ore basic open-hearth process. No scrap is used except in making up the initial bath.

Procedure.—The first thing to do is to prepare the initial bath, which has to be done in pretty much the common way. At the beginning of the week 70,000 lb. of pig iron and 30,000 lb. of scrap iron are melted down and worked till the carbon is reduced to about 0·10%. Now begins the regular working of the process. The slag at this time will contain about 12% SiO_2 and 12% Fe. The next thing is to enrich this slag by adding about 3,600 lb. of mill scale, which raises the iron in the slag to from 25 to 30%. Then from 25,000 to 30,000 lb. of molten cast-iron are poured into the furnace, reacting with a beautiful pyrotechnic effect on the iron oxide of the slag, and doubtless on that in the metal. The thin and fluid layer of slag foams and leaps up to a height of 9 or 10 in., and great flames of carbonic oxide shoot up through it. This violent reaction removes the carbon and phosphorus rapidly from the metal and also reduces the iron oxide from the slag, which is now so nearly free from iron that much of it is drawn off, partly by tilting the furnace slightly and partly by adding a little limestone, which, by causing a slight boiling by the escape of its carbonic acid, makes the slag froth up and so raises its upper surface, so that it may run out of the slag notch.

To restore iron oxide to the slag and thus give it further oxidizing power, about 2,000 lb. of mill cinder (containing about 8% SiO_2 and 66% Fe), and 1,800 lb. of limestone are introduced, and then a second ladle of molten iron—about 10,000 lb.—from 15 to 30 minutes after the first ladle. A very considerable reaction now occurs, but not so much as after the first ladleful was charged.

The bath is then gradually boiled down until it contains about 0·07% C, trace Si, 0·17% Mn, 0·035% P and 0·025% S. The slag at this time will contain about 14% Fe and 12% SiO_2. Then shortly before tapping, the last addition of pig iron, about 3,000 lb., is made. This raises the carbon to about 0·10 or 0·11%, and the phosphorus to 0·04%. The slag will now contain about 12% SiO_2 and 12% Fe, and for the second time some of it is withdrawn.

The melter now assures himself that his temperature is right by the common test of moving an iron rod through the bath and noting how rapidly it is melted away, that the carbon is right by the fracture of a test ingot, and that the phos-

phorus is low enough by a rapid analytical determination, and taps about one-fourth of the charge, say 40,000 lb. of metal, out into a ladle, in which it receives all the final additions, and is at once cast into ingots or other castings. After tapping there still remains in the furnace some 55 tons of molten metallic bath.

In tapping, the furnace is tilted so as to draw the metal from only a little below the slag layer, so as to get the hottest metal. Immediately after tapping the furnace is tipped back so that the tap-hole emerges completely above the slag level, and it is then made up at leisure. At the same time the slag line is fettled by shovelling ground dolomite mixed with resin (Talbot's process) through the charging doors. The slag which was present at the time of tapping is not removed, and as soon as the tap-hole is made up and the fettling completed, the cycle begins again with the addition of 3,000 lb. of scale to enrich the slag so that it may react upon the following charge of pig iron.

Advantages.—(1) First and foremost, we have here not only a practicable, but a convenient way of working very large open-hearth furnaces. The present 75-ton furnace is regarded as simply a beginning. A British furnace is projected which will hold from 100 to 125 tons, and another is contemplated to hold 150 tons. The present 75-ton furnace will have shown—if, indeed, it has not already done so—that very much larger furnaces than have been hitherto used can be worked, with of course proportionate reduction in the cost for labor and fuel. I say convenient way, for this continuous method is certainly far more convenient than the intermittent method for these large furnaces. If we are manufacturing ingots, it is extremely inconvenient to have charges of from 100 or 150 tons delivered at once to the blooming mill, as would be the case with a 100- or 150-ton furnace worked on the intermittent plan. If we had a single 150-ton ladle, this thing itself would be a stupendous thing to manage and carry, and the stopper difficulties would probably be very serious. This difficulty might be turned by pouring the metal into several smaller ladles, say of 40 tons capacity apiece, each of which could be deposited on the floor or in some receiving place, while the subsequent ones were filling from the furnace. This, however, might prove inconvenient, especially from the danger of chilling around the stopper. The plan of pouring say 40 tons into one ladle, pouring this into ingots, returning to the furnace and taking in the same ladle 40 tons more, would involve serious difficulties in maintaining the bath at the proper composition from the time of drawing off the first lot to that of drawing off the last. But however this might be managed, the great difficulty would remain on the intermittent plan, that 150 tons of steel would be delivered either at once or within a very short time, and would have to be accommodated in the soaking pits of the blooming mill. It is here that the Bessemer process has a very great advantage over the open-hearth, in that it delivers to the blooming mill relatively small quantities of say 10 tons each, at very short intervals.

For the pig and ore process which is so extensively used in Great Britain, Mr. Talbot's process offers an advantage in that the whole of the charge may consist of molten metal direct from the blast furnaces, for the tendency of the molten iron to cut the furnace bottom is restrained by the fact that, when poured into the Talbot continuous bath, it does not impinge on the bottom, but spreads

out in every direction in the bath already present. Formerly this advantage would have seemed of greater weight than to-day. For, passing by the successful use of molten pig iron in pig and scrap practice (for here the scrap iron charged before the pig may be thought to protect the bottom from the stream of molten pig) in the Bertrand-Thiel process is has now been found practicable to have some 80% of the bath consist of molten pig iron. But, even though it be thus practicable to use in intermittent practice so large a proportion of molten pig iron, nevertheless we must lay it to the credit of the Talbot process that in it the stream of molten iron does not impinge on the bottom; for, by avoiding this, the process certainly avoids a thing which probably tends to shorten materially the life of the bottom.

Waste in Melting Down.—When in the common intermittent procedure we melt down charges composed in large part of scrap iron, especially if it be fine, *i.e.*, small scrap, a very serious loss of iron is liable to occur from the oxidization and cutting action of the flame upon this scrap iron. Now, while Talbot's process has thus far been applied to pig and ore work only, no reason appears why it should not be applied also to pig and scrap work; and in this case it should effect a considerable saving of iron. For the oxidation of the scrap iron should be much less if it were dropped into the deep Talbot bath and there melted under a layer of already very fluid slag, and thus could at once protect the iron from the flame efficiently. Mr. Talbot's results support this idea; for instead of the usual loss of iron in the process as a whole, the weight of metal produced is greater than that of the pig and scrap iron charged by about 7%. This, of course, is due to the reduction of part of the iron of the scale or ore used, by the carbon, silicon and phosphorus of the pig iron which are oxidized by the reaction

$$C+FeO=CO+Fe.$$

For every 12 lb. of carbon oxidized and removed, 56 lb. or over four times as much iron should be reduced from the slag.

Maintenance of Sides and Bottom.—In the common intermittent working the bottom is exposed to the iron oxide and the silica which, during melting down of the solid charge, form from the oxidation of the iron and silicon of that charge. The sides, too, are exposed to the corrosive action of the ferruginous and silicious slag which is liable to form during and immediately after melting down. In both ways the difficulties of maintaining the bottom and lining are considerably increased. But Talbot's process turns this difficulty, for first the oxidation of iron during melting is greatly restrained, as just explained; second, both the iron oxide and silica which result from the oxidation of iron and silica are prevented from attacking the bottom by the great bath of metal which at all times underlies the slag. The slag cannot attack the bottom because it cannot reach it. The action on the sides, too, is greatly lessened, by the fact that at no time does the slag become ferruginous or silicious. The silica which arises from the oxidation of the silicon at once enters the overlying slag, and is thus enormously diluted; while all the iron oxide which would form even if fine scrap iron were charged (and as already explained, but little should

form) would be either absorbed by the enormous bath of molten metal present or greatly diluted by the relatively large bulk of residual slag present.

The corrosion of the bottom and sides in common intermittent practice comes from the fact that iron oxide and silica form abundantly during the melting down, when the bottom and sides are naked to their attack, before the iron has so far melted that its carbon can reduce the iron oxide, and before the lime has so far melted as to neutralize the silica. In Talbot's process the oxidation of iron is restrained; and the great fly-wheel of bath and slag always present prevent iron oxide from ever reaching the bottom, and so far dilute them that the slag at no time holds enough of either or both to be strongly corrosive.

Of course 1% of silicon yields just as much silica in the continuous as in the intermittent process. The difference is that in the intermittent process there is a period at which this silica not only reaches the bottom, but also forms a high percentage of the then existent slag; whereas in the continuous process there is no such period—just as the thrice-daily-eating citizen avoids totally the anaconda period of the alleged weekly-eating savage.

Maintenance of Roof.—The present roof had run 572 charges at the time of my visit, and it was expected to last at least 1,000 charges. The charges which I here speak of are the small charges of 20 tons each, so that the output at the present time on the present roof is equivalent to about 145 seventy-five-ton charges, or such charges as would be made if the furnace had been worked intermittently instead of continuously. It might perhaps be thought that running the furnace continuously for a week at a time, giving the roof no chance to cool down, would shorten its life, that the cooling which the bricks get between heats in common intermittent practice might materially prolong their life. On the other hand, in some respects the conditions in Mr. Talbot's process favor the roof: (1) The somewhat abrupt changes of temperature to which the roof is exposed in intermittent practice may well be suspected of tending to injure it; (2) we know that there is a great tendency to injure the roof, that is to overheat it, during any period when the metal is not boiling. The slag itself is, of course, a very bad conductor of heat. If it lie tranquil, conducting heat but slowly to the bath underneath, its upper surface tends to become very hot, and to reflect back against the roof with fatal effect the heat of the flame. When the bath boils, the slag is turned over and over, and any individual particle which lies on the surface will be relatively cool, because but a few seconds before it will have been in contact with the bath of metal beneath, which in itself is a very good conductor of heat, and owing to its great mass has enormous capacity for heat.

Now in intermittent working there are two periods when there may be no boiling or but little, and when, consequently, the roof, if not threatened, at least is tried. These are: (1) During the end of the melting down, when enough slag has formed to act as a non-conducting mirror and condenser of heat, to condense and reflect the heat of the flame back against the roof, but when the metal underneath is not boiling. Were we to attempt to make it boil at this period by adding ore we should meet two obstacles: (*a*) The addition of the cool ore would be liable to make the metal freeze again on the

bottom, a difficulty which perhaps might be overcome by the use of molten as distinguished from cold iron oxide; and (*b*) that the temperature is not high enough for boiling. It is a most familiar observation that in the Bessemer process the rapid oxidation of the carbon does not begin till a very high temperature has been reached. At the beginning of the Bessemer process when the temperature is relatively low, the silicon oxidizes rather than the carbon; and it is not until the temperature has risen very high that the carbon begins to burn rapidly. So powerful is this influence of temperature in the Bessemer process, that if the temperature rise slightly above the normal, the oxidation of silicon is greatly restrained and that of carbon accelerated, with the result that, when the removal of the carbon has been completed and the flame has dropped, a large amount of silicon will remain, having escaped oxidation. At this abnormally high temperature the power of carbon to take oxygen so greatly exceeds that of silicon as to restrain greatly the removal of the latter element.

In the Bell-Krupp process a similar thing is noticed. In this process we aim to remove silicon and phosphorus from pig iron without removing carbon, by mixing the molten iron with a bath of iron oxide. We are obliged to keep the temperature relatively low, for if we raise it unduly, then the carbon begins to oxidize in preference to the silicon and phosphorus.

And so it is in the open-hearth process; unless the temperature be high the addition of ore does not bring on a free boil or, as the melter says "the metal doesn't take the ore."

(2) The other period at which the roof is tried because there is no boil is toward the end of the process, when we are waiting for the composition and temperature to reach simultaneously the proper point. Here we wish to avoid a boil, because a boil implies a considerable quantity of iron oxide in the slag, and as this slag is completely thrown away at the end of the intermittent process, this iron oxide would represent a serious financial loss.

Now the first of these periods, that immediately after the melting down, does not occur at all in the Talbot process: when we introduce any given lot of molten iron into the enormous bath, that bath acting as a fly-wheel or diluent, brings the temperature of the iron that is introduced immediately to the point at which a rapid boil begins. Then as regards the second boil-less period, that at the end just before tapping, it is argued with some plausibility that this should be considerably less in Talbot's process than in the intermittent process, because it has no long delays in waiting for the temperature to come right, and this in turn because so large a mass of metal changes temperature relatively slowly, and may always be held much more closely to the normal temperature than a smaller bath can be. This, of course, applies with especial force to the projected 125- and 150-ton furnaces.

Against this it may be urged that, if we tap out the metal from a 75-ton furnace in lots of 20 tons each, we have to bring the metal to the right temperature and composition about four times as often as if we tapped the 75 tons out in one lot, as in the intermittent procedure. But as regards the life of the roof this point is more apparent than real; four brief, boil-less exposures are

probably much less trying than one longer one, just as three moderate meals a day are much less trying to the digestion than one gorge.

To sum this all up, on one hand the life of the roof in the Talbot process may be shortened by the fact that it misses the cooling between heats which arises in the intermittent progress; on the other hand, it should be increased by the fact (1) that the abrupt changes of temperature between heats are avoided; (2) that the boil-less period at the end of melting down is avoided, and (3) that the boil-less period just before tapping, while recurring more frequently, is shortened in duration.

Only long experience can show what the net effect of these different elements is. At present we can say only that there is nothing in the experience thus far to indicate that the life of the roof will be shorter than in intermittent practice, and that there are good reasons to hope that it may be longer.

Wider Range for Composition of Pig.—Instead of specifying the upper limit for silicon as 1%, Mr. Talbot puts it at 1·5%, for a very simple reason already hinted at in passing. In the common intermittent practice, the silicious slag resulting from the oxidation of the silicon of the pig iron in melting down, attacks the bottom, which at that time is not fully protected by the molten iron. To prevent this action the upper limit for silicon in basic open-hearth iron is usually put at 1%. Of course there is the further reason for low silicon, *viz.:* that the silica in the slag may at all times be relatively low, so as not to interfere with the removal of phosphorus.

In Talbot's process the former difficulty does not arise at all; the molten iron at all times covers the bottom and protects it from the action of the silica thus formed. Further, when the silica forms in the Talbot process, it finds an abundance of lime and iron oxide already formed to neutralize it, and thus it is prevented from attacking the sides. In short, the period which we have in the early part of the melting down in common intermittent practice, when the silica arising from the pig iron is neither kept off the bottom by a bath of molten iron nor neutralized by an abundance of lime and iron oxide—this period does not at all arise in the Talbot process. Now the blast furnace can much more easily and cheaply supply cast-iron with the higher upper limit of 1·5% for silicon, and can more readily hold the sulphur below the limit of 0·05% with that silicon limit, than if the silicon limit be put at 1%; and this in itself gives the continuous process a great advantage.

Rapidity of Operation.—There are several reasons why we should expect considerably more rapid working in the continuous than in the intermittent process. (1) The rapid oxidation of carbon, as I have already pointed out, takes place only at a relatively high temperature. In the intermittent process this temperature cannot be reached until a considerable length of time after the charge is melted down. The bath is of course cooled during melting, and after the end of the melting it cannot, in the intermittent process, heat up quickly, because it is protected from the heat by the thick and now tranquil layer of slag floating above it. Only when the charge gets hot enough to boil do we get rapid oxidation. In the Talbot process, on the other hand, the charge is at all times kept

at a temperature high enough to bring on a rapid boil. The moment the pig iron is introduced it begins boiling rapidly.

(2) For much the same reasons as in the case of the Bertrand-Thiel process, more rapid working is permissible in the continuous than in the intermittent process. In the latter, even after the boil begins, the whole of the slag is present, because we may not remove any of it until its iron oxide has been so far reduced that we can throw it away without thereby involving considerable loss. Now when there is so large a quantity of slag present, we cannot have so rapid a boil as when there is only a small quantity of slag present, as is the case in the Talbot process. In this process there is at any one time in the furnace only the slag corresponding to one charge, or 20 tons of metal; whereas, if the same furnace were worked on the intermittent plan, there would be present about four times as much slag, *viz.:* that corresponding to the whole 75 tons of metal. Now a violence of boiling which would be quite permissible with this smaller quantity of slag would, with the larger quantity make the slag overflow at the doors, or froth up into the ports. In short, the smaller quantity and the great fluidity of the hot slag of the Talbot process permit a rapidity of decarburization, and therefore a violence of boiling, which would not be permissible in intermittent working

On the other hand there is one condition in the Talbot process which may be expected to delay matters; every time we tap the metal out of the furnace we must bring it to the right composition and temperature. If, now, we tap the metal from a 75-ton furnace in lots of 20 tons, we will have nearly four times as many of these periods of delay, waiting to adjust the temperature and composition, as if we tapped it out in one large lot of 75 tons. And while each of these periods of delay may be less in the Talbot process than in the other, yet the four taken collectively may exceed very considerably the single delay of the intermittent process.

It seems that Mr. Talbot's output is but slightly greater than that which could be expected from the same furnace if worked intermittently. The basic open-hearth process on pig and ore in the common intermittent working makes seven heats a week, and that is practically what Mr. Talbot's process does. He turns out from 500 to 525 tons of ingots in the 138 hours between midnight on Sunday and six o'clock on Saturday evening, which is equivalent to about seven heats of 75 tons each, or about 20 hours per 75-ton heat.

This is the one feature of the process which disappoints us. The Bertrand-Thiel process, as I show beyond, runs at the rate of four heats per furnace per 24 hours, or six hours per heat per furnace, or more than thrice as fast as the present rate of the Talbot process, when on a mixture but slightly more favorable for rapidity than that of Mr. Talbot, *viz.:* 92% of pig iron and 8% of scrap. Of course the actual output of the Talbot process, though rather small in number of heats per week, is very large in tons; and if, as seems probable, the cost of maintaining so large a furnace is really small per ton of output, this disadvantage of the Talbot as compared with the Bertrand-Thiel process may prove to be more apparent than real.

Over-Oxidizing in Melting.—In common practice it at times occurs that the

metal becomes seriously over-oxidized in melting down. This is liable to occur especially (1) if we use a large quantity of fine scrap; (2) if the supply of gas be insufficient; or, (3) if the melting down, which is a period of strong oxidation, be delayed for any reason, whether for insufficient gas, or for any defect in the regenerators, or in any part of the apparatus. Especially toward the end of the campaign is this difficulty liable to arise, when the furnace has no longer its full heating power. Under these conditions we have not only the corrosion of the bottom and sides already referred to, but great difficulty in bringing the charge to proper composition. As fast as we try to undo the over-oxidation by adding cast iron, or "pigging back," so fast are the carbon and silicon burned out; and it only too often happens that the cubic capacity of the furnace is reached, *i.e.*, that there is no more room to hold further additions of pig, before we have been able to undo this over-oxidation. Under these conditions we have to tap the charge out of the furnace without at any time having been able to collect in it sufficient carbon to free it from oxygen, or at least to free it from the conditions incident to its over-oxidation; and the consequence is that our resulting ingots are not of good quality. This difficulty, like the difficulty of the corrosion of the bottom during melting down, is of course entirely avoided by the continuous process.

Rephosphorization.—Since none of the slag escapes into the casting ladle, the danger of rephosphorization is entirely avoided. It is true that in any tipping furnace we may hold back the greater part of the slag and prevent it from running into the casting ladle, even if we work intermittently. Yet even in this case a very considerable quantity of slag should run into the casting ladle, for it is hard to see how we can get the last of the metal out of the furnace and into the ladle without allowing a considerable amount of slag to accompany it. But in the Talbot process, since the metal is tapped at a level considerably below the slag line, it should be perfectly easy to prevent the slag absolutely and altogether from running into the casting ladle.

To sum this all up, the continuous process offers the following advantages: (1) It delivers the steel in small lots, easily managed in casting, and convenient for the blooming mill to receive, and thus opens up a way, not only practicable but convenient, of working very large furnaces, with corresponding saving in labor and fuel.

(2) It probably leads to very much less waste in melting down, to less difficulty from over-oxidation in melting, and less difficulty in the maintenance of bottom, sides and roof.

(3) It permits the use of a pig iron richer in silicon.

(4) It avoids the danger of rephosphorization.

(5) It is hoped that, at least with still larger furnaces, the output will be greater than on the intermittent plan. There is, however, as yet no reason to hope that it will approach the output of the Bertrand-Thiel process, if we reckon both in the number of complete furnacefuls of steel made per week.

But it should be possible to combine the advantages of these two remarkable processes. The advantages which the continuous process has shown should be applicable to the Bertrand-Thiel process. It should be a great advantage

in the Bertrand-Thiel process to have the preparatory furnace at all times just in the right condition in all respects, temperature included, for most rapid work; and this we ought to be able to bring about by working the preparatory furnace continuously. And for that matter, no good reason appears why the finishing furnace of the Bertrand-Thiel process should not be worked on the continuous plan. If we had a 150-ton furnace turning out as many furnacefuls per week as the little Bertrand-Thiel furnaces do, we should have something far ahead of either the Bertrand-Thiel or the Talbot process as regards output. The Bertrand-Thiel feature would give us the great rapidity measured in number of furnacefuls; the Talbot feature would permit the use of large furnaces, thus increasing the size of each furnaceful; would deliver the metal even from a very large furnace in small manageable lots; would lessen waste in melting down; would prolong the life of sides, bottom and perhaps roof; would permit the use of pig iron richer in silicon, and would prevent rephosphorization.

3. *The Bertrand-Thiel Process.**—The important modification in this process made during 1899 was the introduction of the use of molten cast iron, which has given very gratifying results. From a pair of furnaces eight heats are made per 24 hours, or at the rate of four heats per furnace per day, when running practically on pig and ore. The charge actually consists of molten cast iron, 80%; cold cast iron, 12%; scrap, 8%. Although the molten cast iron contains C, 3·5%; P, 1·5%; Si, 1·0%; total, 6%; thus having only 94% of metal, there is instead of the usual loss by oxidation a gain of 2% from the reduction of iron from the ore; and the yield of clean ingots is given as 98·5%. These results are certainly extremely interesting and valuable.

In THE MINERAL INDUSTRY, Vol. VII., p. 421, in discussing this process, I confessed that I saw no easy explanation of its remarkably large output. On further consideration of the matter, however, the explanation became evident, and I have given it elsewhere.† I there gave three reasons, which are as follows: (1) The room available for the metal is greater, "(*a*) because less slag is formed owing to washing out much of the phosphorus and silicon with a relatively acid slag, and to oxidizing by means of gangueless iron oxide instead of iron ore; (*b*) because since the first slag is removed, a smaller proportion of the total slag is present, on an average of the whole time."

(2) The temperature is higher because "the pig and iron oxide are preheated highly before their reciprocal action begins."

(3) The more rapid action is made possible by the "greater fluidity resulting from that higher temperature of metal and slag, by the small quantity of final slag, and possibly by the larger ratio of iron oxide to lime in that slag."

4. *Fore-Hearths for Open-Hearth Furnaces.*‡—The Wellman-Seaver Engineering Co. has made the important innovation of attaching a large casting shoe, or fore-hearth, such as was used at the stationary open-hearth furnaces

* "Further Notes on the Bertrand-Thiel Process," J. Hartshorne, *Transactions of the American Institute of Mining Engineers*, Washington Meeting, February, 1900; E. Bertrand, private communication, Feb 24, 1900.

† *Engineering and Mining Journal*, LXVIII., Sept. 2, 1899, p. 276.

‡ S. T. Wellman, private communication, Jan. 22, 1900.

of the Bay State Iron Co., in South Boston in 1870, to the Wellman tipping furnaces at Ensley, Ala. This plan seems to have several advantages.

(1) In case of a leaky stopper or of any other trouble connected with the stopper or nozzle, the furnace is tipped back so as to run the charge of metal back into it, so that the stopper or nozzle may be made good.

(2) The metal need not be made so hot in the furnace, as would be expedient if it were to be poured into a ladle, as in common practice. For in this latter case we have to make provision, not only for the cooling off in the ladle, which is greater than in a fore-hearth, but further we have to have a greater factor of safety. That is to say, we have to have the metal not only hot enough, but amply hot, so that there shall be no possibility of its chilling in the ladle; for if it does the matter is extremely serious. With the forehearth, however, no such factor of safety is required, because if by chance the metal when run into it is not quite hot enough to pour, we can by tilting the furnace run it back and heat it hotter. This lower temperature should save the furnace somewhat, should lessen the oxidation of the metal, should lessen the consumption of the manganese in the final additions, and, in the opinion of some, should yield ingots of better quality.

(3) The slag difficulties are greatly diminished. As soon as the metal has been poured into the ingot moulds, the furnace is tipped back so that what little slag is in the fore-hearth shall run back into the furnace; the fore-hearth is then removed and the slag is poured out into a slag ladle, for instance a Weimer cinder car, exactly as in blast furnace practice. During this time the heat has been kept in the furnace, and the slag is so hot that when the cinder car is carried by a locomotive to the slag bank, the slag is poured out easily.

The objection urged beforehand to the use of the fore-hearth was that, as it would be necessary to add the final additions in the furnace, there would be a greater loss of manganese and of silicon; first, because of the "fining" action of the slag lying above the bath in the furnace; and second, because of the longer time which the metal after receiving the final additions would remain in contact with the slag. I am informed, however, that this influence is practically offset by the influence of the lower temperature at which it is possible to have the metal at the moment of making the final additions, as already explained.

5. *The Daelen-Pszczolke Process* has developed no further points of technical interest, unless we include under this head the fact that it is to be adopted at Russian and at two German establishments. It is certainly one of the interesting processes now before metallurgists, and may prove to be of very great importance.

6. *Direct Wire-rod Rolling.*—This has been done by William Garrett at Joliet, Ill., and the Garrett-Cromwell Engineering Co. is now building a direct-rolling plant which consists of (1) a basic open-hearth furnace, (2) a blooming mill, (3) a rod mill.

The steel will be cast in 9-in. ingots, which, after soaking, will be rolled into 4-in. billets, and these in turn will be cut into several parts and rolled immediately at the same heat into 1·75-in. square billets. These in turn, after passing

through another soaking pit, will be rolled by a continuous mill and the finishing rod mill into wire rods. It is estimated that the saving by rolling the 4-in. billets with their initial heat, instead of allowing them to cool down, will be about 50c. a ton, which at 300 tons of rods per day, and 275 working days per year, means a saving of $41,250 yearly.*

7. *Alloy Steels.*—In the case of nickel steel we have to record a considerable broadening of the field of usefulness. In particular, more than one American concern has gone into the manufacture of 25 and 30% nickel steel wire for electrical resistance purpose, and the Pennsylvania Railroad is said to have ordered a considerable trial lot of nickel steel rails.

The use of manganese steel has increased very considerably, and the manufacture of manganese steel safes has already begun on a commercial scale.

8. *The Allotropic Theory of the Hardening of Steel.*—Important evidence supporting this theory has been offered by M. Galy-Acré.† One of the difficulties in the way of the allotropic theory has been the fact that, in the absence of carbon, sudden cooling which might be expected to retain some of the allotropic high-temperature iron, has appeared not to cause any hardening. It is true that several investigators have shown that, even if but very little carbon be present, a degree of hardening can be induced altogether disproportionate to the percentage of carbon. Thus I found that a steel containing only 0·022% C had its elastic limit raised 157% by quenching in melting mercury, and that steel containing only 0·009% C had its elastic limit raised 17% by the same means. Unfortunately it was later found that this latter steel contained in addition 0·177% Si and 0·22% Mn, so that the value of this evidence was lessened.

The importance of M. Galy-Acré's evidence is in showing that, while iron nearly chemically pure, and wholly free from carbon, can be slightly hardened by sudden cooling, the hardening thus induced passes away with time. He thus shows that the allotropic iron, while retainable even in the absence of carbon, is even in the cold extremely unstable, slipping back to the normal or α state; and he thus supports that part of the allotropic theory which holds that the action of carbon in hardening is that of a brake, retaining the glass-hard allotropic iron which, but for the carbon, would during even sudden cooling slip back so nearly completely to the normal state that the slight degree of hardening induced by the very small quantity of allotropic iron retained, would escape any but the most careful scrutiny. He makes this explanation more reasonable by showing, first that the allotropic iron is slightly retainable even in the complete absence of carbon; and second, that when thus retained it really is unstable, thus demonstrating that instability which the allotropic theory demands.

* William Garrett, private communication, Feb. 1, 1900.

† *Compte. Rendus*, p. 1230.

LEAD.

THE production of lead in the United States during 1899 showed a marked decrease from the production in 1898—the amount falling very nearly to the figures quoted for 1897. The grand total of soft, desilverized and antimonial leads produced from both domestic and imported ores and refined from imported base bullion, amounted to 266,269 metric tons (293,508 short tons) against 288,201 metric tons (317,684 short tons) in 1898. The totals of desilverized, soft and antimonial leads in short tons, derived from domestic ores in 1899 and 1898 were respectively as follows: Desilverized, 171,495, 169,364; soft, 40,508, 50,468; antimonial leads, 7,337, 8,643 (of which 2,295 tons were produced from foreign ores); total domestic leads, 217,085, 228,475.

The decrease in the production of domestic lead is due to various causes. In soft lead the decrease was chiefly in the Joplin district, where the closing of the mines and the great demand for zinc ores caused the lead output to decline. In desilverized lead, the principal cause of the decrease was the labor troubles in the Cœur d'Alêne district, Idaho, which resulted in a diminished ore supply. The decrease in the amount of lead from foreign ores was traceable to shorter supplies of ores and base bullion from British Columbia, due to complications in the Canadian tariff. The receipt of lead bullion from Mexico continued very large.

PRODUCTION AND CONSUMPTION OF LEAD IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Stock of Refined, Jan. 1. (a)	Produced from United States Ores.				Imported in Ores and Bullion.	Total Supply.	Consumed.	Exported in all Forms.	Stock of Refined, Dec. 31. (a)
		Desilverized.	Soft.	(b) Antimonial.	Totals.					
1895....	12,634	119,057	32,797	5,000	156,854	100,471	269,959	232,854	18,130	18,975
1896....	18,975	135,332	33,498	5,992	174,822	80,159	273,826	203,819	56,983	13,024
1897....	13,024	141,649	45,710	7,359	197,718	92,117	302,859	211,345	60,353	31,161
1898....	31,161	169,364	50,468	8,643	228,475	89,209	348,845	246,989	78,168	23,688
1899....	23,688	171,495	40,508	7,377	217,085	76,423	317,196	215,662	74,944	26,590

(a) Includes lead in bond, which amounted to 11,160 tons at the end of 1899, 7,345 tons at the end of 1898, 6,694 tons at the end of 1897, 4,124 tons at the end of 1896, and 9,865 tons at the end of 1895. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin; it is, however, impossible to separate this in the statistics; owing to this inability the division of the American production between antimonial and desilverized is not quite accurate, though the error is not important.

The main feature of the year 1899 was the formation of the American Smelting and Refining Co., whose interests embraced all the refining companies of the country, except the Guggenheims, whose smelter is at Pueblo, Colo., and whose refinery is at Perth Amboy, N. J.; the Balbach Co. at Newark, and the refineries on the Western coast. These companies outside of the trust confine

themselves principally to the treatment of imported ores and add a comparatively small amount to the domestic production. The capitalization of the American Smelting and Refining Co. is $65,000,000, divided into 650,000 shares, of which one-half is preferred stock, entitled to cumulative dividends of 7% per annum. They produce annually about 175,000 tons of lead and practically control the market for ores and to a large extent the market for lead. The opposing interests are the soft lead producers of the Missouri district, the sale of whose output has practically been confined to St. Louis and adjoining territory. A strong tendency existed toward consolidation in other branches of the lead industry. Efforts were made to unite the shot, sheet and pipe interests under one control and to combine all the producers of white lead; although negotiations were pushed strongly they did not meet with success.

Colorado.—The strike among the smelter workmen began to affect ore production early in June and many weeks elapsed before smelters resumed full operations. Notwithstanding this disturbance the production of lead in this State during 1899 was greater than in 1898. Hon. H. A. Lee, Commissioner of Mines, of Colorado, reports the following statistics: The production of lead in Colorado in 1899 was 69,024 short tons, against 56,708 tons in 1898. Of this Leadville produced 24,299 tons, against 17,973 in the previous year; Pitkin County (chiefly Aspen) increased from 7,951 to 12,729 tons. The production in San Juan County increased from 7,330 to 8,006 tons; and Hinsdale County (Lake City) increased from 4,914 to 5,286 tons. The balance of the total production was derived from other counties in the State, Clear Creek (Georgetown), Ouray, Mineral (Creede), Summit and San Miguel being the most important producers.

Idaho.—The production of lead in the Cœur d'Alêne district in 1899 was 50,006 short tons, against 56,339 tons in 1898. The output of silver in 1899 was 2,736,872 oz. against 3,339,524 in 1898. The statistics of silver and lead production in this State are based on direct returns from the mines, with an allowance of 6% for loss of lead in smelting and 2% for loss of silver. The labor troubles between the Miners' Union and the Bunker Hill & Sullivan Co., which culminated on April 29, 1899, in the destruction of the 400-ton concentrating mill by a mob of miners using dynamite, interfered with the usual output. The Tiger & Poorman mine is now operated by the Buffalo Hump Mining Co. Their production in 1899 was about one-third of their regular output, the decrease being due to labor troubles and shut-downs during the year. There are seven important producers in the Cœur d'Alêne district who mechanically concentrate the low grade ore to a product averaging about 50% Pb and 30 oz. Ag.

Iowa.—The production of lead in the mines at Dubuque and vicinity was about 500 tons. Most of this ore is smelted at Dubuque.

Missouri.—The production of lead in S. E. Missouri declined from 35,769 tons in 1898 to 31,355 tons in 1899. While several new companies were formed in 1899, they have not as yet reported any output. The older companies continued their operations with but little change. Of these the more important producers are the St. Joe & Doe Run Lead Co., the Picher Lead Co., the Central

Lead Co., the Desloge Consolidated Lead Co., the Missouri Smelting Co. and Mine La Motte.

In the Joplin district (including S. E. Kansas and Iowa) there was a decrease in the production of lead, due to the greater activity in zinc mining which, on account of the high prices of zinc, yielded a greater profit to the miners; there was also a stoppage of output by the mines. The average monthly prices for lead ore per 1,000 lb. in 1899 were as follows: January, $23·9375; February, $26·50; March, $25·80; April, $25·25; May, $26; June, $26; July, $26·90; August, $27·25; September, $27; October, $26·90; November, $27·20; December, $27.

Montana.—The production of lead in Montana in 1899 was reported by E. B. Braden, United States Assay Office at Helena, as 10,172 short tons, against 10,702 in 1898. Practically all the lead ore produced came from the districts in Cascade, Broadwater and Meagher counties. The recently erected 100-ton smelting plant at Twin Bridges has proved of considerable benefit to the industry.

Utah.—On April 15, 1899, the smelters in this State were transferred to the American Smelting and Refining Co. As a result the Hanauer plant was placed out of commission, as the high grade Utah ore could be treated to better advantage by the smelting plants of the company at Denver, Omaha or Kansas City, leaving only the low grade ore to be smelted at Salt Lake City.

Washington.—Of the two smelters in Washington the Puget Sound Reduction Co. refined its own bullion, while the Tacoma Smelting Works shipped its bullion to the Selby Smelting & Refining Co. in California for treatment. The Puget Sound Reduction Co. reported a diminished output in 1899.

LEAD PRODUCTION OF THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Austria.	Belgium.	Canada.	Chile.	France.	Germany.	Greece.	Hungary.	Italy.
1895.....	8,085	15,573	7,467	93	7,627	111,058	19,845	2,377	20,353
1896.....	9,769	17,222	10,977	594	8,232	113,792	15,180	1,911	20,786
1897.....	9,860	17,023	17,608	370	9,916	118,881	16,486	2,527	22,407
1898.....	10,340	19,330	14,477	*d* 375	10,920	132,742	19,193	2,305	24,543

Year.	Japan.	Mexico.	New South Wales. (*c*)	Russia.	Spain.	Sweden.	United Kingdom. (*b*)		United States.	Totals.
							Foreign Ores.	Domestic Ores.		
1895....	1,950	68,000	38,000	412	160,786	1,256	16,036	29,464	142,298	650,580
1896....	1,958	63,000	30,000	262	167,016	1,530	25,899	31,311	158,479	676,662
1897....	1,737	71,637	22,000	450	*e*189,216	1,480	13,312	26,988	179,369	721,167
1898....	*d* 1,950	71,442	*d* 23,000	*d* 450	*e*193,764	1,559	23,239	25,761	207,271	782,261

(*a*) The statistics for Austria, Belgium, Canada, France, Germany, Hungary, Italy, Japan, Russia, Spain and Sweden are from the official reports of the respective governments except where otherwise noted. Those for Greece are based on the authorities given under the general table of mineral production of Greece in a subsequent part of this volume. Those for the United States are from data collected by THE MINERAL INDUSTRY. Those for the United Kingdom as specified in note *b*.

(*b*) The production of lead in the United Kingdom is given in two columns. One gives the amount of lead derived from domestic ores, the yield of which is calculated at 95%, as reported in the official British blue books. The other column gives the production of lead in the United Kingdom from foreign ores smelted there. These figures, which are not reported in the official blue books, are obtained by deducting the production of British lead from the total output of the lead smelters of the United Kingdom as stated in the *Statistische Zusammenstellungen über Blei. Kupfer. Zink und Zinn* of the Metallgesellschaft. Frankfurt-am-Main.

(*c*) From the *Statistische Zusammenstellungen* of the Metallgesellschaft, Frankfurt-am-Main. These figures comprise only the lead exported to Europe and America. The actual production of lead in New South Wales is much larger than shown here, especially in late years, during which there have been large exports to China, but it is impossible to obtain statistics as to what it really amounts to.

(*d*) Estimated. (*e*) From the *Revista Minera*, etc.

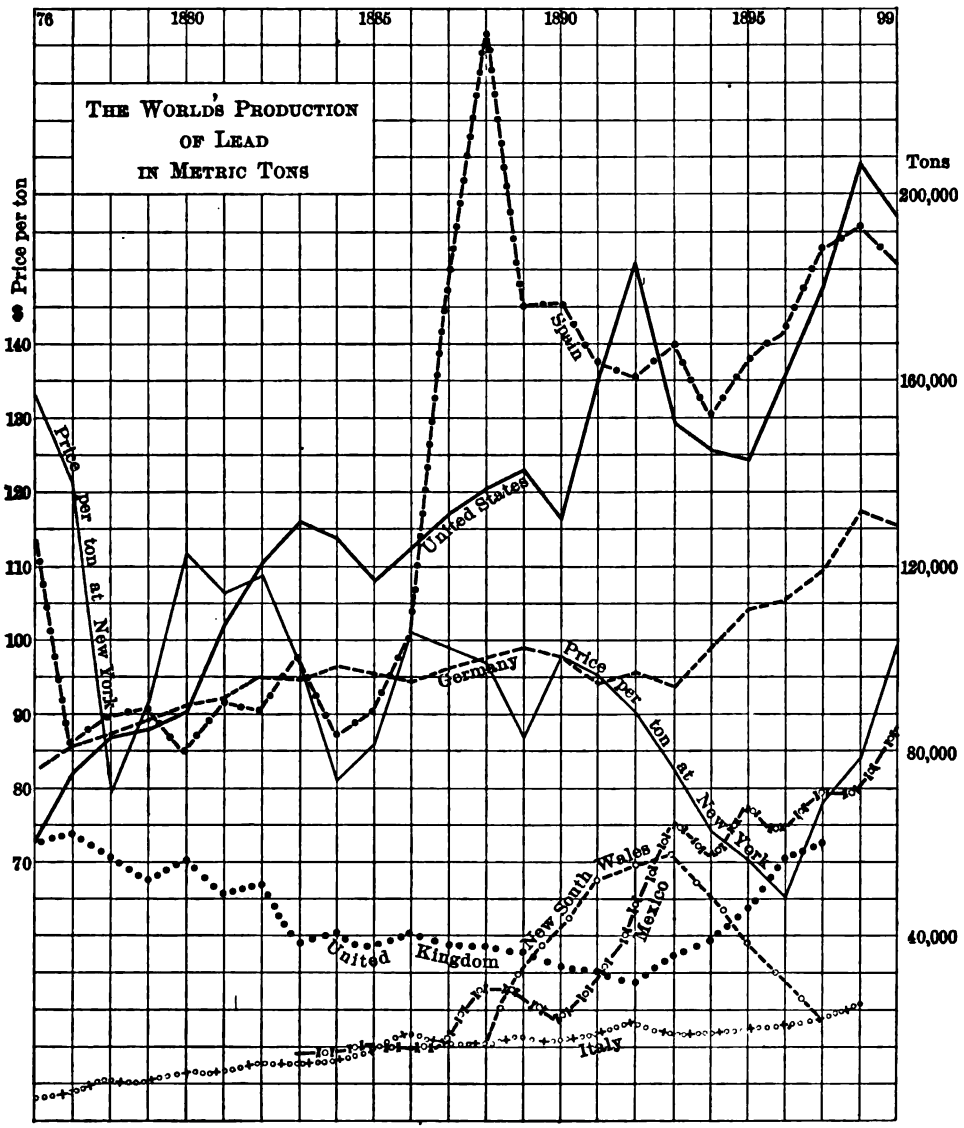
PRODUCTION, IMPORTS, EXPORTS AND CONSUMPTION OF LEAD IN THE CHIEF COUNTRIES OF THE WORLD. (IN METRIC TONS.)

		Aust'a-Hungary. (a)	Belgium. (b)	France. (c)	Germany. (d)	Italy. (e)	Russia. (f)	Spain. (g)	Switzerland (h)	United Kingdom. (i)	United States. (j)
1895	Production	10,362	15,573	7,027	111,058	20,353	412	160,786	*Nil.*	45,500	142,298
	Imports	8,974	45,594	67,183	28,449	947	29,556	*Nil.*	3,571	165,531	91,454
	Totals	19,336	61,167	74,810	139,507	21,300	29,968	160,786	3,571	211,031	233,752
	Exports	60	39,996	10,734	27,855	3,646	*Nil.*	151,129	298	22,920	16,447
	Consumption	19,276	21,171	64,076	111,652	19,147	29,968	9,657	3,273	188,111	217,305
1896	Production	11,680	17,222	8,222	113,792	20,786	262	170,790	*Nil.*	57,200	158,479
	Imports	7,221	35,221	80,572	33,016	1,166	31,384	*Nil.*	4,350	170,484	72,720
	Totals	18,901	52,443	88,804	146,808	21,952	31,646	170,790	4,350	227,684	231,199
	Exports	88	31,366	11,275	24,828	1,419	*Nil.*	166,194	244	18,976	54,752
	Consumption	18,913	21,077	77,529	121,980	20,533	31,646	4,666	4,106	208,708	176,447
1897	Production	12,337	17,023	9,916	118,881	22,407	450	189,216	*Nil.*	60,000	179,369
	Imports	5,937	43,840	88,050	35,092	1,178	24,300	*Nil.*	4,536	170,120	84,060
	Totals	18,274	60,863	97,966	153,973	23,585	24,750	189,216	4,536	230,120	263,429
	Exports	55	35,968	10,448	24,075	2,790	*Nil.*	181,216	262	20,749	54,633
	Consumption	18,219	24,875	87,518	129,898	20,795	24,750	8,000	4,274	209,371	208,796
1898	Production	12,645	19,330	10,920	132,742	24,543	.....	198,764	*Nil.*	49,000	196,937
	Imports	9,746	49,500	74,902	47,497	1,431	.....	*Nil.*	5,405	197,591	81,326
	Totals	22,391	68,830	85,822	180,239	26,074	.....	198,764	5,405	246,591	278,263
	Exports	353	43,000	3,663	24,867	5,870	.....	188,264	175	18,964	69,559
	Consumption	22,038	25,830	82,159	155,372	20,204	.....	5,500	5,230	227,627	208,704

(a) From *Statistisches Jahrbuch des K. K. Ackerbau Ministerium* and *Magyar Statisztikai Evkönyv.*
(b) Production from *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques.* Imports and exports from *Annuaire Statistique de la Belgique.*
(c) From *Statistique de l'Industrie Minérale.*
(d) Production, imports, and exports from *Statistisches Jahrbuch für das Deutsche Reich.*
(e) From *Rivista del Servizio Minerario.*
(f) From *Sbornik Statisticheskikh Svedenie o Gornozavodskoi Promyshlennostie Rossie v zavodskom Godu,* St. Petersburg.
(g) From the *Reports of the Comision Ejecutiva de Estadistica Minera.* Imports and exports from the *Revista Minera de Espana.*
(h) From *British Statistical Abstracts* and from figures furnished by the *Direction Générale des Douanes Fédérales,* Berne.
(i) Production from statistics of the Metallgesellschaft, Frankfurt-am-Main. Imports and exports from Board of Trade returns.
(j) The statistics of production are those collected by THE MINERAL INDUSTRY; those of imports and exports are from the reports of the Bureau of Statistics, Washington.

Australia.—The mines of Broken Hill, New South Wales, still continue to produce the greater part of the output. The Broken Hill Proprietary Co., which is the most important producer, made the following report for the six months ending May 31, 1899: During this time there were treated 148,149 tons of ore which yielded a total product of 18,255 tons of lead, 330 tons of hard (antimonial) lead, 2,268,631 oz. of silver and 5,537 oz. gold, and matte containing 30 tons of copper and 55,305 oz. silver. The average result on the basis of ore treated was 15·57% lead, 15·69 oz. silver, and 0·038 oz. gold per ton. The ore is derived from open cut working and underground, 76,942 tons coming from the former. Of the total 202,693 tons of ore mined, 59,959 tons were sent to the Port Pirie smeltery, 34,143 tons to the chloridizing plant and 108,382 tons to the concentrators. The increased crushing plant for the sulphide concentration enabled the company to handle 110,797 tons against 79,742 tons during the previous six months at practically the same cost. A slime treatment plant has been added, which has yielded a better recovery of lead. The mill is being extended to double its capacity. The smelting plant at Port

Pirie has been enlarged and now consists of 11 furnaces of 120 tons each and two of 80 tons each daily capacity. This enlargement was necessary to counteract the slow smelting caused by the fine condition of the ore. The experiments



The production in the United States, Spain, Mexico, New South Wales and Italy includes only lead derived from domestic ores; for the United Kingdom and Germany the output of smelters has been used. The production of New South Wales, which is based on exports, is somewhat larger than shown in the diagram, as the amount shipped to China has not been included.

made in briquetting the fine ore were very successful, and larger quantities of ore were put through the furnace, with a higher percentage of bullion recovery. The average extraction per ton of ore smelted was 17·6% Pb, 17·3 oz. Ag.

The total quantity of ore and by-products (matte, dross, etc.) smelted in the furnaces was 107,721 tons net. Plans are under consideration to increase the capacity of the briquetting plant so as to handle the entire amount of fines. With the extension of the concentrating, smelting and briquetting plants at Port Pirie, it will be possible in the future to handle smelting material more advantageously, as a more suitable proportion of the oxidized ores to the concentrates in the furnace charges can then be made. The averages per ton of ore reported for the semester ending May 31; 1899, reduced to United States currency were: receipts, $12·85; expenses, $9·49; profit, $3·36. The report for the semester ending November 30, 1899, states that the experimental work with magnetic separators of various types to recover the zinc contents of the ore and at the same time a portion of the lead and silver that at present goes with the zinc has been very satisfactory, and while not terminated as yet, they indicate a successful solution of the vexatious problem. The gross profit for the year ending November 30, 1899, amounted to £144,492 12s. 2d., from which £96,000 were distributed in dividends; 95,154 tons of ore were smelted, with an average of 18·48% Pb and 22·68 oz. Ag per ton. Incidentally 9,675·88 oz. fine gold were produced.

The report of the Sulphide Corporation, Ltd., for the year ending June, 1899, contains some interesting data of the result of the concentrating plant in treating zinc lead sulphides. The following table shows the classification of products:

Products.	Tonnage.	Proportion.	Assay Value.			Contents.		
			Ag. Oz.	Pb. %	Zn. %	Ag. Oz.	Pb. Tons.	Zn. Tons.
Concentrates.....	21,274·10	20·83	28 7	63·6	9·9	610,976	13,540·28	2,109·71
Middlings.....	21,105·60	20·66	9·2	7·4	28·9	193,406	1,571·24	6,098·49
Slimes.....	8,590·00	8·41	16·8	20·9	23·2	143,993	1,799·03	1,996·21
Tailings.....	49,845·95	48·81	6·8	4·4	24·0	338,502	2,191·98	11,936·48
Zincs.....	1,317·84	1·29	12·8	12·1	31·7	16,847	159·38	417·30
Net ore treated...	102,133·49	100·00	12·8	16·9	22·1	1,303,734	19,261·86	22,558·19

The "zincs" were the product of treating middlings by jigging, which gave a recovery of 81% of the silver, 86·3% of the lead and 70·89% of the zinc. The value of the middlings has in consequence been greatly enhanced.

Canada.—The production of lead in ore in Canada during 1899 was 10,932 short tons, derived chiefly from the Hall mines. The report from this company, however, was not as satisfactory as had been expected. The quantity and quality of the ore fell below the estimates and working was handicapped by labor complications at this and at other mines. The profit amounted to only $45,000, though debentures to the extent of $122,800 had been issued during the year. It is proposed to issue the remaining $125,000 in debentures which the shareholders had authorized previously. The ore for the most part is shipped to the United States for reduction. A certain amount, however, is treated locally at the furnace at Nelson, B. C., operated by the Hall Mines, Ltd. The bullion produced averages 195 oz. Ag, 3 oz. Au and 98% Pb. The Trail smelter has been greatly improved under the management of the Canadian Pacific Railway

Co., and, while now running on copper ores, it is reported that it can be made ready to treat lead ores at short notice. It is probable that the Trail works can compete favorably with the smelters in the United States for the treatment of the ores of the Slocan district, but it is handicapped in the sale of the product by the fact that there are no refineries in Canada for transforming the base bullion into marketable forms of pure lead, pure silver, or doré bars. At present the bullion is shipped to the United States and refined in bond, which can be done under the United States law without the payment of an import duty, but the existing Canadian law requires the refined lead to pay a duty of 15% upon its re-entrance into Canada; at the same time, lead imported from England is favored in the Canadian tariff by a differential reduction of 25% of the duty, and, as a result, the Canadian producer after having his lead refined in the United States may be forced to sell it in England and have it reimported from there. The total annual consumption of lead in Canada is 3,000 tons, which yields an income to the Government of about $37,000. It would hardly be prudent to erect a lead refining plant in Canada until a sufficient supply of base bullion was assured. In view of the limited consumption of refined lead in Canada and the existence of the present import duty into the United States of 1·5c. per lb. of lead in ores, 2·125c. per lb. of lead in pig, bars, etc., and 2·5c. per lb. of lead in sheets, pipes and manufactured forms, the outlook for refining base bullion in this region is not very favorable. Considering the advantage that will accrue to the Government from the treatment of the total amount of lead ore mined annually in British Columbia (40,000 to 50,000 tons) it is probable that the tariff will be adjusted to favor home smelting and refining. The consumption of lead would be increased by promoting the establishment in Canada of manufacturing industries involving the use of lead.*

Germany.—The production of lead in Germany in 1899 was 129,225 metric tons, against 132,742 in 1898, a decrease of 3,517 tons. The production of lead ore in 1899 likewise decreased, and for the two years was respectively 144,370 and 149,311 metric tons, showing a diminution of 4,941 tons for 1899. The amount of litharge produced in 1899 was reported at 3,562 metric tons.

Mexico.—The production of lead in ores and bullion in Mexico in 1899 was 84,656 metric tons, against 71,442 in 1898, about 90% of the production each year being exported to the United States. The Compañia Minera de Peñoles, of Mapimi, Durango, increased its output materially in 1899, and shipped considerable of its bullion to the United States for refining, the balance going to Europe.

The first refinery to be erected on Mexican soil is now in course of construction at Monterey under the operation of the Compañia Minera Fundidora y Afinadora de Monterey. The refinery is the ordinary Parkes' process with a daily capacity of 40 tons of bullion, with provision, however, for extension should increased tonnage be necessary. There will also be a parting plant for operating under the Moebius patents.

In a paper read before the Colorado Scientific Society, January 6, 1900, H. F. Furman gives a brief description of the mines and smelter of the Compañia

* *Engineering and Mining Journal*, June 3, 1899, p. 644.

Minera de Peñoles. The Ojuela and San Ignace silver-lead mines are situated 9 km. south of the town of Mapimi. The limestones surrounding the ore bodies are nearly horizontal to the eastward, while to the westward they are badly faulted and twisted by a volcanic dike which intersects the formation. This volcanic intrusion antedates and has much to do with the deposition of the ore which appears to have replaced the limestone in irregular masses. The bottom of the great stope of the Ojuela mine is 2,250 ft. below the surface. At the 2,200-ft. level the stope is about 1,000 ft. across and the overlying ground to the height of about 1,000 ft. has been stoped continuously. The ores contain cerussite, anglesite, galena, boulangerite and various oxidized iron, lead, antimony, arsenic and sulphur compounds. They are for the most part oxidized and generally occur with considerable galena. The following table presents the average partial analysis of the ores as they are "bedded" at the smelting works:

Name of Ore Mixture.	SiO_2. %	Fe_2O_3. %	Al_2O_3. %	$CaCO_3$. %	As. %	Pb. %	Ag. Oz. pr. Ton.
Ojuela	7·0	30·0	10·0	9·0	6·0	23·0	18·0
San Ignace	8·0	35·0	12·0	6·0	11·0	18·0	18·0
Socovon	1·0	35·0	12·0	5·0	9·0	20·0	23·0
Rescates	50·0	12·0	10·0	18·0	.....	2·0	35·0

In addition to the constituents given in the above table the ores carry considerable antimony, small amounts of gold, copper, manganese, zinc and sulphur. The Rescates ore mixture is made up of low grade silicious silver ores, which are purchased to flux the basic lead ore mined by the company. The smelting plant at Mapimi consists of 10 shaft furnaces of special construction to meet the requirements of the strongly basic bullion and the large amount of speiss produced. The furnaces are 36×100 in. cross-section at the tuyeres and are equipped with steel water-jackets. The hearth slopes to the front and has a separate tap for the bullion 6 in. below the level of the slag tap and located at the side of the furnace near the front. The ore column is 11·5 ft. high. It was found necessary to replace the ordinary deep crucible and external lead well with the inclined hearth, as the bullion was so base that it was practically impossible to keep the connection open between the crucible and the lead well. With this modified furnace the slag and speiss are tapped into settling pots, the slag overflowing into slag pots. At the time of tapping the bullion, about every two hours, the settling pot is removed and the molten bullion, slag, and speiss flow into a pot. The lighter slag and speiss are allowed to chill partially and are removed from the underlying molten bullion, which is afterward paddled, skimmed and ladled into moulds. The slags have the following range of composition: SiO_2, 23 to 32%; FeO, 27 to 42%; CaO, 10 to 24%. The best results were obtained with SiO_2, 28%; FeO, 35%; CaO, 15%; Al_2O_3, 12 to 14%. The average content of the slags was a little less than 1 oz. silver per ton and about 1·5% of lead. The average analysis of the speiss was: Fe, 43·4%; As, 31%; Pb, 13·2%; Ag, 31 oz. per ton; Au, 0·22 oz. per ton, with considerable antimony, copper and sulphur. While this speiss contains considerable metallic values it is not profitable to treat it under existing conditions. The amount produced, however, is small, as less than one-quarter of the arsenic in the ore mixture is reduced to speiss.

Spain.—The production of lead in Spain in 1899, according to the advance report of the Revista Minera, was 184,007 metric tons, of which 162,131 were exported in pigs, bars, etc., 664 in manufactures, 6,212 in ores and the balance 15,000 was consumed in Spain. The quantity of lead ore mined was 334,000 tons. Of the lead exported 68,955 tons were argentiferous and 93,176 tons were soft.

The Lead Markets in 1899.

The condition of the trade during 1899 may be considered, on the whole, as satisfactory, although the fluctuations in prices were very great, exceeding the wide range of the preceding year.

The labor troubles in the Cœur d'Alêne district, Idaho, and in Colorado, interfered considerably with the production, and the resultant shortage at the time was met by the stocks of lead on hand, held by the various refiners. The rapid exhaustion of this reserve caused a rise of price from month to month and great difficulty was encountered in securing prompt deliveries. The shortage of supply in September and October would have undoubtedly carried prices to importation rates had it not been for the control of the American Smelting and Refining Co., whose manipulations held the market firm and steady and prevented a flood of imported lead which would have overstocked the market and caused a decline in price with a subsequent bad effect for some time to come. Consumption during the year exceeded all previous records, especially for the manufacture of shot, sheet and pipe. The use of lead for electrical purposes was far greater than ever before. The white lead industry as well was of goodly proportion. The average price during 1899 was almost equal to the high figures of 1890 and 1891.

The end of the year placed lead in a stronger position than for many years past. There were no important reserve stocks of refined lead and very little was in store at New York, Chicago or St. Louis. At the refining and smelting works the same state of affairs existed, nearly the entire production being shipped from day to day as it became ready for the market. Considering that the business community expects the volume of trade during 1900 to equal if not surpass that of 1899, the outlook is bright for a further healthy development of the lead industry in all its branches. The demand for lead in Europe by consumers was large throughout the year, and was met in part by the exports of Mexican lead, refined here in bond; and in part by the shipment to Great Britain of large quantities of lead from the Broken Hill mines in Australia, which have heretofore marketed most of their product in the East.

New York.—The demand was unusually good during January; values improved and 4·375c. was realized toward the end of the month. Heavy fluctuations in the Eastern markets occurred in February, while in the West the movements were less violent. Quotations ranged from 4·75 to 4·375c. During April and May, the market was quiet, rising gradually until the middle of July. The labor troubles in Idaho and Colorado imparted an upward tendency to the market and prices increased from 4·52c. in July to 4·55c. in August; quotations kept on the increase, 4·575c. for October and 4·60c. for November

being recorded, the year closing with the high price of 4·65c. The possibility of importation disturbing the home market has always threatened the industry, and it is pleasing to record that practically every pound of lead refined in bond in the United States during 1899 was exported; with the result that this industry continues without detriment to the home producer, while it enables our refiners to compete with Europe for the Mexican and British Columbian business.

AVERAGE MONTHLY PRICES OF LEAD IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1895	3·10	3·12	3·12	3·08	3·16	3·25	3·25	3·50	3·85	3·83	3·25	3·22	3·23
1896	3·08	3·19	3·14	3·07	3·08	3·03	2·96	2·78	2·77	2·80	2·96	3·04	2·98
1897	3·04	3·28	3·41	3·32	3·26	3·33	3·72	3·84	4·30	4·00	3·76	3·70	3·58
1898	3·65	3·71	3·72	3·68	3·64	3·82	3·96	4·00	3·99	3·78	3·70	3·76	3·78
1899	4·18	4·49	4·87	4·31	4·44	4·43	4·52	4·57	4·58	4·58	4·58	4·64	4·47

London.—The year began with a marked improvement, and consumers were buying freely at the parity of about £13 7s. 6d. for soft foreign and £13 10s. English. Heavy arrivals brought good prices at first, while speculators bought good quantities for forward delivery. These arrivals, however, soon began to weigh upon the market, and from £13 15s. prices dropped again to the above level. The large speculative demand early in February advanced the price quickly to £15 10s., followed by further heavy arrivals, which were pressed for sale, unsettled matters, and £14 5s. and £14 10s. ruled for soft foreign and English lead, respectively. A further decline brought more demand from consumers, and the value fell, early in March, to £13 17s. 6d. The tone then improved in the comparative absence of sellers, and £14 5s. to £14 7s. 6d. were paid. The American also held for higher values and £14 10s. for soft foreign ruled during the first half of April. The French Government bought good quantities. At this level, however, speculators began to realize their holdings and the market suffered to the extent of 5s. or so. Business during May was very small indeed and £14 2s. 6d., or less, was pretty generally accepted. The settlement of a strike in the building trade brought a firmer feeling in June and consumers bought quite freely at about £14 5s. A scarcity in the London supplies brought a further advance, and prices were quoted firm at £14 7s. 6d. to £14 10s.

A strike among Colorado producers and small arrivals in this country contributed further to the firmness of the market in July. Soft foreign brought £14 10s., and still better prices were obtained for immediate delivery. A continued absence of sellers in August further strengthened the position and consumers' business became quite active at the higher values; £15 was reached for English and £14 18s. 9d. for foreign. The arrivals continued on a small scale in September and consumers, being still badly covered, had reluctantly to pay still higher values. By the end of the month there was quite a famine, owing partly to the difficulty of getting deliveries from Spain, and spot lead brought about £16, while forward lead was nearly 10s. lower.

October brought no relief, and even £17 was paid for spot lead, but this price scared off consumers somewhat, who confined their purchases to immediate requirements. Continental sellers were, however, attracted by our figures, and

their freer offerings brought about a more normal state of affairs. The market remained very firm throughout the month and again reached £16 17s. 6d. Renewed demand from consumers in November, and a continued scarcity of spot material, caused a further advance to £17 12s. 6d., but the famine was then temporarily relieved by some heavy arrivals and spot was quoted at £17 5s. Arrival lead brought £16 10s. to £17 and English £17 to £17 5s. At the end of the month the scarcity was felt more keenly than ever and the tendency was strong and upward. Consumers now bought very freely, with apparent willingness to pay the higher values, especially for early delivery. The Government was an important buyer. After a temporary reaction, prices were firmer again in December at £17 2s. 6d. to £17 12s. 6d. for foreign lead; £17 15s. was then paid for spot, but owing to the near approach of the Christmas holidays the demand fell off, cold weather restricted consumption, and prices became easier at about £16 7s. 6d. for soft foreign, with English brands at £16 12s. 6d. to £16 15s.

UNITED KINGDOM: IMPORTS AND EXPORTS OF LEAD. (IN TONS OF 2,240 LB.)

	1899.	1898.	1897.	1896.	1895.	1894.	1893.	1892.	1891.	1890.	1889.
Imports.....	198,377	194,479	167,442	167,594	162,924	161,861	191,174	182,782	169,724	158,649	145,451
Exports.....	40,281	38,075	40,503	41,247	41,666	47,060	48,935	58,101	48,726	55,536	52,153

WHITE LEAD, RED LEAD, LITHARGE AND ORANGE MINERAL.

The following tables give the production in the United States and imports of these substances reduced to a basis of dry white lead, red lead, etc., excluding the oil contents of any marketed as ground in oil:

PRODUCTION OF RED LEAD, WHITE LEAD, LITHARGE AND ORANGE MINERAL.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897.....	7,798	$744,709	105,804	$9,522,360	8,591	$773,190	477	$76,320
1898.....	9,160	916,000	98,172	9,391,738	7,460	710,192	653	88,937
1899.....	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200

UNITED STATES: IMPORTS OF RED LEAD, WHITE LEAD, LITHARGE AND ORANGE MINERAL.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1895.....	1,764,274	$53,139	1,897,892	$79,887	97,667	$2,812	1,699,867	$66,492
1896.....	1,543,262	47,450	1,183,538	52,409	51,050	1,615	1,359,651	51,027
1897.....	1,386,070	46,992	1,101,829	48,988	60,984	1,931	1,486,042	67,549
1898.....	682,449	25,780	506,739	24,334	56,417	2,021	795,116	37,745
1899.....	1,021,573	43,812	584,409	30,211	55,127	3 614	1,141,887	58,242

Eighteen hundred and ninety-nine was one of the most prosperous years in the lead trade. At the end of 1898 prices were firm and the stocks in the hands of dealers extremely low. The year closed with white lead and oil at 5·5c. per lb. Business was active in the early months of the year and corroders were behind in meeting their contracts. The prices of all raw material had advanced so an

increased price in dry lead and lead in oil was momentarily expected. This advance, however, did not come until March, when prices were increased 0·25c. per lb. The best known makers' brands of foreign lead were also similarly advanced. In anticipation of the increased prices in lead products a movement to corner the market had set in and it seems that the corroders and the National Lead Co. met this attempt by pursuing Fabian tactics, notwithstanding the rise in all crude material. Stocks continued light and in May a great demand arose. It was estimated that the painting done during the early part of the year was the greatest in amount for many years past. In July and August, which are usually dull months in the trade, retail dealers throughout the country reported excellent business, and corroders were still behind in their contracts. In August the long expected rise came, the lead products being advanced 0·25c. to correspond with the enhanced price of the raw metal, as well as the increased demand and consumption. There was no change at this time in the price of foreign brands, but early in September the agents advanced prices 0·25c., making it 8·75 to 9c. for the old established brands. The latter part of September the demand fell off slightly. Though it was expected the National Lead Co. would increase the prices again, such action was not taken. In November the agents of foreign brands increased prices from 9 to 9·25c. for best known brands. The latter part of December the National Lead Co. made the third increase of the year to 6·25c., at which figure the market was held firm at the end of the year. Foreign lead also advanced to 9·375 and 9·625c.

Method of Reducing Lead Oxide.—G. Bishof* has patented an apparatus for reducing litharge to lead sub-oxide in connection with the manufacture of paint. A vertical, cylindrical vessel inclosed in an outer vessel of larger diameter is employed. In the annular space between the two, gas is burned so as to maintain the temperature in the vessel at 250 to 300°. Through the closed top of the vessel a feed hopper and shoot leads down into the inner vessel which has in the bottom an inlet for water-gas, supplied from a compressor. At the top there is an outlet pipe leading to a gas holder. The discharge passage is at the bottom and is water-jacketed. The inner vessel is supplied with an agitating spindle. The operation is as follows: After the vessel is heated and the spindle started in revolution, litharge is fed in through the hopper and is raised by the revolving blades of the spindle against the mouth of the shoot so as to seal it against the escape of gas. Water gas is forced into the material and gives a portion of its hydrogen to some of the oxygen of the litharge. The resultant steam, together with the unspent gas, passes into a gas holder, where it is freed from water and can be used again. The material, thus reduced to sub-oxide, is cooled as it descends through the tapering water-jacketed outlet and is discharged periodically.

Manufacture of White Lead.—In Germany in 1899, the Elektrochemische Industrie Gesellschaft, at Cologne on the Rhine, produced electrolytically 2,500 tons of white lead under the Luckow patent. The process is based on the use of two dissolved salts, the anion of the one dissolving the anode while the anion of the other precipitates the dissolved metal. The first salt is used in a state of

* English Patent No. 13,202, June 13, 1898.

solution corresponding to its minimum resistance, while the solution of the salt which precipitates the dissolved metal is calculated according to the ratio of the electro-chemical equivalent and the force of the current. Thus a 7% sodium chlorate solution and a 0·011% sodium carbonate solution are the most convenient for the two salts. The anodes are of crude lead and the cathodes of soft lead. A single tank contains 10 anodes and 11 cathodes, each having an area of 20 to 30 sq. cm. and spaced 0·15 m. apart. The current density is 0·5 ampère per sq. cm. and the voltage is from 1·10 to 1·30 at the temperature 15 to 17°C. A stream of CO_2 is passed into the electrolyte to aid its circulation and to regenerate the principal salt.

Herman C. Walterock* has patented a process of manufacturing white lead electrolytically by using lead anodes in an electrolyte capable of dissolving lead and containing an alkaline bi-carbonate. It is necessary to maintain the temperature below 25°C. The mixed electrolyte and precipitate is continuously withdrawn from the vat and the white lead is removed by filtration. Carbon dioxide is passed through the filtrate in order to regenerate it and is returned to the electrolytic vat.

M. Ribau uses a solution of sodium carbonate and sodium chlorate with lead anodes and cathodes. Lead chlorate is first produced which, reacting on the sodium carbonate, becomes lead hydro-carbonate. A mechanical device is used to remove continuously the white lead adhering to the metallic lead sheets.

Composition of White Lead.—E. Lenoble† has analysed several samples of commercial white lead. The hygroscopic moisture was determined by drying at 100°C.; the carbon dioxide as loss of weight after addition of nitric acid; the lead oxide by ignition of 1 g. of the sample; the acetic acid by distillation of 1 g. with tartaric acid; the combined water by difference, and the density by use of toluene. The formula $Pb(OH)_2.2PbCO_3$ was adopted, and, in the author's opinion appears to represent white leads of the best kind. These appear to be the more suitable for use, the specifically lighter they are.

Relative Efficiency of Lead and Zinc Paints.—G. R. Henderson made a series of exposure tests to determine the relative efficiency of these paints.

The materials painted were tin, galvanized iron, sheet iron poplar, white pine and yellow pine, and the experiments showed that the lead priming and zinc coating were generally good for tin, galvanized iron, poplar and white pine. Sheet iron showed up best with both coats of mixed paints. Yellow pine appeared best with the first coat of lead and the second coat of lead and zinc mixed. Comparing the materials which were painted, we find that generally poplar retains the paint better than white pine, and would, therefore, be preferred for siding on buildings, etc. Yellow pine seems to be the worst of all for this purpose. Black iron, as a whole, seems to retain the paint better than either tin or galvanized iron.

* United States Patent No. 631,889. † *Annales Chemie Analytique Appliqué*, 4, p. 118.

Recent Improvements in Lead Smelting.

By H. O. Hofman.

B. & H. Ztg., Berg- und Hüttenmannische Zeitung; *Bull. Soc. Enc.*, Bulletin de la Société d'Encouragement pour l'Industrie Nationale; *Ch. Ztg.*, Chemiker Zeitung; *Dingl. Pol.*, Dingler's Polytechnisches Journal; *E. & M. J.*, Engineering and Mining Journal; *Freib. J.*, Freiberger Jahrbuch; *Geol. Surv. Ia.*, Geological Survey of Iowa; *I. A.*, Iron Age; *Jernk.*, Jernkontorets Annaler; *J. Frankl. I.*, Journal of the Franklin Institute; *Jl. Soc. Chem. Ind.*, Journal of the Society of Chemical Industry; *Met.*, The Metallographist; *M. I.*, The Mineral Industry; *M. S. P.*, The Mining and Scientific Press; *O. Z.*, Oesterreichische Zeitung für Berg- und Hüttenwesen; *S. M. Q.*, Columbia School of Mines Quarterly; *T. Q.*, Technology Quarterly; *Trans. A. I. M. E.*, Transactions of the American Institute of Mining Engineers.

INTRODUCTORY.

Lead-Tin Alloys.—Sperry* investigated the mechanical properties of the alloys of tin and lead. The different alloys were prepared by melting together Banca tin and desilverized lead in an iron ladle at a low temperature and stirring with a pine stick, and then poured into cylindrical moulds giving bars 1·5 in. diameter and 12 in. long. These were turned to give a perfectly cylindrical surface. The cylinders for the compression tests were prepared in the same way, but reduced to 1·25 in. diameter and 3 in. in length. No difficulty was encountered in making the test for tension. For the compression tests, the shortening of the

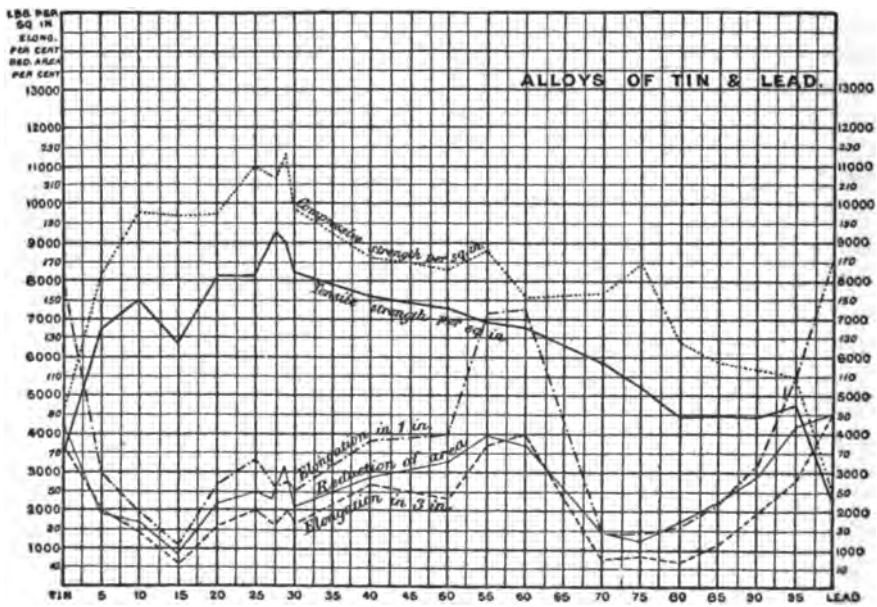


Fig. 1.—Curves Showing the Mechanical Properties of Alloys of Tin and Lead.

test piece 0·1 in. for each part of a load added was noted, and when the total equalled 1 in., the load was assumed to be ultimate. Such an arbitrary end-point had to be adopted because the test piece would have bulged if the load had been further increased. The shrinkage was determined by casting the melted alloy at the lowest possible temperature in a mould which at 20°C. would give a bar 0·5×0·75×12 in. The difference in length between the cast bar and 12 in. gave the shrinkage. The data obtained are sufficiently accurate for

* *Jl. Soc. Chem. Ind.*, February, 1899, pp. 113–116.

practical purposes. In making the tests, a uniform speed of 0·6875 in. per minute was settled upon after some preliminary experimenting.

In the subjoined table and preceding diagram (Fig. 1) are given the results of the experiments:

SUMMARY OF RESULTS ON ALLOYS OF TIN AND LEAD.

Experiment No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.
Composition:											
Tin (%)	100	95	90	85	80	75	72·5	71	70	60	50
Lead (%)	0·00	5	19	15	20	25	27·5	29	30	40	50
Diameter (in.)	1·382	1·436	1·385	1·415	1·465	1·346	1·295	1·385	1·441	1·445	1·429
Breaking strain (lb.)	5,360	10,770	11,280	10,050	13,700	11,530	12,160	12,590	13,400	12,430	11,600
Breaking strain (lb. per sq. in.)	3,573	6,652	7,490	6,393	8,130	8,108	9,233	8,999	8,220	7,540	7,236
Elongation in 1 in. (%)	158	60	38	22	52	66	52	56	50	76	80
Elongation in 8 in. (%)	74	40	27	12	31	40	32	39	32	53	46
Reduction of area (%)	84	39	34	19	43	50	46	62	42	57	64
Diameter (in.)	1·220	1·264	1·279	1·345	1·255	1·265	1·249	1·288	1·270	1·260	1·220
Load necessary to produce a reduction of 0·10 in. in 1 in. (lb.)	5,500	10,250	12,500	13,750	12,500	13,750	13,200	13,600	12,500	10,660	9,660
Load pr. sq. in. necess'y to pr'duce a reduction of 0·10 in. in 1 in. (lb.)	4,709	8,17·	9,733	9,688	9,700	10,947	10,775	11,305	9,873	8,560	8,275
Shrinkage (in. per ft.)	0·08	0·06	0·05	0·07	0·65	0·07	0·06	0·06	0·06	0·06	0·08

Experiment No.	12.	13.	14.	15.	16.	17.	18.	19.	20.	21.
Composition:										
Tin (%)	45	40	30	25	20	15	15	10	5	.....
Lead (%)	55	60	70	75	80	85	85	90	95	100
Diameter (in.)	1·370	1·411	1·448	1·364	1·450	1·441	1·442	1·442	1·427	1·400
Breaking strain (lb.)	10,180	10,600	9,670	7,600	7,350	7,390	7,300	7,270	7,660	3,280
Breaking strain (lb. per sq. in.)	6,906	6,781	5,875	5,200	4,451	4,533	4,469	4,450	4,790	2,098
Elongation in 1 in. (%)	145	144	29	29	32	39	48	64	110	170
Elongation in 8 in. (%)	73	80	14	16	13	20	23	38	55	90
Reduction of area (%)	79	75	29	27	34	38	49	59	84	90
Diameter (in.)	1·296	1·274	1·257	1·234	1·270	1·266	1·275	1·259	1·231	1·217
Load necessary to produce a reduction of 0·10 in. in 1 in. (lb.)	10,500	9,500	9,500	10,000	8,000	7,380	7,500	7,000	6,500	2,800
Load pr. sq. in. necess'y to pr'duce a reduction of 0·10 in. in 1 in. (lb.)	8,758	7,456	7,660	8,368	6,318	5,830	5,877	5,620	5,460	2,409
Shrinkage (in. per ft.)	0·06	0·06	0·06	0·05	0·06	0·06	0·06	0·08	0·1:	0·12

Lead-Tin and Lead-Antimony Alloys.—Richards* devised a method for testing qhickly alloys of lead and tin, and of lead and antimony, which consists in casting the alloy to be tested into the same form of bullet as a series of standard alloys, and finding the one of the series which corresponds in weight with that of the alloy in question. Lead can be alloyed with all proportions of antimony up to 24%, and this forms the highest percentage of antimony that can be tested. Lead with 2% Sb and less is heavier than pure lead, but when poured on a flat plate its surface, when cooled, has a fine white crystalline appearance. The regular set of alloys is supplemented by another containing 2, 1·9, 1·8% Sb down to pure lead. A second set of standards, beginning with pure lead, then taking 99% Pb and 1% Sn, 98% Pb and 2% Sn, and so on, up to pure tin, was prepared to test lead-tin alloys.

Antifriction Alloys.—Charpy,† in a study of white antifriction alloys, discusses, among others, the alloys of lead and antimony; of lead, tin, and bismuth; of lead, tin, and antimony; and of copper, tin and lead.

Lead Ores.—From a description by Peterson‡ of the lead-zinc deposits of

* *J. Frankl. I.*, May, 1899, pp. 398-400; *I. A.*, May 11, 1899.

† *Bull. Soc. Enc.*, June, 1898; *Met.*, January, 1899, p. 9.

‡ *Jernk.*, through *B. & H. Ztg.*, 1899, p. 487.

Bleiberg, Carinthia, the following data are taken: The ore as mined assays 6 to 7% Pb; the concentration works furnish three finished products: galena, with 73 to 74% Pb; blende, with 46% Zn; smithsonite, with 42% Zn. An analysis of galena concentrates of the Friedrich mine showed Pb, 77·183%; Zn, 1·550%; Fe_2O_3, 0·450%; As, 0·099%; S, 13·730%; $CaCO_3$, 2·350%; $MgCO_3$, 0·420%; Insoluble, 3·200%.

Assay Furnaces.—Hoskins & Co., Chicago,* have brought into the market an assay furnace, which is a combination of a crucible and a muffle furnace and is more suited for a mine or mill than for a smelting plant.

Braun & Co., Los Angeles, Cal.,† have brought out a combination of crucible and muffle assay furnace, called "L and C." A gasolene flame entering on one side first heats the crucibles and then plays around the muffle. In order to insure draught through the muffle, the opening at the back is connected by a fire clay tube and a sheet-iron elbow with the stovepipe. The standard size takes a muffle 6×10 in., has a melting compartment 7×7×7 in. which holds four crucibles each of a capacity of 2 assay tons of pulp. The weight of the furnace complete is 80 lb.

The same firm announces the Calkins cupel machines‡ which consist of a hopper, feeding the moistened bone-ash; a compound lever with plunger and two disks, the upper one of which contains the openings in which the cupel is compressed, while the lower one with a single opening larger than the cupel, serves as a support in compressing the cupel and as a discharge for the finished cupel when its opening has been brought underneath one of those in the upper disk.

Proof Gold and Silver.—Whitehead§ describes the preparation of proof gold and proof silver for assay purposes at the United States mint. The method for gold is: Roll out ten or less ounces fine gold into a ribbon, place in a flask, add the required amount of hydrochloric acid, and then nitric acid in small amounts until the gold is dissolved. Transfer the flask to a sand-bath, evaporate the solution until all the nitric acid is removed, which may require the addition of more hydrochloric acid; add a few c.c. of hydrobromic acid, dilute with water until the solution contains not more than 1 oz. of gold per gallon; agitate thoroughly, allow the silver salt to settle, siphon off the clear supernatant gold solution through a double filter, without disturbing the silver salt on the bottom; (do not wash the filter). Precipitate the gold with sulphurous acid; filter and wash by decantation. Redissolve the gold and treat as before, but precipitate with oxalic acid in the form of a warm concentrated solution. Repeat the solution and precipitation, washing the gold successively with water, with nitric acid, with water, with hydrochloric acid, and finally again with water. Dry the gold, mix with 20% its weight of flux (3 parts of borax glass to 1 part of pure niter), melt in a porcelain crucible, pour into a chalk-lined mould, and free the bar from adhering impurities by brushing, and then boiling in hydrochloric acid. Dry the bar; cut it in pieces and pass them between

* *E. & M. J.*, April 1, 1899.
† *E. & M. J.*, April 29, 1899.
‡ *M. S. P.*, Oct. 21, 1899; *E. & M. J.*, Nov. 25, 1899; United States Patent No. 637,081, Nov. 11, 1899.
§ *E. & M. J.*, Dec. 30, 1899.

carefully cleaned rolls; boil the ribbons in acid and complete the operation by washing and heating to redness in a muffle.

The method for silver is: Electrolyze fine silver (999 fine) using an electrolyte of silver nitrate with 1% free nitric acid, and wrapping the anode bar in filter paper and placing it in a cotton bag. Collect the crystals of pure silver, deposited on the cathode; wash with water, dry and melt, first with the same flux as given above, then without flux, stirring the melted silver with a piece of dry wood until all the absorbed oxygen has been removed; pour into a chalk-lined mould, clean the bar with brush and dilute sulphuric acid, and roll out into ribbon form.

Priwoznik,* after condemning the method advocated by Pack (THE MINERAL INDUSTRY, Vol. VII., p. 452) for producing chemically pure silver (also condemned by Whitehead)† refers to two other methods by Pfeiffer‡ and Kuhn,§ both of which are modifications, though not improvements on the author's electrolytic method of reducing silver chloride, published 20 years ago,‖ in which a battery with porous cell is used. The jar contains the pure silver chloride moistened with dilute hydrochloric acid, a silver electrode, the porous cell, a zinc electrode and dilute acid.

Assay of Telluride Ores.—Lodge¶ made some experiments in the assay of rich telluride gold ores by the scorification and crucible methods. The data are given in the two subjoined tables. He concludes that satisfactory results can be obtained by either method as long as the ore has been ground to a sufficient degree of fineness to have the gold mineral uniformly divided through the gangue of the sample. For rich ores, he believes the sample should be ground and passed through a 140-mesh sieve, or finer if possible.

Assay of Gold-Silver-Platinum Alloys and Ores.—Schiffner** has investigated the assay of gold-silver-platinum alloys and prepared jeweler sweeps which are frequently shipped to the Freiberg smelting works for treatment. Four tables contain the results of the long series of assays made to test the accuracy of the different methods tried. The following method is the final outcome of the work: Alloys are cupelled direct with the necessary lead; sweeps are first scorified and then cupelled. The cupellation is to be carried on at a temperature sufficiently high to prevent the formation of any feather litharge. The resulting buttons brighten as in the ordinary assay, or they remain dull and flatten out, which is the case if the alloy is rich in platinum.

If the buttons are bright, they are weighed as representing a pure gold-silver-platinum alloy. If they contain 4 parts silver to 1 part gold-platinum, they are treated three times with sulphuric acid of 66°B. and then boiled for half an hour with water; if not, they are first re-cupelled with the necessary silver. The residue from the acid treatment is gold-platinum, the difference in weight between gold-silver-platinum and gold-platinum gives the silver. The gold-platinum residue is melted down with from 12 to 20 times its weight of silver, boiled three times with nitric acid (sp. gr. 1·2), once with water, and the

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* *O. Z.*, 1899, p. 233. † *E. & M. J.*, Dec. 30, 1899, p. 785. ‡ *Ch. Ztg.*, 1898, No. 76, p 775.
§ *Ch. Ztg.*, 1898, No. 82, p. 868. ‖ *Dingl. Pol.*, 1880, CCXXXV., p. 117. ¶ *T. Q.*, 1899, 12, 171.
** *Freib. J.*, 1899, A., p. 70.

residual gold weighed. If this does not show the pure gold color, the whole operation is repeated. The difference in weight between platinum-gold and gold gives the platinum.

SCORIFICATION METHOD.

Charges.	No. of Experiments.	Wght. of Lead Button.	Wght. after Scorifing.	Wght. after Third Rescorification.	Wght. of Gold after Parting. (Grams.)	Gold found in Slag.	Gold found in Cupel.	Total Gold.	Ounces per Ton.	Per Cent. of Total Gold Found in Slag.	Per Cent. of Total Gold Found in Cupel.
I. Ore, 0·1 A. T.	1*a*	30	20	.....	0·0065.	0·00005	0·00004	0·00631	63 1/10	0·79	0·63
I. Granulated lead, 60 g.	2*b*	30	Cupe	lled.	0·00648	.....	.....	.....	64 1/10	.....	.....
I. Half mixed with ore, and half placed on top).	3*c*	31	Cupe	lled.	0·00690	Trace.	Trace.	0·00690	69	0·00	0·00
Borax glass cover.											
II. Ore, 0·1 A. T.	1*e*	34	32	.....	0·00686	0·00004	None.	0·00690	69	0·58	None.
II. Granulated lead, 45 g., (half mixed with ore..	2*f*	25	Cupe	lled.	0·00648	0·00005	0·00005	0·00658	65 8/10	0·76	0·76
II. PbO cover, 10 g.	3*g*	25	Cupe	lled.	0·00650	0·00010	0·00006	0·00666	66 6/10	1·50	0·90
Borax glass, 0·3 g. as a cover											
III. Ore, 0·1 A. T.	1*h*	48	37	20	0·00684	0·00005	None.	0·00689	68 9/10	0·71	None.
III. Granulated lead, 60 g.	2*j*	49	25	.....	0·00647	Trace.	0·00005	0·00652	65 2/10	0·00	0·77
III. PbO 1 A. T.	.....	...	.....	.....	.....	.....	.....	.....	.....	.....	.....

Assays *a*, *e* and *h* were all made at the same time, and under conditions as nearly alike as possible.
Assays *b*, *f* and *j* were all made at the same time, and under like conditions.

CRUCIBLE METHOD.

Charges.	No. of Experiments.	Wght. of Lead.	Wght. after Scorifying Once.	Wght after Scorifying Twice.	Wght. of Gold Obtained in Grams.	Wght. of Gold Found in Slag.	Wght. of Gold Found in Cupel.	Total Gold Accounted for.	Ounces per Ton.	Per Cent. of Total Gold Found in Slag.	Per Cent. of Total Gold Found in Cupel.
I. Mixture. Ore, 0·5 A. T.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
I. Mixture. Soda, 60 g.	1	30	.....	.....	0·08391	0·00007	0·00008	0·08406	68·12	0·21	0·23
I. Mixture. PbO, 120 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
I. Mixture. Argols, 1 g.	2	27	.....	.....	0·08417	0·00019	Lost.	0·08436	68·72	0·56	.....
Borax glass cover.											
II. Mixture. Ore, 0·5 A. T.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
II. Mixture. Soda, 20 g.	1	80	47	21	0·08463	None.	None.	0·08463	69·26	None.	None.
II. Mixture. PbO, 80 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
II. Mixture. Gran. Pb, 50 g.	2	75	51	25	0·08387	0·00008	0·00012	.....	.....	0·24	0·35
II. Mixture. Argols, 1 g.	.....	.....	.....	.....	.....	.....	.....	0·08407	68·14	.....	.....
Cover of salt.											
III. Mixture. Ore, 0·5 A. T.	1	29	.....	.....	0·08393	Trace.	0·00008	0·08401	68·02	.....	0·24
III. Mixture. Soda, 40 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
III. Mixture. PbO, 60 g.	2	25	.....	.....	0·08414	0·00005	0·00014	0·08433	68·66	0·15	0·41
III. Mixture. Borax glass, 15 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
III. Mixture. Argols, 1 g.	3	25	.....	.....	0·08401	Trace.	0·00016	0·08417	68·34	.....	0·46
Cover of salt.											
IV. Mixture. Ore, 1 A. T.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
IV. Mixture. Soda, 60 g.	1	25	.....	.....	0·06698	0·00009	0·00005	0·06712	67·12	0·13	0·07
IV. Mixture. Borax glass, 15 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
IV. Mixture. PbO, 60 g.	2	28	.....	.....	0·06820	0·00011	0·00015	0·06846	68·46	0·16	0·22
IV. Mixture. Niter, 1 g.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cover of salt.											

If the flattened buttons obtained in the cupellation were rough, dull, or did not brighten (*i.e.*, containing over 6% Pt), they are treated as above, without weighing the gold-platinum residue, and the gold and platinum only are determined.

The assay for silver is made from a separate sample. The alloy is melted down in a crucible with four times its weight of lead. Sweeps are scorified with lead to the point where there remain four parts of lead with one part of

alloy. The resulting lead buttons are dissolved in nitric acid, silver is precipitated from the solution with hydrochloric acid, and the silver chloride filtered. The gold-platinum residue is dissolved in nitro-hydrochloric acid and any silver chloride formed is filtered and added to the first filter. The filters are burned and the ashes scorified with lead and cupelled. Should iridium (osmium) be present, which is rarely the case, part of it will remain with the gold-platinum in the form of fine, iron-black flitters and part will be decanted with the solution. To determine iridium, a separate set of samples is scorified and cupelled; the resultant buttons are dissolved in sulphuric acid, the residue treated with nitro-hydrochloric acid, and the remaining iridium (osmium) weighed, after removing any silver-chloride with ammonia.

Assay of Iron in Lead Ores.—Jones* challenges the statement made by the reviewer† that the Zimmermann-Reinhart method of titrating iron with potassium permanganate in a hydrochloric acid solution generally gave accurate results, by restricting it to rich iron ores with 65 to 75% Fe free from impurities practically always present in the iron ores treated by lead smelters. The reviewer‡ acknowledges his error.

Reduction of Costs.—Kirchoff,§ in a presidential address on "A Decade of Progress in Reducing Costs," gives the following figures, which show the reduction of cost in smelting and refining for a period of 11 years:

Item.	1887.	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.
Cost of smelting and refining.....	100·0	98·0	97·0	95·0	93·0	91·0	88·0	85·0	79·0	74·0	65·0
Fuel consumption per ton of ore smelted	100·0	97·5	97·1	95·4	95·2	92·6	92·1	91·0	85·6	79·0	71·2

The reduction in cost is due to the introduction of labor-saving appliances, and to the use of larger furnaces; that in fuel, to the larger furnaces, and more intelligent methods of firing.

SMELTING OF LEAD ORES.‖

In the Ore Hearth.—Leonard¶ has a note on early lead smelting in Iowa, in which is described and illustrated a primitive form of hearth-furnace.

In the Blast Furnaces: Roasting of Ores.—Ernst** discusses the new method of roasting lead ores, patented by Huntington and Heberlein (THE MINERAL INDUSTRY, Vol. VII., p. 452), which is in operation at the lead works of Pertusola, Italy. There are two stages in the process, sulphatizing and finishing. If galena ore, mixed with 6 to 15% burnt lime, varying with the percentage of sulphur, is given a partial oxidizing roast in a reverberatory furnace at a temperature reaching 700°C. and not falling below 450°C., so as to leave at least 5% sulphide-sulphur undecomposed, from 25 to 35% will be converted into sulphate, only 2% into oxide, no metallic lead will be set free, and the loss of lead by volatilization will be practically nothing. If the galena, on the other hand, be treated in the same way without the addition of lime, sulphate and oxide will be formed in the proportion of 3 to 4, some metallic lead will be set free

* *E. & M. J.*, Sept. 9, 1899. † *M. I.*, VII., p. 451. ‡ *E. & M. J.*, Oct. 7, 1899. § *Trans. A. I. M. E.*, Feb. 1899.
‖ See THE MINERAL INDUSTRY, Vol. VII. ¶ *Geol. Surv. Ia.*, VI., p. 16. ** *O. Z.*, 1899, p. 619.

and the loss by volatilization will be considerable. The sulphatization is believed to be caused either by a problematic dioxide of lime CaO_2 or by catalytic effect of lime, *viz.:*

$$4CaO_2+PbS=PbSO_4+4CaO \text{ or}$$
$$4CaO,O+PbS=PbSO_4+4CaO.$$

If the hot pulverulent sulphate is now transferred to a pear-shaped vessel and air under a pressure of 0·75 to 1·25 in. of quicksilver forced through it, the temperature will at first rise on account of the continued oxidation of the remaining sulphide; but now lead sulphide will react upon lead sulphate giving sulphur dioxide, and in the presence of the abundance of air, lead oxide, instead of metallic lead.

$$PbS+3PbSO_4=4PbO+4SO_2.$$
$$PbS+PbSO_4+2O=2PbO+2SO_2.$$

The mass becomes pasty, gives off concentrated sulphur dioxide and hardens when the reaction is finished. Most of the lime has been converted into sulphate.

The process is also successful in the presence of associated sulphide minerals when the sulphur dioxide can be utilized for manufacturing sulphuric acid. With this new method of desulphurizing, the works at Pertusola require a single blast furnace instead of three or four to treat the ores mined; they operate with 50% less fuel, labor and cost for tools than with the former roasting and reduction method and have a greater yield in lead as well as in silver.

Roasting Furnaces.—The Jackling straight-line mechanical roasting furnace* has given satisfaction at the De Lamar cyanide works, Mercur, Utah. In principle it resembles somewhat the Brown furnaces, with the exception that the roasted ore on leaving the furnace is raised by elevators onto an upper cooling floor of corrugated iron, over which it is moved by the rakes on their return trip to the feed-end of the furnace. While traveling over the cooling floor, the cooling is assisted by spraying with water. This furnace, with a roasting hearth 12×120 ft. roasts in 24 hours 70 tons base ore with 2 to 5% S and 0·95 to 2·5% As, reducing the arsenic to 0·10 to 0·15%; the consumption of coal during this time was 7 tons. The ore passes through the furnace in eight hours.

A number of patents have been granted for improvements in standard roasting furnaces and for new designs.

Holthoff† has improved his mechanical single-hearth furnace with cooling hearth. Ropp‡ has improved some minor details in his mechanical straight-line roasting furnace. Keiper§ constructed a mechanical reverberatory roasting furnace with a circular horizontal rotary hearth and vertical central shaft similar to that of Brunton, used formerly for roasting tin concentrates. The furnace of Hayes‖ is similar.

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* *E. & M. J.*, Dec. 30, 1899.

† United States Patent No. 620,132, Feb. 28, 1899.

‡ United States Patent No. 634,247, Oct. 3, 1899.

§ United States Patent No. 625,336, May 23, 1899, and No. 630,510, Aug. 8, 1899.

‖ United States Patent No. 637,864, Nov. 28, 1899

Stewart* proposes to improve the Brückner cylinder roasting furnace by placing a boiler on the roof of the fire-box.

Keller† finds it an advantage in his multiple hearth furnace to support the free end of the stirring arm in a manner similar to that of Wethey.

Devereux‡ constructed a multiple hearth furnace with stirring device similar to that used by H. F. Brown.

C. M. Allen§ adapted his two-hearth mechanical roaster to permit the roof being used as a drying floor.

Wright patented‖ a mechanical muffle roasting furnace similar in principle to that of McDougal or Herreshoff.

Hutson devised some improvements¶ in the rake used in the McDougal type of furnace.

Richards,** of Mount Morgan, Queensland, constructed a muffle furnace with muffles one above the other, having openings at alternate opposite ends for transferring the ore, and for the passage of the gases; and openings at the sides for admitting steam.

Two other patents for inclined revolving roasting chambers have been taken out by Clark†† and Boulton.‡‡

Argall patented§§ a rotary ore cooler to be used in connection with his multi-tubular mechanical reverberatory roasting furnace. Another ore cooler to be attached to a Pierce turret furnace, although not exclusively so, is that by Rothwell.‖‖

The Blast Furnace.—Kochinke¶¶ has published an interesting illustrated paper on the history of the blast furnaces of Freiberg, Saxony, starting with the year 1160, when the Freiberg mines were first opened, up to present times. At the Halsbrücken works a blast furnace with a double row of tuyeres similar to the one of Pertusola, Italy, is in operation; the lower row has 12 tuyeres 29·5 in. above the bottom of the crucible, the upper row also 12 tuyeres 4 in. higher up.

Kloz*** constructed a blast furnace of peculiar form for the smelting of the refractory ores of the Pulaiayo mine, Bolivia, at Playa Blanca, near Antofagasta. An average analysis of the ore shows SiO_2, 20%; Fe, 20%; S, 30%; Zn, 18%; Pb, 7%; Cu, 1%; Sb, 1%; besides small amounts Sn and As, but no Bi. There occurs in this sulphide ore some oxide ore with SiO_2 50%, and Zn 5%, which is not separated. The furnace excepting the top and the bottom, consists of sections of cast-iron plates water-cooled by 1-in. wrought-iron pipes around which the iron was cast. The shaft-jackets rest

* United States Patent No. 623,273, Sept. 19. 1899.
† United States Patent No. 624,048, May 22, 1899.
‡ United States Patent No. 628,077, July 4, 1899.
§ United States Patent No. 633,777, Sept. 26, 1899.
‖ United States Patent No. 629,023, July 18, 1899.
¶ United States Patent No. 626,281, June 6, 1899.
** United States Patent No. 630,108, Aug. 1, 1899.
†† United States Patent No. 620,837, March 14, 1899
‡‡ United States Patent No. 635,410, Oct. 24, 1899.
§§ United States Patent Nos. 622,230 and 623,231, April 18, 1899
‖‖ *E. & M. J.*, July 29, 1899.
¶¶ *Freib. J.*, 1899, pp. 107-138.
*** *E. & M. J.*, Sept. 2, 1899.

upon pillars and the bosh and tuyere-jackets are suspended from them, leaving the crucible entirely free. This is a cast-iron box which is brought into position on a car and raised by jack-screws. It consists of a bottom plate, with rim 2 in. thick, and six side plates bolted together, and needs to be lined

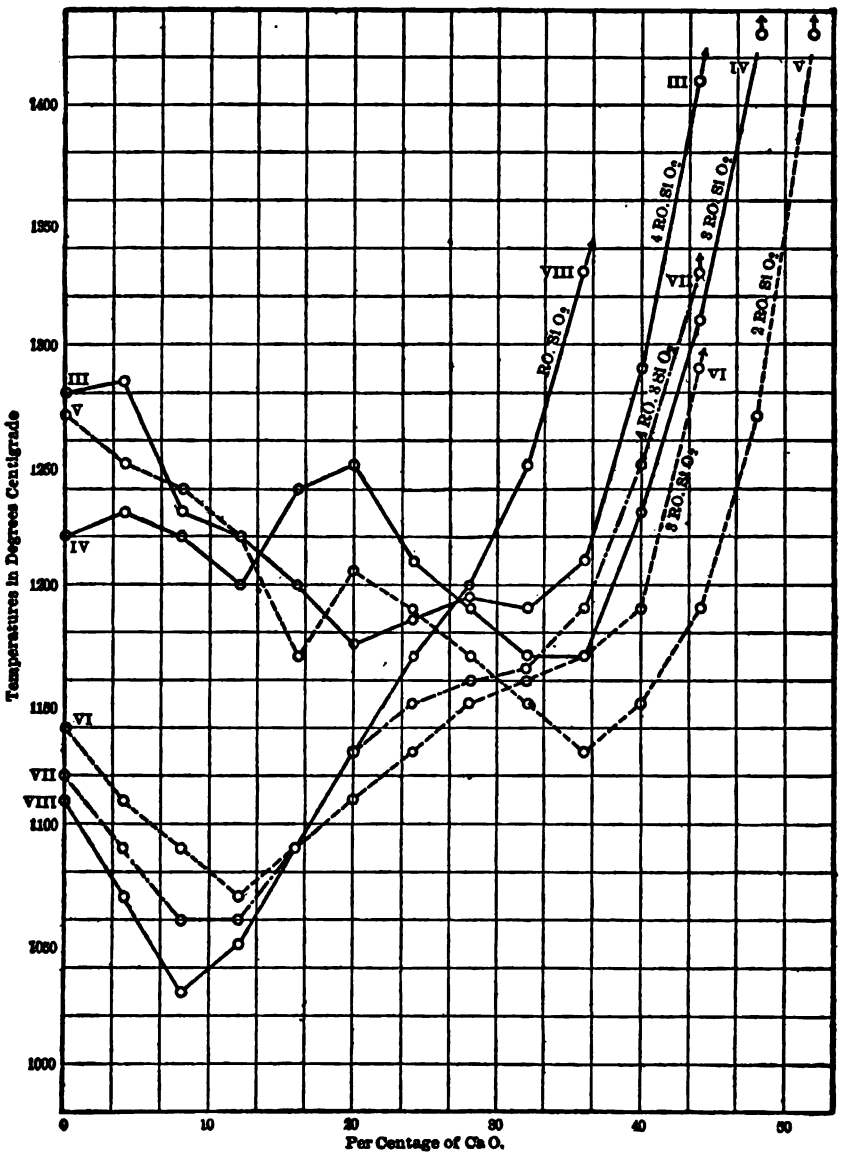


FIG. 2.—DIAGRAM SHOWING THE FORMATION TEMPERATURES OF FERRO-CALCIC SILICATES.

with a 2-in. course of red brick only, as shortly after blowing-in, an antimonial speiss is formed which prevents leakage of lead. In running, lead is tapped near the bottom while matte and slag are tapped together into an overflow pot in which the matte settles out. It is found that with a charge containing 17

to 18% Zn, a furnace will run two to three months without requiring any barring down of the wall accretions. When this becomes necessary the furnace is blown down, and the crucible lowered and wheeled aside. The barring is begun by cutting a channel down the middle of the sides, which allows the crusts to glide downward to the furnace floor where they are cooled with water, broken up, and wheeled away.

Lead Slags.—Hofman* investigated the temperatures at which certain ferrous and calcic silicates are formed in fusion, and the effect upon these temperatures of certain metallic oxides. The constituents (silica, ferrous oxalate, manganous carbonate, calcium carbonate, magnesium carbonate, barium carbonate, alumina and zinc oxide) were mixed in the desired proportions with a 10% dextrine solution, formed into small-size Seger cones (0·78125 in.

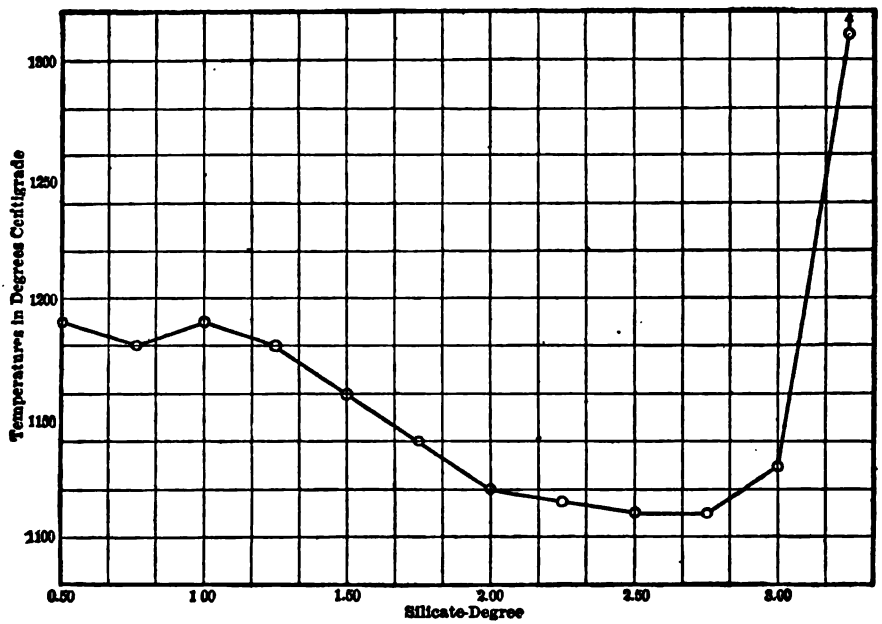


FIG. 3.—CROSS SERIES, SHOWING EFFECT OF INCREASE OF SILICATES UPON FORMATION TEMPERATURE.

high and 0·375 in. base) and melted down with Seger cones in a Seger gas furnace with a strongly reducing atmosphere to keep the iron in the ferrous state. The following diagrams represent graphically the results of the investigation. Fig. 2 represents the formation-temperatures of the sub-silicates, 4RO, SiO_2 (III.), and 3RO, SiO_2 (IV.), of the singulo-silicate 2RO, SiO_2 (V.), of the three-to-four silicate 3RO, $2SiO_2$ (VI.), of the sesqui-silicate, 4RO, $3SiO_2$ (VII.), and of the bi-silicate RO, SiO_2 (VIII.). The curves show that the pure ferrous silicate of a slag always has a higher formation-temperature than when part of the ferrous oxide has been replaced by lime, and that after a certain replacement the formation-temperature rises again more or less steadily until practically infusible mixtures have been reached.

* *Trans. A. I. M. E.*, California meeting, 1899.

It is also interesting to note the unexpected result, that, with the pure ferrous silicate, the formation-temperature falls as the silicate-degree rises, but that with slags more acid than the singulo-silicate (V.) only a comparatively small amount of ferrous oxide can be replaced by calcium oxide before the curve is reversed, and quickly ascends the temperature scale.

Fig. 3 represents the effect that an increase of silicate-degree has on the formation-temperatures of silica-iron-lime slags. In the mixtures the ratio of FeO:CaO is kept constant at 2:1, so as to eliminate any influence that a varying ratio of the bases might have. From the silicate-degree 0·50 to 1·25 the formation-temperatures are approximately constant. They then fall until a minimum is reached with a silicate-degree of 2·50 to 2·75, and then rise quickly.

For the study of the effect that the leading metallic oxides have on the

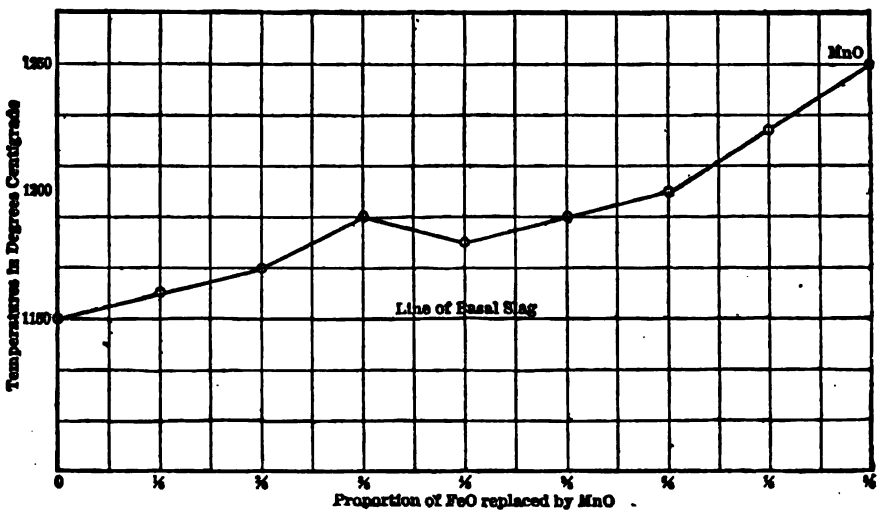


FIG. 4.—DIAGRAM SHOWING EFFECT OF REPLACEMENT OF FeO BY MnO.

formation-temperature of slags when acting as replacing constituents, the singulo-silicate SiO_2, 32·10%; FeO, 35·90%; CaO, 32·00% was chosen, as it afforded the largest range of experiment and had a low formation-temperature, *viz.:* 1,150°C. In the diagrams it is called *basal slag*. Fig. 4, representing the effect of manganous oxide in replacing equivalent amounts of ferrous oxide, shows that it raises this temperature approximately in proportion to the degree of replacement.

In Fig. 5 are represented the changes in formation-temperatures caused by the replacement of lime by magnesia, zinc oxide and baryta severally and combined. Magnesia (curve I.) is seen to raise the formation very decidedly; zinc oxide (curve IV.) sometimes raises, then again it lowers it, giving a very irregular curve; baryta lowers it very uniformly, proving this oxide to be a powerful flux; the combination of magnesia and zinc oxide (curve V.) does not improve matters, while as soon as baryta is added to magnesia (curve III.),

zinc oxide (curve VI.) or magnesia zinc oxide (curve VII.), the temperatures are lowered and the curves are straightened out.

The replacement of slag constituents by alumina is represented in Fig. 6. If it be figured in as silica (curve I.) the formation-temperature rises quickly;

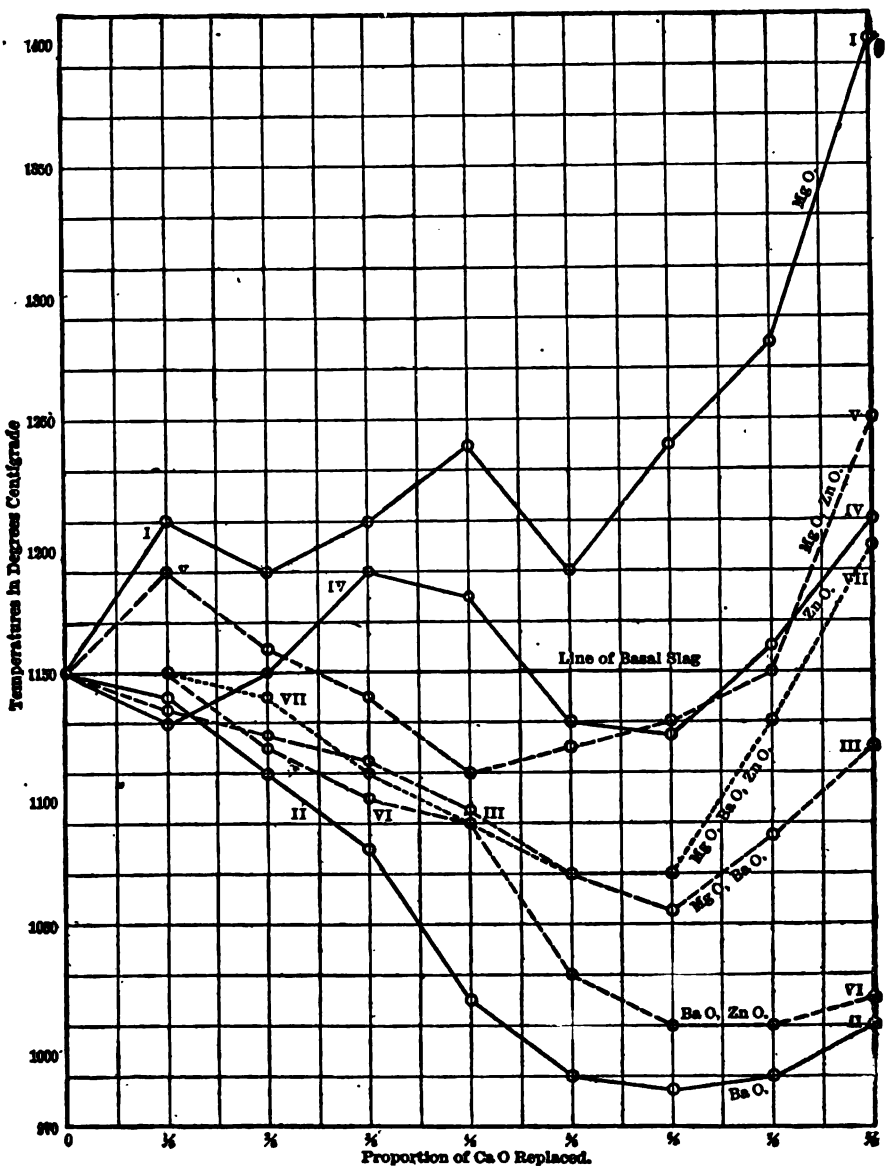


FIG 5.—DIAGRAM SHOWING EFFECT OF REPLACEMENT OF CaO BY MgO, BaO AND ZnO.

if as lime, the temperature sinks somewhat, but remains constant within definite limits; a combination of the two methods would appear to leave the temperature of the basal slag undisturbed. Figuring alumina as replacing ferrous oxide

(curve II.) is not satisfactory, and is even less so when it is to replace both ferrous oxide and lime (curve IV.).

Smelting at Pertusola.—Heberlein* publishes some data about the blast furnace work at Pertusola, Italy. The blast furnace is 23 ft. high, 7·5 ft. in diameter at the tuyeres, of which there are 15 placed in two rows. It is claimed that this arrangement assists in oxidizing sulphur, increases the smelting power, saves coke, and causes a greater heat with consequently fluidity of slag. In 24 hours there were put through on the average 217·73 metric tons of charge containing 33·95% Pb; the fuel consumption was 8·05% coke. No matte was produced. The slag made has the following composition: SiO_2, 26·70%; FeO, 32·95%; ZnO, 12·80%; CaO, 19·71%; MgO, 1·80%; Al_2O_3,

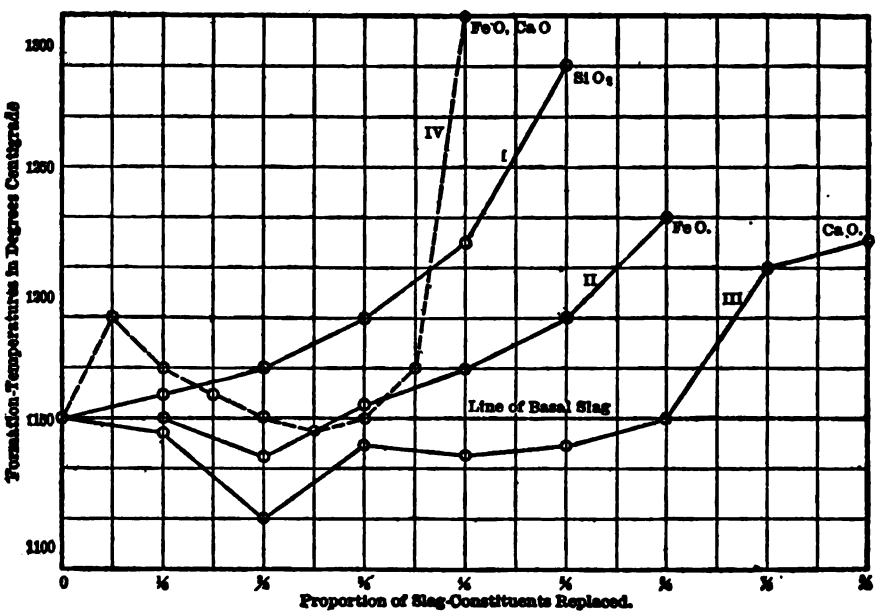


FIG. 6.—DIAGRAM SHOWING EFFECT OF REPLACEMENT OF SLAG-CONSTITUENTS BY Al_2O_3.

3·10%; PbO, 1·16%; S, 1·03%; Ag, 0·0009% (0·26 oz. per ton); total, 99·3309%.

Smelting Charges.—An editorial in the *Engineering and Mining Journal*,† cites a contract in Utah, in which no deductions were made for as much as 20% Zn, for a lead ore with 20 to 40% Pb, 1·5 to 3·5% Cu, and considerable iron, beside silver and gold. Watt,‡ of Hailey, Id., states that this must have been a very exceptional case, as the zinc limit at present is 10%; this limit has been mentioned for many years.

Matte Settling Furnace.—Iles§ discusses in a general way the apparatus for settling out matte and gives some facts concerning his patented reverberatory furnace used for this purpose. An average of the slag made at the Globe works, Denver, Colo., showed SiO_2, 31·35%; Fe(Mn)O, 35·07%; Ca(Mg.Ba)O, 20·25%; ZnO, 6·23%; Al_2O_3, 6·07%; an average of the matte Fe, 30 to 40%;

* *B. & H. Ztg.*, 1899, p. 189. † Sept. 23, 1899, p. 361. ‡ *E. & M. J.*, Oct. 14, 1899. § *Ibid.*, Dec. 2, 16, 1899.

Pb, 12 to 25%; Cu, 15 to 30%; Zn, 3 to 12%; S, 15 to 20%; Au, 0·08 to 0·20 oz.; Ag, 80 to 200 oz. During six months there was produced 0·95 ton slag for every ton of ore. The handling of 500 tons slag required three reverberatory settling furnaces, two in operation, while the third is held in reserve. Where much matte is made it is advisable to use a matte box, which will collect most of the matte and thus relieve the reverberatory furnace. As it is necessary to have a certain quantity of matte in the reverberatory furnace to prevent materials rich in zinc from settling and building on the bottom, Iles advises that of the five furnaces serving one reverberatory-settler, four be supplied with matte boxes, and the fifth allowed to discharge its slag and matte mixture directly into the settler. While the settler removes most of the silver from the slag, some always remains. An average analysis of the fumes issuing from the settler showed: S, 7·40%; Zn, 20·40%; As, 1·50%; Sb, 2·20%; Pb, 39·10%; another sample of fume withdrawn 2 ft. from the top of a 60-ft. stack showed: SiO_2, 1·70%; Fe_2O_3, 1·00%; ZnO, 21·00%; SO_2, 1·60%; SO_3, 9·75%; CaO, 0·60%; As_2O_3, 0·18%; Al_2O_3, 1·20%; K_2O, 0·60%; Na_2O, 0·85%; C, 24·40%; PbS, 14·64%; PbO, 21·64%; NH_3, trace; total, 99·16%.

Weathering of Slag.—Iles-Shelby* analyzed a whitish efflorescent salt found at or near the bottom of the slag dump of the works of the Globe Smelting and Refining Co., Denver, Colo., and found it to contain: SO_3, 48·40%; K_2O, 28·44%; Na_2O, 13·48%; CaO, 1·80%; MgO, 1·08%; Fe_2O_3, 0·29%; MnO, 0·26%; Cl, 0·47%; ignition loss, 2·40%; total, 96·67%.

The rational analysis calculated from it gives:

Potassium sulphate	52·64	Manganous sulphate	0·56
Sodium sulphate	29·97	Sulph. anhydride in combination with other bases	1·89
Sodium chloride	0·77	Loss in ignition	2·40
Calcium sulphate	4·37		
Magnesium sulphate	3·24	Total	96·57
Ferric sulphate	0.78		

The authors believe the missing 3·43% to consist of the alkalies lithium, cæsium or rubidium, probably in the form of sulphate.

Canby† calls attention to the occurrence of similar salts in piles of matte from the Hall Mine Smelter, Nelson, B. C., at the Argentine works of the Consolidated Kansas City Smelting and Refining Co., after they had been exposed to moisture and warmth in the open air.

Losses in Smelting.—Iles‡ discusses silver and lead losses in lead smelting, the data being based on the results obtained at the works of the Globe Smelting and Refining Co., Denver, Colo., mainly in 1892-93. The figures are discussed by Godshall,§ who supplements them by some of his own, obtained at Everett, Wash. Of neither article could an adequate abstract be made, and the reader is referred to the originals.

Natural and Forced Draught.—Iles‖ holds the opinion that in most metallurgical furnaces the natural draught by means of a stack can be replaced by the artificial draught produced with a fan. At the Globe works a 10-ft. fan

* *E. & M. J.*, June 3, 1899. † *Ibid*, July 1. 1899. ‡ *Ibid*, Sept. 9, 16, 23, 1899. § *Ibid*, Jan. 13, 1900.
‖ *Ibid*, Dec. 3, 1899.

handles all the gases from the blast furnace. The fumes from melting silver bars being rich are saved by filtering; those from the Faber du Faur retorts, while under normal conditions not sufficiently rich to be saved, are nevertheless collected in order to obviate any losses in case of breakage of retorts. The fumes from the softening furnaces are not sufficiently rich to be worth collecting. The author further gives some constructive details in regard to fans.

Flue-Dust.—At the works of the Globe Smelting and Refining Co., Denver, Colo., Iles* found that a liquid collected during cold weather at the extreme end of the 8-ft. diameter conduit-pipe, leading to the bag-house, and saturated the cloth bag attached to a thimble on the lower side which served to discharge the collected dust. This liquid formed long yellowish icicles, which were gathered, dissolved in water, filtered, and gave the following weights in g. per liter of solution: $FeSO_4$ 88·40, $MnSO_4$ 0·68, $ZnSO_4$ 24·56, As_2O_3 1·45, NaCl 7·26, Na_2SO_4 3·69, K_2SO_4 2·22, $(NH_4)_2SO_4$ 33·00, H_2SO_4 1·22, H_2SO_3 15·00, organic matter 31·94, total 209·42 g. The gases issuing from a lead blast furnace always contain water, sulphuric and sulphurous acids, and being condensed, especially in cold weather, will attack flue-dust and form sulphates. The author believes that some manganese was volatilized as chloride. The ammonia and organic matter resulted of course, from the coke. The tests showed that some arsenic was present as sulphide.

Canby† believes that ammonia is very common in flue-dust, being present as sulphate. He bases his belief on the facts observed at the lead works of Argentine, Kan., El Paso, N. M., and San Luis Potosi, Mexico, that when flue-dust is mixed with milk of lime in the wet pan previous to briquetting in a machine, ammonia is liberated in irritating quantities.

Briquetting.—The Henry S. Mould Co., Pittsburg, has brought out a compact briquetting machine, 6 ft. high, covering a floor-space 12×16 ft., and weighing 30,000 lb. It contains six cylindrical moulds, each with a compression plunger exerting a pressure of 10,000 lb. to the sq. in. The plungers are operated by a heavy pitman and crank shaft. With a speed of 25 strokes per minute, the machine turns out 6×25 or 150 briquettes of about 3 in. diameter and 3 to 3·5 in. long.

The Chisholm, Boyd and White Co., of Chicago‡ have improved the auxiliary machinery of their well-known briquetting plant, by bringing together into a compact whole a lime slacking-machine, an automatic feeder, and a conveyor-mixer from which the material to be briquetted passes into the machine.

Dust Collection.—Sergeant patented§ an oblong dust chamber having baffle plates, with transverse air-pipes at the turning points of the air current, and a bottom consisting of a series of hoppers.

DESILVERIZATION OF BASE BULLION.

Parkes Process.—Iles‖ publishes the following assays and analyses of refinery products, most of which are from the works of the Globe Smelting and Refining Co., Denver, Colo.:

* *E. & M. J.*, Nov. 11, 1899. † *Ibid*, Nov. 25, 1899. ‡ *Ibid*, May 13, 1899.
§ United States Patent No. 626,569, June 6, 1899. ‖ *S. M. Q.*, 1899, XX., p. 397.

REFINERY PRODUCTS.

No.	Description.	SiO_2.	Fe.	Mn.	S.	Zn.	Cu.	As.	Sb.	Al_2O_3	Fire Assays. Ag, oz.	Pb, %	Au, oz.	Remarks.	
1	Fume from the hood over breast of cupel furnace	.....	.....	.....	.....	.....	.....	.....	.....	.....	134·80	41·80	0·32	Balbach refinery, Newark, N. J. Walker.	.....
2	Fume from the hood over breast of cupel furnace	.....	.....	.....	.....	.....	.....	.....	.....	.....	135·20	41·80	0·32	Balbach refinery, Newark, N. J.	.....
3	Fume from the hood over breast of cupel furnace	.....	.....	.....	.....	.....	.....	.....	.....	.....	135·20	38·50	0·32	Balbach refinery, Newark, N. J.	.....
4	Dust from cupel furnace dust chamber	.....	.....	.....	.....	.....	.....	.....	.....	.....	71·50	16·60	Tr.	Globe refinery.	.....
5	Dust from cupel and concentrat'g furnace chamb.	.....	.....	.....	.....	.....	.....	.....	.....	.....	106·60	27·60	0·40	Light gray colors, putty-like when moistened.	Globe
6	Cupel furnace bottom	.....	.....	.....	.....	.....	1·4	.....	.....	.....	702·00	45·00	10·40	Sample of the entire bottom.	"
7	Cupel furnace bottom	.....	.....	.....	.....	.....	2·3	.....	.....	.....	547·00	37·60	6·00	Sample of the entire bottom.	"
8	Dust near cupel furnace, in flue	.....	.....	.....	.....	.....	.....	.....	.....	.....	67·00	57·00	0·32		"
9	Dust from softening furnace flue	.....	.....	.....	.....	.....	.....	.....	.....	.....	30·00	33·80	0·16	Sample taken 10 ft. from furnace.	"
10	First skimmings of softening furnace	.....	.....	.....	.....	.....	.....	.....	.....	.....	147·40	96·50	4·10	Germania refinery, Salt Lake, Utah.	"
11	Fire-brick from side of softening furnace	.....	.....	.....	.....	.....	.....	.....	.....	.....	126·00	35·50	Tr.	Sample taken during repairs upon furnace.	"
12	Mixed fume from softening antimony and refining furnace	.....	.....	.....	.....	.....	.....	7·06	4·28	.....	5·00	20·80	Tr.	Globe.	"
13	White fume from outside door of soften'g furnace	.....	.....	.....	.....	.....	.....	2·56	32·73	.....	7·00	20·60	0·05	Sample taken while using air to hasten softening.	"
14	First, or copper skims, from softening furnace	0·60	1.20	N.	2·00	N.	18·20	.....	.....	.....	180·00	53·00	12 00		"
15	Liquated first skims of softening furnace	2·00	2·80	N.	3·60	N.	26·80	7·31	1.18	.....	75·00	50·00	0·30		"
16	Liquated first skims of softening furnace	.....	.....	.....	.....	.....	26·20	.....	.....	.....	.....	.....	.....	Month's average.	"
17	Skimmings from refining furnace	1·60	0·60	....	N.	8·00	Tr.	N.	4·02	.....	1·00	78·00	M. Tr.		"
18	Skimmings from refining furnace	0·70	0·60	N.	N.	6·60	0·20	....	N. D.	.....	0·70	71·00	N.		"
19	Skimmings from refining furnace	.....	.....	.....	.....	8·00	.....	.....	.....	.....	0·70	78·00	N.		"
20	Retort fume on beams, running two weeks	.....	.....	.....	.....	.....	.....	.....	.....	.....	21·00	.....	.....		"
21	Retort fume on beams, running two weeks	.....	.....	.....	.....	.....	.....	.....	.....	.....	13·00	1·20	Tr.	Product consists mainly of ZnO.	"
22	Retort fume at base of stack	.....	.....	.....	.....	.....	.....	.....	.....	.....	10·80	Tr.	0·16	Product consists mainly of ZnO and ashes of coke.	"
23	Retort ashes	63·2	2·4	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	Very liable to cont'n gold, silver & lead. Colo. coke.	"
24	Retort fume, base of stack, large sample	.....	.....	.....	.....	.....	.....	.....	.....	.....	29·70	0·60	0·32	Presence of gold particularly noticeable.	"
25	Retort fume, average of all dust in flue	.....	.....	.....	.....	12·20	.....	.....	.....	.....	16·80	0·50	0·20		"
26	Retort fume, collected on top of retort flue	.....	.....	.....	.....	.....	.....	.....	.....	.....	38·50	Tr.	Tr.		"
27	Retort fume, collected on top of iron columns	.....	.....	.....	.....	.....	.....	.....	.....	.....	44·00	Tr.	Tr.	Sample close to retorts.	"
28	Retort ashes	29·4	4·0	.....	.....	0·8	.....	.....	.....	12·80	1·00	Tr.	Tr.	Using Connersville coke.	"
29	Retort ashes	43·0	6·0	.....	.....	.....	.....	.....	.....	32·20	1·40	Tr.	Tr.	Gunnison, Colo. coke.	"
30	Retort fume, settled on plank near retorts	.....	.....	.....	.....	25·20	.....	.....	.....	.....	44·00	1·30	1·12	Rather large sample.	"
31	Dust from the base of an 100-ft. refining stack	.....	.....	.....	.....	.....	.....	.....	.....	.....	71·20	38·70	0·08	General mixture of smoke from all kinds of refining furnaces.	"
32	First zinc skimmings from kettles	.....	.....	.....	.....	18·20	4·40	.....	.....	.....	2904·00	67·00	3·00	Park's process—zinc skimming.	"
33	Copper skimmings in liquating hard lead	3·80	4·8	....	5·7	0·90	4·10	1·10	17·0	.....	4·80	31·20	0·14	Antimonial lead, copper skims.	"
34	Antimonial, or hard lead	.....	.....	.....	.....	.....	0·50	1·25	25·97	.....	2·40	.....	0·05	Analysis by Von Shultz and Low, Denver, Colo.	Rand.
35	Antimonial, or hard lead	.....	.....	.....	.....	.....	0·60	Tr.	25·95	.....	.....	.....	.....	Analysis by Prof. W. B. Potter, St. Louis, Mo.	"
36	Antimonial, or hard lead	.....	.....	.....	.....	.....	0·34	0·94	26·86	.....	1·92	71·52	0·02	Analysis by Globe laboratory, Sixth Run.	Globe
37	Litharge, average of one month's run	.....	.....	.....	.....	2·10	4·10	0·20	.....	.....	.....	.....	.....		"
38	First zinc skimmings from kettles	.....	.....	.....	.....	11·5	5·4	.....	.....	.....	2217·00	73·50	63·40		"
39	Unusually hard bullion, 30 to 34 hours for softening	.....	.....	.....	.....	.....	0·71	0·28	3·27	.....	240·00	95·35	2·00	Avalysis by Prof. W. B. Potter, St. Louis, Mo.	"
40	Fume from antimony furnace	.....	.....	.....	.....	.....	.....	5·04	2·88	.....	7·20	38·30	Tr.	Collected with textile fabric.	"
41	Fume from softening furnace	.....	.....	.....	.....	.....	.....	0·38	10·45	.....	32·50	19·20	Tr.	Collected with textile fabric.	"
42	Fume from cupel, concentrat., & silver melt'g fur.	.....	.....	.....	.....	0·6	.....	N.	Tr.	.....	85·90	72·50	0·10	Collected with textile fabric.	"
43	Fume from softening, refining and antimony fur.	.....	.....	.....	.....	.....	.....	7·09	3·56	.....	7·00	20·50	Tr.	Collected with textile fabric. [nace fume.	"
44	Fume from softening, refining and antimony fur.	.....	.....	.....	.....	.....	.....	.....	.....	.....	288·00	28·00	0·36	Collected with textile fabric, mostly softening fur-	"
45	Fume from antimony, soft'ing, & concentrat. fur.	.....	.....	.....	.....	.....	.....	.....	.....	.....	149·00	62·00	Tr.	Collected with textile fabric.	"
46	Fume from softening and concentrator furnaces	.....	.....	.....	.....	.....	.....	.....	.....	.....	115·00	52·00	Tr.	Collected with textile fabric.	"
47	Fume from softening and concentrator furnaces	.....	.....	.....	.....	.....	.....	.....	.....	.....	86·00	72·00	Tr.	Collected with textile fabric.	"
48	Fume from softening and concentrator furnaces	.....	.....	.....	.....	.....	.....	.....	.....	.....	87·00	72·00	Tr.	Collected with textile fabric.	"

Jones,* in discussing the results obtained by Blakemore,† in desilverizing base bullion rich in gold and copper by means of the Parkes process, notes the large amount of rich dross drawn off from the softening furnace and the kettle and advocates matting the charge in the softening furnace‡ when a matte with over 40% copper carrying little or no gold will result, and one-quarter of the present amount of dross will be produced. He also states that he found no difficulty in the Transvaal in separating by zinking the gold from the silver in bullion that contained 90 to 100 oz. gold to 20 to 40 oz. silver per ton; and that at the Argentine works of the Consolidated Kansas City Smelting and Refining Co., where three-quarters of the bullion was treated in this way, the fine silver contained only 0·01 to 0·02 thousandths of gold. The proportion of gold and silver in the bullion is not given.

Electrolytic Refining.—Tommasi§ has issued an illustrated circular in which he outlines his process for the electrolytic refining of lead, to which reference has already been made.

Cupellation.—Kochinke,‖ in a general article on the refractory materials used at the Freiberg, Saxony, smelting works, gives the following data relating to the materials used in lining cupelling, softening, and refining furnaces. The material with which the tests are filled consists 2 vols. blue marl from Steinbach: ($CaCO_3$ 37·0%, $MgCO_3$ 20%, $Fe(Al)_2O_3$ 1·9%, clay substance 38%) and 1 vol. yellow marl from Tarnowitz ($CaCO_3$ 54·53%, $MgCO_3$ 20·55%, SiO_2 12·72%, Al_2O_3 6·07%, $FeCO_3$ 1·77%, H_2O 4·50%) crushed in a ball mill with 10% of clay from Löthain (Al_2O_3 30·44%, SiO_2 58·28%, MgO 0·08%, CaO 0·26%, Fe_2O_3 0·90%, K_2O 0·06%, loss on ignition 10·04% with a refractory quotient [Bischof] 7·547). For lining the softening or refining furnaces a mixture of 45 vols. blue marl, 22 yellow marl, and 33 clay is used. In order to counteract the corrosive action of litharge, at first magnesia bricks made by Späther, at Göppersdorf, Silesia, of the composition MgO 85·80%, CaO trace, Fe_2O_3 6·10%, Al_2O_3 3·00%, SiO_2 4·60%, were used, and put in place with a basic mortar furnished by the maker.

While the bricks stood well at first, it was found impossible to patch the furnace, as the mortar would not adhere to the hot brick. As a result, the life of the bricks was too short to warrant their use of material of such high cost. Similar magnesia bricks, but with a pitch bond, made at Duisburg were equally unsatisfactory. At present there are in use refractory clay bricks, especially high in alumina, consisting of non-plastic clay mixed with a small percentage of plastic clay as a bond, moulded by hand, air-dried and placed in the furnace, where they become hard and strong and wear better than if they had been first kiln-burnt. Since the first successful application in lead softening and refining furnaces of these fire bricks rich in alumina, they have been used in shaft furnaces, crucible furnaces, and as supports for the muffles which are used to distil calcined and roasted zinc ores.

Parting Doré Silver.—The discussion of the priority of the Gutzkow process for parting gold and silver (THE MINERAL INDUSTRY, Vol. VI., p. 459) is concluded in the *Berg- und Hüttenmannische Zeitung,* 1899, p. 75 (Gutzkow) and p. 128 (Rössler).

* *E. & M. J.*, Sept. 2. 1899. † *M. I.*, VII., p. 468. ‡ See also, Hofman, *Lead*, 5th Edition, p. 435.
§ *M. I.*, V., 412; VI., 459. ‖ *Frieb. J.*, 1898, 96, 108.

LITHIUM.

By Hans A. Frasch.

The metal lithium is classed among the rare elements and in its metallic form it is seldom produced except as a chemical specimen. The salts of lithium, however, exist in several minerals of common occurrence, and analyses of them are given in the following table. The element lithium itself has as yet no application in the arts, although its salts are used for the manufacture of lithia water, which has a very extensive use medicinally for rheumatic and gouty complaints. The carbonate is mainly used for this purpose, and while there are a few natural lithia waters by far the greater part of the lithia water of commerce is manufactured.

A new method for the preparation of metallic lithium has been described by Kahlenberg,* which is certainly unique. Advantage is taken of the fact that pyridine will not be decomposed by the metal, while at the same time the chloride is soluble in this substance. The method of operation consists in electrolyzing a concentrated solution of lithium chloride in pyridine, using a carbon plate as anode and a bright iron plate or foil as cathode. A voltage of 14 between the electrodes and a current of 0·2 to 0·3 ampère per sq. cm. of cathode surface gives a dense, well adhering deposit which is silver white. Lithium now costs $3 to $5 a gm., while by this process it can be prepared very cheaply. The details of the method as to the best yield under the most favorable conditions have not yet been studied.

Moissan,† on experimenting with metallic lithium, was able to prepare an alloy with ammonia, and on continuing the work finds that with methylamine, lithium forms a compound which is solid at ordinary temperatures.

The salts of lithium are mainly produced from the carbonate which is obtained by various methods from the minerals lepidolite and spodumene.

Lepidolite is represented by the formula $3R_2 3Al_2Si_6O_{24}$, the radical being potassium, sodium, lithium, or hydrogen, and a part of the oxygen replaced by fluorine. It occurs in this country in Maine, Connecticut and California,‡

* *Journal Phys. Chem.*, Dec., 1899. † *Comptes Rendus*, Vol. CXXVIII., p. 26, Jan. 1, 1899.
‡ The Mineral Industry, Vol. VII.

and is the most important of the lithium minerals; it contains from 2 to 5% of lithia (Li_2O).

Spodumene has the formula $3(LiNa)_2SiO_3,4(Al_2Si_3O_9)$. It occurs in this country at Sterling, Me., and in South Dakota, and contains an average of 5% of lithia. Other minerals of minor importance are:

Petalite, $3(LiNa)_2SiO_5,4(Al_2Li_6O_{15})$, which occurs in Massachusetts and in Canada, and averages 3% lithia. *Amblygonite*, $2Al_2(PO_4)_2,3RF$, which occurs in Maine and Florida, and contains 6% of lithia and 40 to 48% phosphoric acid, and *Triphylite*, $R_3PO_4,R''_3(PO_4)_2$, which bears 7 to 8% of lithia, and is found at Norwich, Mass.

Small amounts of lithium are found in tourmaline, in sea water and in many mineral waters. It is also present in the salt brine obtained at Cleveland, O., where it separates from the concentrated brine, together with gypsum and calcium carbonate, if the brine has been slightly treated with sodium carbonate before evaporation. This gypsum precipitate contains lithium compounds in sufficient quantities to suggest the recovery of its salts.

Rammelsberg gives the analysis of different lithium-bearing minerals as follows:

Component	Lepidolite.	Spodumene.	Petalite.	Amblygonite.	Triphylite.
	%	%	%	%	%
Silica	50·91	65·02	77·19	.....	0·25
Alumina	28·17	29·14	18·58	36·88	.....
Ferrous oxide	.....	.....	.....	.....	39·97
Manganese oxide	1·20	.....	.....	.....	9·8
Lime	.....	0·50	.....	.....	.....
Magnesia	.....	0·14	.....	.....	.....
Potassium	9·50	0·14	.....	0·43	0·58
Sodium	.....	0·46	1·19	3·29	1·45
Lithium	5·67	5·47	3·30	6·68	7·28
Fluorine	3·90	.....	.....	8·11	.....
Phosphoric acid	.....	.....	.....	47·58	40·72

The properties of the salts of lithium stand between the alkalies and alkaline earths. The carbonate and phosphate are barely soluble; the solubility of the sulphate and carbonate is greater in cold than in warm water. The solubility of the hydrate and chloride resembles the corresponding alkaline earth salts. In all other respects the properties of the lithium salts incline to the salts of the alkalies.

The Manufacture of Lithium Salts.—Lithium carbonate is the most important lithium compound, and from it many lithium salts are obtained. The carbonate is mainly produced from lepidolite, although spodumene and triphylite are also employed. The process of manufacture consists in the production of a concentrated solution of lithium chloride; conversion of the latter into the form of carbonate by treatment with the carbonate of sodium, potassium or ammonium, and a final separation of the carbonate, accomplished by various ways. There are two methods of manufacture, one involving a preliminary fusion of the mineral, and the other submitting the pulverized mineral direct to the acid treatment. The selection of the process is governed by the mineral available. Lepidolite is treated by both and triphylite by the direct acid method.

To manufacture lithium carbonate* from lepidolite, the mineral is fused in crucibles or preferably in furnaces similar to those employed for the manufacture of sodium silicate; the fused mass is poured into water and reduced to a pulp. The amount of hydrochloric acid required for the complete decomposition of the mass is then determined and one-half of this quantity is mixed with the pulp and allowed to rest for 24 hours. If the mass is too stiff a small amount of water may be added, and the whole is then heated nearly to the boiling point, when the remaining half of the hydrochloric acid is added with constant agitation. It is then allowed to stand for several hours until the silica is completely separated. To ascertain if enough acid is present, the filtrate of a small portion when treated with a few drops of a solution of sodium carbonate should not show a permanent precipitate. While the mass is yet hot, the iron is oxidized by nitric acid or sodium hypochlorite and the silica separated by filtration, satisfactorily done in a vacuum filter. The filtrate is brought to boiling and is carefully treated under constant agitation with sodium carbonate until the ferric oxide, alumina, lime, magnesia, etc., are precipitated and subsequently removed by a filter press; the slightly alkaline filtrate contains the chlorides of lithium, sodium, potassium and possibly rubidium, cæsium, and thallium, and a small amount of silica. It is evaporated in iron vessels producing a further separation of a small amount of silica and magnesium carbonate. The concentrated solution is acidulated with hydrochloric acid and the lithium is precipitated as lithium carbonate by the addition of sodium carbonate. The remaining solution at times is treated for the recovery of rubidium, cæsium and thallium.

In some instances sulphuric acid is used in place of the hydrochloric acid on account of the easier separation of the earthy salts.

The sulphuric acid treatment is generally applied to the raw mineral and several methods of separation are in use. Filsinger† recommends the treatment of the finely pulverized lepidolite with concentrated sulphuric acid, which may be spent acid from the nitration of organic products or from oil refineries. The mixture is moderately heated in iron pans or bricked receptacles connected with a calcining furnace similar to the ordinary salt-cake furnace used in the manufacture of hydrochloric acid. The pulp is occasionally agitated and further heated until the mass forms hard, irregular lumps, which are transferred to the furnace proper and are calcined until all fluoric acid and the excess of sulphuric acid are removed by evaporation. The calcined mass is then systematically leached with water in lead-lined vessels until the residue consists of nearly pure silica with less than 2 to 3% of undecomposed lepidolite. The aqueous solution obtained by the leaching process is concentrated in copper vessels and treated while hot with a sufficient amount of potassium sulphate to combine with the aluminum sulphate and form alum, which separates on cooling and is recovered as a by-product in a centrifugal machine. The alumina and ferric salts, still in solution after the separation of the alum, are precipitated by the addition of milk of lime to the boiling solution and collected in a filter press; the filtrate is treated with barium chloride, whereby the

* *Journal für Prakt. Chem.*, 68, 310; 95, 143; 95, 43. † *Polytech. Journal*, 222, 271, 385.

sulphates are transformed to the chlorides and barium sulphate is precipitated and obtained as a by-product. The remaining filtrate containing the chlorides of lithium, sodium, potassium, calcium, etc., is evaporated to dryness and the residuum boiled with alcohol in a closed vessel provided with a reflux condenser. In this manner the chlorides of lithium and calcium are dissolved. The alcoholic solution is then evaporated with the recovery of the alcohol, the residuum is dissolved in water, and the calcium removed by the addition of ammonium oxalate, which leaves lithium chloride only in the solution; the lithium carbonate is precipitated from this by treatment at a boiling temperature with ammonium carbonate. To remove the last traces of base metals which may be present in the final solution, it may be treated with a small amount of ammonium sulphide before the addition of ammonium oxalate. The lithium carbonate obtained by this method is washed with 60% alcohol until it ceases to show chlorine reaction.

This process furnishes at once a marketable product and appears advantageous on account of the by-products obtained. It is a question, however, in this country whether the cost of the more complicated apparatus and expensive reagents does not overbalance the low-priced by-products.

Schering* uses sulphuric acid and treats the leaching product with milk of lime until slightly alkaline; the lithium is precipitated from the filtrate by potassium carbonate. The lithium carbonate so obtained is redissolved in hydrochloric acid, traces of lime are removed by the addition of a small amount of ammonium oxalate, and the lithium carbonate is precipitated from the hot and concentrated solution by ammonium carbonate. This process appears advantageous when not too much alumina is present in the mineral, otherwise the precipitate of aluminum hydrate is too voluminous and the precipitation has to be conducted in greatly diluted solutions, which may result in a lower yield of lithium.

The use of gypsum in place of sulphuric acid has been suggested to cheapen the process of extraction. By this method equal quantities of finely pulverized lepidolite and gypsum are fused and the product leached in the same manner employed when sulphuric acid has been first used; the aqueous solution is evaporated and the potassium sulphate and gypsum are crystallized and separated from the remaining solution, the impurities are removed by the ordinary addition of ammonia, ammonium sulphide and ammonium oxalate, and the lithium is precipitated from the filtrate by ammonium carbonate and purified by repeated precipitations and washings with alcohol. It is claimed that two-thirds of the lithium present in lepidolite can be obtained by this method.

Lithium carbonate is obtained from triphylite by treating the finely pulverized mineral with hydrochloric acid and oxidizing the iron by the addition of sodium nitrate or nitric acid; ferric chloride is added to bind all the phosphoric acid, and the mixture is evaporated to dryness. The soluble chlorides are then extracted with boiling water, leaving the residuum with the ferric phosphate and some ferrous chloride. From the solution, manganese and iron are precipitated by barium sulphide, and excess of the latter reagent is re-

* *Development of the Chemical Industry*, 1875, p. 1014.

moved by careful addition of sulphuric acid; the barium sulphate thus formed is removed and after the addition of nitric acid the solution is evaporated to dryness until all the hydrochloric acid is expelled. The nitrates are decomposed by heating the mass with copper filings and the alkalies are brought in solution by boiling the fused mass with water; from this solution lithium carbonate is precipitated by treatment with carbonic acid gas. The carbonate is redissolved and purified by the usual methods.

Lithium carbonate fuses at light red heat and forms on cooling a crystalline mass. At an increased temperature in the atmosphere carbonic acid is liberated and peroxide is formed; if the air is excluded pure oxide remains. If heated with carbon, lithium oxide is obtained. If precipitated from a solution of lithium sulphate or chloride by sodium or potassium carbonate, the lithium carbonate is obtained in the form of a gritty powder; but if precipitated from the chloride by ammonium carbonate the light and voluminous product is obtained which is demanded by the market. The solubility of lithium carbonate is greater in cold than in warm water; 100 parts of water dissolve at 0° C.—1·539; 10°, 1·406; 20°, 1·329; 50°, 1·181; 75°, 0·866; 100°, 0·728 parts of lithium carbonate.* It may therefore be separated from alkali carbonates by treatment with a small amount of boiling water and from the carbonates of the earthy alkalies by dissolving it in a large amount of cold water. A solution of lithium carbonate precipitates calcium and barium from their soluble salts.

The production of lithium carbonate in this country is in an early stage of development. Schieffelin & Co., of New York City, are erecting a plant which will treat one ton of lepidolite a day and expect to operate it early in 1900. They have contracted for the delivery of 400 tons of lepidolite from the mines near Pala, Cal., during 1900. This company is also considering the erection of a plant in California for the treatment of lepidolite. Germany supplies nearly the entire demand, a great part of which is exported to England and America. It is estimated that at least 20,000 lb. of lithium carbonate is consumed annually in the United States. The price ranges from $4 to $5 per lb.

Lithium chloride is obtained by dissolving the carbonate in hydrochloric acid and evaporating the solution; it is very hygroscopic, crystallizes without water in octahedra and has the taste of table salt. It is easily soluble in alcohol or in a mixture of alcohol and ether, and by this means it may be separated from many of the other chlorides. It forms double salts with manganese, iron, nickel and cobalt corresponding to the formula $2MnCl_2$, $LiCl$, $6H_2O$.†

Lithium sulphate is produced by decomposing the chloride or carbonate with sulphuric acid and evaporating the resulting solution. Like lithium carbonate, the sulphate is more soluble in cold than in warm water; 100 parts water dissolve at 0° C., 2·83; 20°, 2·91; 45°, 3·06; 65°, 3·30; 100°, 3·42 parts of lithium sulphate respectively. It is insoluble in absolute alcohol; it does not form alum with aluminum sulphate, nor does it combine with magnesium sulphate; it forms, however, double salts with potassium, sodium and ammonium sulphate.

* *Chem. Berichte.*, 17, c. 406. † *Chem. Central Blatt*, 1892, II., 393.

Lithium Phosphate.—Two salts are produced, the neutral Li_3PO_4 and the acid salt LiH_2PO_4. The former is produced by boiling a solution of the normal sodium phosphate with a solution of lithium sulphate, when the lithium phosphate, $(2Li_3PO_4)\ H_2O$, precipitates in the form of a heavy white powder. The reaction is: $3Li_2SO_4+2Na_2HPO_4=2Li_3PO_4+2Na_2SO_4+H_2SO_4$. The sulphuric acid which is liberated prevents the complete precipitation of the lithium phosphate. To obtain a complete precipitation the sulphuric acid must be neutralized and free ammonia must be present. If sodium is employed for neutralization the precipitate always contains sodium.* If dried at 100° C. lithium phosphate loses its water of crystallization. It crystallizes if fused in lithium chloride, from which the crystals may be separated by careful treatment with water. It is soluble in all diluted acids.

The acid salt, LiH_2PO_4, is obtained by dissolving the neutral salt, or lithium carbonate, in an excess of phosphoric acid and evaporating the solution to crystallization. This salt if heated forms first the pyro, and then the metaphosphate of lithium.

Other lithium salts, the acetate, iodide and nitrate, are produced by dissolving the carbonate in their corresponding acids. The arseniate is produced by the same method used for the phosphate.

* *Jahresbericht*, 1882, 267.

MAGNESIUM AND MAGNESITE.

The production of magnesite in California fell off during the year. This State still remains the only producer. In 1899 the United States imported 20,807 short tons, mainly from Greece and Styria. A certain quantity also came from Germany. Magnesite bricks are manufactured in Greece, and though

PRODUCTION OF MAGNESITE IN CALIFORNIA.

Year.	Crude.			Calcined.		Crude Equivalent of Calcined. Short Tons.	Total Crude.	
	Short Tons.	Value at Mine.	Per Ton.	Short Tons.	Value at Works.		Short Tons.	Value.
1895.....	520	$2,600	$5·00	840	$12,180	1,680	2,200	$11,000
1896.....	489	2,300	4·70	789	11,135	1,578	2,067	9,715
1897.....	379	1,516	4·00	764	10,700	1,528	1,907	7,628
1898.....	200	800	4·00	1,013	14,200	2,155	2,355	9,430
1899.....	400	1,200	3·00	800	12,200	1,600	2,000	7,600

imported in large quantities the amount is not known, as they are not enumerated in the customs returns. For the total production in Greece see statistics under Greece in the latter part of this volume.

Imported magnesite has been utilized by the Fayette Manufacturing Co., and by the Harbison & Walker Co., of Pittsburg, in the manufacture of magnesite bricks. Some difficulty has been experienced in making a brick as uniform as those produced in Greece. Imported bricks are valued at $170 per thousand ex-ship at New York, Philadelphia or Baltimore.

Magnesite is largely used now in the manufacture of carbonic acid gas. The gas can be generated by treating the magnesite with sulphuric acid or simply heating it in a proper receptacle. In the former method the by-product is used in the manufacture of Epsom and Glauber salts, while in the latter the calcined product is made into bricks. The demand for magnesite for this purpose is steadily increasing. It has uses as a fire proofing material and there is considerable consumption in filling wall papers.

Austria.—There are important mines in Styria and at Minsan in Hungary. The largest factory in Styria is one at Mittendorf near Veitsch. A fair proportion of the production is shipped to the United States in the calcined form. The Hungarian deposits are situated near the Tolsvar railway station and are controlled by the Ungarische Actien Gesellschaft für Magnesit Producte of

Budapest. This mineral contains 94·8% magnesium carbonate, 0·1% calcium carbonate, 3·2% ferrous oxide, 1·1% alumina and 0·8% silica. The Styrian product is much more impure and carries only 88% magnesium carbonate.

Germany.—The Deutsche Magnesit Werke at Frankenstein in Silesia has constructed works for calcining magnesite and for the manufacture of bricks and fire-proof tiles for roofing and flooring. The crude magnesite has been consumed principally in the manufacture of carbonic acid gas. The mineral assays from 92 to 98% magnesium carbonate with a low silica content.

Greece.—The Société des Travaux Publiques et Communaux of Athens has a capital of $1,000,000. Its magnesite deposits are on the island of Euboea on the eastern coast of Greece. The magnesite occurs on the surface and is mined by open cuts. A narrow gauge railway connects the mines with the shipping port. The cost of producing the mineral is about $3·50 per long ton, and the transportation charge to St. Theodore or Kymassi is about $1, making $4·50 in all. Freight to the United States costs about $2·50 a ton and the selling price of crude magnesite ex-ship New York, Philadelphia or Baltimore is $7 to $7·50 per long ton. The production of crude magnesite in Greece in 1899 was 17,184 metric tons; calcined magnesite, 3,087; magnesite bricks, 542 metric tons. The greater part of the crude magnesite was shipped to the United States. An important branch of the industry in Greece is the manufacture of calcined magnesite and bricks. The works at Kymassi produce calcined magnesite and bricks. These latter are consumed mostly in Europe. Calcined magnesite is worth about $16·50 per short ton in bags, f. o. b. cars at New York, Philadelphia or Baltimore.

MANGANESE.

THE production in the United States in 1899 of manganese ores of all grades is given in the following table, which does not include the manganese iron ores of Colorado and other Western States consumed as a flux in silver-lead smelting, although account is taken of that portion sold to steel makers for the manufacture of ferromanganese and spiegeleisen. The different grades of ore include the high grade ores proper, pyrolusite and psilomelane, the low grade ores of Michigan and Wisconsin which are really manganiferous iron ores and contain but 6 to 8% Mn; and the franklinite residuum from the zinc oxide containing on the average 14% Mn.

The table in the section on the next page relating to the consumption of manganese in the United States gives the average manganese content of the ores of the different States. The composition and occurrence of the ores of manganese have been fully set forth in previous volumes of THE MINERAL INDUSTRY.

The subjoined table shows the relative productions during 1898 and 1899 of the high grade ore from Arkansas, California, Georgia, Tennessee and Virginia; of Colorado ores used for steel making purposes, of the low grade ore from Michigan and Wisconsin, and the manganiferous residuum from the zinc oxide works of New Jersey and Pennsylvania. The increase in the production of manganese ores in 1899 over 1898 was the natural result of the increased output of steel during the year. The total production, excepting a comparatively unimportant amount consumed by chemical works and glass factories, was used for the manufacture of ferromanganese and spiegeleisen.

STATISTICS OF MANGANESE ORE IN THE UNITED STATES. (IN TONS OF 2,240 LB.)

Year.	Production.											Imports.		Consumption.	
	Arkansas.	California.	Colorado	Georgia.	Michigan and Wisconsin.	New Jersey.	Tennessee.	Virginia.	Elsewh're	Totals.		Tons	Value.	Tons.	Value.
										Tons.	Value.				
					a	*b*									
1895	2,500	660	7,966	2,614	112,265	42,726	.....	1,556	.....	170,509	$898,107	66,111	$747,910	256,620	$1,076,017
1896	3,088	318	9,072	2,538	110,317	35,655	.....	1,588	.....	162,526	339,083	31,489	250,468	194,015	589,551
1897	4,490	450	18,600	992	80,961	50,000	190	2,408	c1,300	158,600	328,176	39,574	340,945	198,174	669,121
1898	2,775	398	17,792	2,477	112,318	47,470	1,250	3,307	.....	187,782	416,627	114,885	831,967	302,667	1,248,594
1899	855	966	29,161	1,638	83,702	53,921	30	3,626	75	143,956	306,476	188,349	1,584,528	331,605	1,891,004

(*a*) Manganiferous iron ore. (*b*) Franklinite residuum. (*c*) Vermont manganiferous iron ore.

The price of manganese ore advanced in 1899, and is determined by the Illinois Steel Co., according to the following schedule which is based on ores containing not more than 8% SiO_2 and 0·1% P. Deductions are made of 15c. per ton for each 1% of SiO_2 in excess of 8%, and of 1c. per unit of manganese for each 0·02% P in excess of 0·1%. Ores delivered at the works of the company, South Chicago, Ill. Settlements are based on analysis of samples dried at 212°F.; the percentage of moisture in the samples as taken being deducted from the weight.

Tenor in Mn.	Price per Unit.		Tenor in Mn.	Price per Unit.	
%	Fe.	Mn.	%	Fe.	Mn.
Over 49.	6c.	30c.	37 to 40	4c.	26c.
46 to 49	6c.	29c.	34 to 37	4c.	25c.
43 to 46	5c.	28c.	31 to 34	3c.	24c.
40 to 43	5c.	27c.	28 to 31	3c.	23c.

Schedules of prices offered by other consumers differed but slightly in detail from the above.

American Consumption and Supply.

By Alfred A. Wheeler.

The consumption of manganese ores in the United States is divided between native and imported ores. During the six years from 1892 to 1897 the native ores have only twice exceeded in metallic contents one-half of the total manganese consumption of the country. In two other years they were considerably less than one-third of the amount consumed; and their average proportion for the period named has been less than 12%. The foreign ores, on the other hand, have supplied on the average more than 58% of the total consumption.

MANGANESE PRODUCTION AND CONSUMPTION OF THE UNITED STATES. (IN TONS OF 2,240 LB.)

Year.	50% Ores from Ark., Ga., Va., Cal. & Tenn.	30% Ore from Colorado.	12% Ore from New Jersey	7% Ore from Mich. and Wisconsin.	Equivalent Total Tonnage of 50% Ore.	Imported Ores of 50% Grade.	Total Consumpt'n of 50% Grade & Equivalent.	Per Cent. of Imp'd Ore.
1895.....	7,550	7,968	42,726	112,265	87,902	86,111	124,013	69·4
1896.....	7,482	9,072	35,635	110,317	86,925	31,489	68,414	46·0
1897.....	8,440	18,600	59,000	80,261	42,836	39,574	82,310	48·1
1898.....	9,902	17,792	47,470	142,318	51,295	114,885	166,180	69·1
1899.....	6,397	29,161	53,921	53,702	44,353	188,349	232,702	80·1

This result could not be surmised from the usual statistical tables, in which it has been the habit to group together the native ores of all grades in one sum total of production. As the ores differ greatly from one another in metallic tenor they cannot properly be added together, and a statement of their total tonnage is without scientific value. The native ores are of two classes—manganese ores proper and merely manganiferous. The manganese ores since 1892 have been derived from Arkansas, Georgia, Virginia, California and Tennessee in the order named. Three-quarters of the amount mined have come from Arkansas and Georgia. For the same period the metallic tenor has been 53% in Arkansas, 45% in Georgia and Virginia, and 55% in California; and com-

puting an average metallic tenor for the whole tonnage yielded, it will be found to be but little below 50%. The whole tonnage, however, was inconsiderable. These strictly manganese ores produced less than 5% of the metallic manganese of the United States.

The remaining 95% of the country's production was derived from manganiferous ores. The iron ores of Michigan and Wisconsin carry about 8% Mn. In New Jersey the residue of a zinc ore (franklinite) has yielded an average of 12% Mn for many years. In Colorado iron ores at Leadville produce from 14 to 35% Mn. In order to compare the total product from these manganiferous sources with that of the manganese ores, it is necessary to reduce it to an equivalent tonnage of the same metallic tenor, which has been shown to be about 50%. This percentage happens to be also the average tenor of the imported ores. Cargo lots are of record which have averaged 54·56, 56·78 and even 57·50% Mn, but these are exceptions and the average commercial standard of imported ores may be taken as 50%. Applying these principles, the preceding table has been compiled, which presents, for the first time with substantial accuracy the manganese production and consumption of the United States. The native product and imported ores are in long tons (of 2,240 lb.).

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*) (IN METRIC TONS.)

Year.	Austria-Hungary.	Belgium.	Bosnia. (*b*)	Brazil. (*c*)	Canada	Chile. (*d*)	Colombia.	France.	Germany	Greece.	India.
1894.....	8,803	22,048	6,588	1,390	67	47,994	.....	32,751	43,702	9,319	11,583
1895.....	7,733	22,478	8,145	5,490	113	24,075	6,025	30,871	41,327	7,250	16,070
1896.....	5,941	23,265	6,821	14,120	112	26,152	10,608	31,318	45,062	15,500	57,783
1897.....	10,043	28,372	5,344	16,054	14	23,528	8,382	37,212	46,427	11,868	74,862
1898.....	14,219	16,440	5,320	26,417	45	20,851	11,176	31,935	43,354	14,097	61,469

Year.	Italy.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	South Australia (*d*)	Spain.	Sweden.	United Kingdom.	United States.
1894....	760·0	13,368	543	.....	142	243,469	176	340	3,359	1,688	241,613
1895....	1,569·0	17,141	213	1,240	361	203,081	49	10,162	3,117	1,298	173,237
1896....	1,890·5	17,967	66	1,494	305	208,025	*Nil.*	38,265	2,056	1,097	165,135
1897....	1,634·0	17,351	192	1,652	403	370,195	*Nil.*	100,566	2,749	609	161,188
1898....	3,009·0	(*c*)	220	907	68	(*c*)	*Nil.*	102,228	2,358	285	190,787

(*a*) From official statistics, except for the United States and Colombia, for which direct reports have been received from the producers. (*b*) Includes Herzegovina. (*c*) Statistics not yet published. (*d*) Export returns. (*e*) Shipments as stated in British Diplomatic and Consular Reports.

Austria-Hungary.—With the increase of steel manufacture at home, a decline is shown in the already small tonnage of exported manganese ore. It is probable that Austria-Hungary will shortly join the list of importers.

Brazil.—The exportation of manganese began on a very small scale in 1894. The first deposits were opened in the State of Minas and the ore was shipped by railroad to Rio de Janeiro. The principal developments have taken place in the high interior and very mountainous regions north of Rio de Janeiro and distant from the seaboard from 250 to 350 miles.

Three companies are now operating mines between Miguel Burnier and Lafayette stations on the Central do Brazil Railway near Ouro Preto. These mines are in the high mountainous regions of the Brazilian gold fields. The ores are in clays derived by decomposition from the hard rocks that originally held the

manganese and are interbedded with limestone, shales and quartzites with associated "jacotinga" iron ore. The mining is done mostly in cuts. An average analysis by Booth, Garret & Blair of three shipload lots of ores from the Usina Wigg mines at Miguel Burnier gave 53·96% Mn, 1·12% SiO_2, 0·024% P. The exportation of manganese ore from the Miguel Burnier station, State of Minas Geraes, via Rio de Janeiro was* in 1894, 1,400 metric tons; 1895, 6,765 metric tons; 1896, 13,020 metric tons; 1897, 17,967 metric tons; 1898, 29,630 metric tons; 1899 (to September) 60,107 metric tons. These figures should be increased at least 15 or 20% in order to include the manganese ores exported from Lafayette station and from Ouro Preto.

An important development has also been made in the State of Bahia, at Pedras Pretas on the Nazareth-Amargosa Railway, 26 km. from tide-water, and 74 km. west of the city of Bahia. The ore is found in clays derived by decomposition from crystalline schists, probably of Cambrian or pre-Cambrian age. Shafts have been sunk to a depth of 100 ft., but the bulk of the ore has come from open cuts at the surface of the ground. The ore is psilomelane and is found in lumps of all sizes up to a ton and a half in weight. The ore body is reported to be from 5 to 30 ft. thick, and it is thought that the surface sheet is about 20 ft. thick tapering to a thin edge. The company owns a tramway connecting with the Nazareth-Amargosa Railway, which carries the ore to the wharf at Nazareth, where it is loaded on small sailing vessels and sent to Itaparica, a small town on the Bay of Bahia, and there reloaded on sailing vessels for Europe or the United States.

The labor, railroad transportation and ocean freights to Philadelphia are all so low that the total cost of laying down the ore in Philadelphia is only $4·95 a ton. There are only two mines in the Bahia district, the bulk of the Bahia shipments coming from the Pedras Pretas mine.

The State Government maintains an inspection of the ore shipped and imposes an export duty of 7% on its market value. The following statistics of the manganese ore shipped from the port of Bahia have been furnished by J. C. Branner: 1898, 878 tons to New York; 1899, 4,521 tons to Baltimore and Philadelphia.

It has been reported lately that manganese deposits exist near Saô Vincente on the bay of Santos in the State of Saô Paulo. If good ores are found there in quantity, water transportation will place them in position to compete successfully with the Minas ores which require to be shipped by rail nearly 500 km.

Prof. J. C. Branner regards the manganese deposits of that country as destined to make Brazil one of the principal competitors in the world's markets. The ores are all oxides. Many analyses made by the Brazilian School of Mines show percentages of metallic manganese running from 40 to 62%, and Ledoux & Co., of New York, has found that 40,000 tons of Brazilian manganese ore gave the remarkable average return of 54·08% Mn, 0·03% P, 1·05% SiO_2, 0·09% Fe and 7·6% H_2O. The ports of Rio de Janeiro and Bahia place these rich yet cheaply produced ores within easy reach of the world's principal consumers.

* Information supplied by Antonio Olyntho, of Ouro Preto.

Canada.—The production of manganese ore in 1899 was 278 metric tons.

Chile.—Manganese is mined in Coquimbo and Atacama, with shipping ports respectively at Coquimbo and Carrizal. The ores occur in stratified form and are manganese oxides with occasional traces of a manganese carbonate. They average nearly 50% Mn. The silica averages 10% and the phosphorus is a little below 0·1%. In Coquimbo the ore is stratified with shales and sandstones, in Carrizal with limestone. The beds at Coquimbo vary from a few inches to 6 ft. in thickness and are remarkably continuous though of varying grades of ore. In the Carrizal district the deposits between almost vertical walls are often 20 ft. thick. Both districts are almost 50 miles east of the coast, with which they are connected by railroad. The beds of Coquimbo are about 15 miles from the nearest railroad. Delivered on shipboard the cost per ton is said to vary from $5 to $7·50.

Colombia.—Important exports of manganese ores have been made from the Isthmus of Panama since 1895. The mines are near the Caribbean Sea, about 45 miles northeast of Colon. The ores are oxides and occur in pockets imbedded in clay, which has been eroded in places, leaving isolated boulders of manganese ore. Two cargoes of ore averaged from picked samples 57% Mn, usually the ores carry excessive percentage of silica, averaging probably 18%. The silicious ore can be recognized at sight, as the silica in dry ore shows itself in minute white specks, and is also considerably lighter in weight. "By these two tests," writes Mr. E. J. Chibas, "the undesirable silicious ores are separated before shipment." The amount of phosphorus is insignificant and far less than the limit allowed.

The total exportation of manganese ore from Colombia in 1899 amounted to 10,000 long tons, which was shipped by the Caribbean Manganese Co. to the United States.

Cuba.—Manganese mining began in 1887 and three groups of deposits north and west of Santiago comprise the only manganese mines that have been worked in the island. The formation is in irregular pockets like that of the nearest deposits on the American continent. "For the most part," writes E. J. Chibas, "the ores occur as lumps of various sizes more or less mixed with clay and fragments of jasper. The large lumps can be hand-sorted, but the small ones can be saved only by washing." The ores are pyrolusite and other oxides. The average of metallic manganese has been 49% Mn, about 7% SiO_2, and less than 0·1% P. The exports of this high grade of ore for the year 1891 exceeded 25,000 tons. In 1893 and 1894 there was a marked decrease in the output in sympathy with the depressed condition of the iron trade of the United States, and for several years afterward the internal troubles of Cuba hindered development. Early in 1895 the railroad from Santiago was extended to one of the principal deposits, and the restoration of order was followed in 1899 by the purchase of several of these properties by American steel makers.

France.—There are nine manganese mines worked in France. The largest producer is the mine of Las Cabesses (Ariège) which produced in 1898 7,000 metric tons of calcined ore, and 6,400 of hand-picked carbonates. The mines

of Romanèche and of Grand-Filon (Saône-et-Loire) produced about 7,200 tons of dioxide.

Greece.—The metallic tenor of the exported Grecian ores has not been ascertained. The ores are partly manganiferous iron ores and partly manganese ores proper. One of the latter deposits occurs in stratified layers about 4 ft. thick, rising abruptly from the seashore to a height of 400 ft., and is therefore mined economically. Numerous concessions for the development of manganese properties were issued by the Greek Government in 1899, and it is believed the output of ore will soon be largely augmented.

India.—Manganese deposits were discovered a few years ago. The deposits are of irregular order and extend over a large area. On the east coast of India, in the Madras presidency, midway between Madras and Calcutta, the ore outcrops in low mounds and is found in detached pieces of all sizes from a pebble up to large boulders. The country is level and but little above sea-level. The ore consists of oxides showing a metallic tenor of about 45% Mn, but with an excess of both silica and phosphorus. Sorting the ores is therefore necessary, as well as washing the smaller pieces to cleanse them of the adherent clay. It will probably require all the assistance that India can give in cheap life and labor to make these scattered nodules profitable.

Japan.—In 1898 there was a decrease in the exportation of manganese ores due to the opening of extensive Japanese steel works, the requirements of which are likely to remove Japan from the list of manganese exporters.

Russia.—The mines in the Caucasus have been the chief source of the world's manganese supply since they were opened in 1879. The mines are situated in the District of Sharopan in the Government of Kutais and are at present the largest known deposits of manganese ore in the world. The area covered by the deposits is now sufficiently defined by working to make it probable that the bed will yet yield 80,000,000 tons of marketable ore. The total production of ore up to 1897 is given at 1,682,400 metric tons. The ore occurs in distinctly stratified deposits in a sandstone formation and crops out high on the hillsides in an almost horizontal bed interstratified with chalk, sand and other materials. The average thickness of the bed is nearly 7 ft. The ore is principally wad, mixed with other oxides and is high in metallic tenor, the exported ores, though imperfectly sorted, averaging 50% Mn. The silica does not exceed 8% and the phosphorus at present averages about 0·16%. At times the ore has been comparatively free from the latter objectionable constituent.

The physical characteristics of the ore are not favorable to commercial value; the production of large pieces is small and a great part of the ore is soft and grinds to a powder during mining and transportation, so that the loss is large; in addition, consumers sometimes object to the high proportion of fine ore present. The mines being in the mountains, the ore must first be hauled about 25 miles to a narrow gauge railroad; thence 25 miles to the main (broad gauge) line, which in turn carries it 83 miles to Poti, a port at the east end of the Black Sea. The approximate total cost delivered on shipboard at Poti is at present $6·37 per ton; delivered at English ports, $9·40 per ton, and at American ports, $9·75 per ton. Though not the only manganese exported from Russia, the

Caucasian ore is the highest in metal and the most accessible. The steel works recently established in Russia by British capital are likely to diminish the tonnage of manganese ores available for export.

EXPORTS OF MANGANESE ORE FROM POTI AND BATUM.

Destination.	1894.	1895.	1896.	1897.	1898.
Austria.....	None.	None.	None.	None.	2,300
Belgium.....	2,520	None.	220	None.	6,080
France.....	None.	150.	5,650	None.	5,100
Germany.....	51,455	59,565	58,925	75,967	111,216
Russia.....	9,890	9,600	20,175	28,446	32,070
United Kingdom.	65,110	60,616	77,754	68,650	77,950
United States.....	28,300	55,787	3,600	42,200	42,200
Totals.....	157,275	185,718	166,324	215,263	276,916

Spain.—The mines of the district of Huelva were formerly worked near the surface for manganese oxides, but the metallic tenor was not high enough for competition with the Russian ores. At deeper levels large quantities of manganese carbonates were discovered of a stated metallic content, averaging 35% Mn. The whole deposit is described by a German geologist as "a mixture of manganese carbonate and manganese silicate of great and uniform thickness." It would appear, therefore, to have the advantage of being a stratified deposit, carrying, however, only a moderate percentage of metal, and requiring the separation of silicious ore. There are no reports upon its phosphorus content.

Turkey.—A considerable tonnage of manganese of a low metallic tenor was produced at Macedonia in 1899.

The Consumers of the World's Manganese Ores.

The importers of the world's manganese are the four leaders in steel making; but as each produces manganese at home in different proportions, their importation of foreign ores bears no direct relation to their production of steel. The United States, it has been shown, produces an annual output of nearly 40,000 tons of manganese ores of the average metallic tenor of 50% Mn. No returns are available which give the percentage of metallic manganese in the native ores of England, France and Germany. Comparative prices of the native ore and the imported ore consumed in these countries during the same year show that the native ores are of low metallic tenor. With nothing but these prices as a basis of computation, the native ores of England and Germany would appear to hold not more than 12% Mn, while those of "France and her colonies" would appear not to exceed 32% Mn. The British ore is derived from a mine in Wales which yields a carbonate of manganese. The imports of manganese ores into the various countries will be found in the tables at the end of this volume.

In estimating the total consumption of manganese by the world, it has to be remembered that the exports added to the home product of the importing countries do not cover all that is consumed. At least two of the exporting countries, *viz.:* Russia and Japan, are consumers of a portion of their gross product, and two of the other countries, which do not figure either as exporters or importers, produce manganese in considerable quantity which is consumed by their steel

works. Belgium, the sixth steel producer of the world in 1897, produced from 1892 to 1896 a yearly average of 20,277 tons of manganiferous ores. They were derived from the iron ores of the Lienne Valley, which carry about 20% Mn and are smelted at Targnon, not far from Liege, where 11,391 metric tons of manganese pig iron were produced in 1896. In Sweden, for the same period of five years, an average annual product of 4,635 tons of manganese ores was all consumed in her steadily increasing production of steel. Italy also consumed at home an average of 1,254 tons of manganese ores and 7,019 tons of manganiferous ores, which was produced annually during the same period.

Manganese and Its Alloys.—The production of manganese and its present commercial value result almost entirely from its consumption by the steel makers of the world. At least nine-tenths of the world's supply is used for the manufacture of spiegeleisen and ferromanganese, and the proportion consumed for this purpose is steadily increasing. Spiegeleisen is a pig iron containing manganese in varying proportions up to 30%. It has large whitish grains which possess brilliant facets of crystallization sometimes an inch square, from whence the German name "spiegeleisen," meaning literally "looking-glass iron." Pig iron containing more than 30% Mn is called ferromanganese and is classed into two grades, "low ferro" when the manganese content is between 30 and 60% and "high ferro" when between 60 and 92%. At the present time, standard ferromanganese contains 80% Mn. The high grades require more manganese and more fuel to produce them, but they render better service than the poorer grades and on this account the low ferro and spiegeleisen are used to a much smaller extent. The present consumption by weight of standard ferromanganese (80% Mn) and of spiegeleisen (20% Mn) is about in the ratio of four to one.

The quality of "toughness" which is imparted to steel by the presence of from 3 to 20% Mn forming the so-called "manganese steel" renders it particularly adapted to many purposes where great abrasion is usually encountered, as in mine car-wheels, milling and crushing machinery, pins, car-rails on curves and switches where heavy traffic is common, and other uses of like character. Among the more recent uses to which manganese steel has been applied with very satisfactory results is in the manufacture of safes. The safe is cast in one solid piece, and the door, also of manganese steel, is grooved and fitted into the doorway so accurately that the joint is perfectly tight and explosive liquids cannot be forced into it. In finishing, the rough ordinary files and cold chisels are useless and air-driven abrasive wheels are employed.

MICA.

THERE was no great change in the mica industry of the United States during 1899. The production of sheet mica fell off somewhat in North Carolina and Virginia, and to a marked extent in New Hampshire. South Dakota produced more than it had previously.

PRODUCTION OF MICA IN THE UNITED STATES.

State.	1898.				1899.			
	Sheet.		Scrap.		Sheet.		Scrap.	
	Pounds	Value.	Short Tons.	Value.	Pounds	Value.	Short Tons.	Value.
New Hampshire	36,500	$9,125	2,875	$28,750	9,000	$4,000	2,134	$19,206
North Carolina and Virginia	71,468	80,307	644	10,987	66,087	52,726	2,388	16,750
South Dakota	950	800	*Nil.*	*Nil.*	20,299	18,000	*Nil.*	*Nil.*
Other States	2,000	2,000	100	100	2,200	2,200	2,500	15,000
Totals	110,918	$92,232	3,529	$39,837	97,586	$76,926	6,917	$50,956

Nevada.—There was no production in Nevada, although the mica in Lincoln County, near Rioville, is said to be of excellent quality and to exist in large quantity.

New Hampshire.—The mica mines of New Hampshire, operated by A. J. Hoskins, have not been as actively worked as they were in the previous year, though the production of scrap has been fairly constant. A few of the smaller producers suspended operations, as scrap mica at the mine fell off somewhat in value.

North Carolina.—While some former producers suspended operations during 1899, others have been so satisfied with the year's operations that they are contemplating an increase in size of plant, mainly to produce scrap and ground mica.

Virginia.—The production of scrap increased greatly, though the product was not valued as high as that of New Hampshire.

South Dakota.—The production of sheet mica increased, but no scrap was produced, owing to excessive cost of transportation to the market.

MICA IN FOREIGN COUNTRIES.

Canada.—(By J. Obalski.)—There has been a great development in the amber mica industry in Ottawa County this year, which is due mostly to the demand

for small size and the increase in price for the same, affording the smaller companies an opportunity to sell all their output, and to get ready cash for their work. About 35 companies or individuals have been producing more or less mica, and employing about 500 men for a period from 3 to 10 months. The total production in 1899 was 567,633 lb. of thumb-trimmed mica of different sizes, valued at $100,000, and 240 short tons of untrimmed mica, valued at $30,000, say a total value of $130,000 at the shipping point. The actual value may be considered as over $150,000. After supplying local consumption the balance is shipped mainly to the United States. The total value of mica exported from Canada for the fiscal year ending June 30th, as per customs' returns, was 1,186,185 lb., valued at $148,239.

Norway.—The muscovite mica found at the Godfjeld mine on the southwest coast of Norway forms a lens or shoot lying between quartz and feldspar or wholly in the quartz. The shoot varies in dimensions. At times it extends almost across the vein, as a whole, either between the feldspar and the quartz or entirely in the quartz. Its color is green; but the greater part of the mica has a brown coloration, due to the numerous stains, rutile enclosures or growths in the crystals. The size of the plates obtainable varied from small pieces up to sheets 12 and 24 in.

Near the surface the mica is much stained, but it improves in depth down to about 50 ft. Very frequently much disappointment is caused by the outside appearance of the crystals, as, on splitting them it was found that they were valueless owing to enclosed garnets, tourmaline, and numerous secondary growths of mica. The tourmaline crystals were found lying between the cleavage of the plates, yet often they penetrated the crystal parallel with the vertical axis. The secondary growths of mica, though usually crystalline, had no definite position of their axes in regard to those of the older crystals.

The large crystals of mica have a very interesting but at the same time objectionable peculiarity. The two basal planes are never parallel to each other, and consequently, the majority of plates split out of the mother crystal were thicker at one end than at the other. At a depth of about 60 ft. the character of the mica begins to change considerably, and although there still are crystals which yield a small quota of plates, the greater part of them are either rotten or so intergrown that they will not cleave, but break up into flakes; further, a good many crystals have undergone an entire change, a greenish, steatitic substance taking the place of the mica. This substance gives reactions for alumina, magnesia, potash and silica. In places it is compact, while in others it became earthy, or occasionally fibrous and silky in nature. It hardens shortly after exposure to the air, and does not contain either garnets or tourmaline, though the adjoining mica is full of small garnets. At the lowest place in the workings, a little below the 100 ft. level, there is still mica, but it is very rotten and practically of no commercial value.

The mica when first taken from the mine is very soft, but after a few months it becomes much harder and less pliant.

MOLYBDENUM.

THE production of molybdenum and ferromolybdenum in 1899 was more than threefold the output in 1898, the statistics for the two years being 36,000 lb. and 11,650 lb. respectively. Although molybdenum ores occur in many localities in the United States, the greater part of the supply from which the products were smelted came from Arizona and New Mexico. The increase in production was due to the greater demand for special molybdenum steels which possess the rare quality of toughness and the property of self hardening so requisite for the maintenance of hardness when the steel is heated. In this respect molybdenum acts in a manner similar to tungsten.

For the manufacture of these steels which contain up to 4% Mo, the element is added in the form of metallic molybdenum, ferromolybdenum and nickel molybdenum; ferromolybdenum contains from 50 to 55% Mo, while the composition of nickel molybdenum is given as 75% Mo and 25% Ni.

The value of molybdenum ore depends upon its richness and purity, and in 1899 was quoted at $240 per long ton for ore averaging about 50% Mo. The price in 1899 of metallic molybdenum of 95% Mo content and higher was $1·36 per lb.

There is but little to report in the technology or trade during 1899. A new deposit of molybdenite was discovered south of Lake Chelan in the Okanogan region of Washington. Its remote location and the precipitous character of the country, however, will render its production expensive. A vein of molybdenite 1 ft. wide was found below the White Pass Railroad tunnel in Alaska, containing, it is reported, $50 in gold per ton. A company has been formed in Skaguay to mine the ore.

In Sweden, molybdenite is found in proximity to several of the iron ore deposits, but has rarely been developed. The largest deposits are located at Vestanfors and on Ekholm Island. When molybdenite sold in 1896 for $0·50 per kg., the Knaben molybdenite mines in Fjotland produced 5,000 kg. annually of the mineral, but on the price falling in 1897, the work was suspended, as its mining then became unprofitable.

Molybdenum has been found recently in the vicinity of the Thames Goldfield, New Zealand, though not in quantity sufficient to work at present. Other occurrences in New Zealand are in the Tararu district and in connection with auriferous quartz at Paparoa, South Island.

MONAZITE.

THE production of monazite in the United States increased in the past year from 150,000 lb. ($7,500) in 1898 to 330,000 lb., valued at $18,480 in 1899. The production in 1897 was 40,000 lb. ($2,000); in 1896, 17,500 lb. ($875); in 1895, 19,000 lb. ($114,000); in 1894, 750,000 lb. ($45,000); and in 1893, 130,000 lb. ($7,600). Cerium nitrate is now quoted at 90c. per pound.

North and South Carolina.—(By A. E. Heighway).—While there has been a great activity in the mining of monazite sands in North and South Carolina during the past twelve months there has never been any approach to the active mining or the price realized five or six years ago. There has been a slight advance in price over that of the past two years and in consequence an increased production. There has been a more direct effort also to obtain monazite sands by agents of the manufacturers.

The Empten Chemical Co. has obtained from 50 to 60 tons in the past twelve months through purchase and its own operations. The Carolina Monazite Co., of Shelby, N. C., now has on hand 50 or more tons of monazite sand. At Cleveland mills, Cleveland County, N. C., there has been some activity in mining, particularly during the last six months. The quality has been good and the price paid for the sand has varied from 5·5 to 6·25c. per lb. It has been shipped mostly to Gloucester, N. J.

The total production for the two States during the past twelve months has been about 200 tons. The sand has averaged from 60 to 80% monazite, the balance iron, rutile and garnet. The price has averaged 5·5c. per lb. This price yields little profit to the miner. An advance would give a much larger production. No sands have been exported.

Brazil.—An analysis of monazite from Bahia shows thorium, 1·5 to 3%; yttrium, 1 to 3%; cerium, 62 to 70%; aluminum, 3%; iron, 2·5 to 5%; lanthanum, 2·5%. In 1898, 2,338 sacks and 220 bbl. of this sand were exported, nearly all to Southampton. The sacks were of heavy cotton, holding 45 kilo. A barrel contains about as much as two sacks. Owing to its high content of thorium this monazite is in great demand.

Very rich deposits of monazite have lately been discovered in the State of Bahia. They are said to be richer than the famous sands of Prado in the same State. According to the *Bulletin of the American Republics** analyses of samples have been made with the following result: One kilo of the mineral yielded 140 g. of cerium oxalate, 305 g. of thorium sulphate, 120 g. of zinc oxalate, 95 g. of cerium carbonate, 20 g. of iron and 420 g. of mica, lime and clay. For the processes in the treatment and analysis of thorium and cerium minerals the reader is referred to the article on "Rare Elements" later in this volume.

* November, 1899.

NICKEL AND COBALT.

THE entire United States production in 1899, amounting to 22,500 lb., was derived from Mine La Motte, Mo., the Lancaster Gap mines still remaining inactive. Some attempts have been made to work the Nevada properties, but as yet these have not resulted in any production. There is a steady increase in production in other countries, owing principally to the demand for nickel steel for armor plate and for industrial purposes where a hard tough steel is required. The year closed with slightly higher prices than it opened. In January the market quotation was 35c. per lb. This price was maintained steadily until October, when a sudden demand with depleted stocks raised quotations to 40c. for large orders and from that to 45c. for small lots, at which figures the market closed for the year. The contemplated expansion of the navies of several European powers as well as of the United States will doubtless maintain the demand and the prices obtained at the end of the year.

Nickel telluride has been identified in ores from Calaveras County, Cal. It is regarded by W. F. Hillebrand as identical with the melonite (Ni_2Te_3) classified by Genth, which was obtained from the same locality. To this new mineral he gives the formula $NiTe_2$.

UNITED STATES NICKEL STATISTICS.

Year.	Production.						Imports. (b)		Exports.	
	From Domestic Ore		From Foreign Ore.							
	Metallic.		Ni in Sulphide, Oxide, etc. (a)		Metallic.					
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1895.....	10,302	$3,091	1,503,216	$300,643	1,175,445	$296,861	176,541	$60,669	1,061,295	$289,897
1896.....	17,170	4,464	2,096,890	419,378	1,600,049	400,012	102,412	32,532	2,756,604	606,853
1897.....	33,700	11,668	1,459,390	505,241	2,640,000	913,968	177,392	53,018	4,255,558	997,391
1898.....	11,145	3,845	3,516,427	1,213,167	3,611,357	1,245,918	(c)	(c)	5,657,620	1,359,609
1899.....	22,500	8,156	3,008,123	1,088,652	5,045,221	1,828,893	(c)	(c)	5,004,377	1,151,923

(a) The nickel reported as in oxide is now mostly converted into metal before consumption. (b) Includes nickel oxide and nickel alloys, in which the nickel is the chief element of value, but not manufactures, ore, or matte. (c) None reported.

UNITED STATES PRODUCTION AND IMPORTS OF COBALT OXIDE.

Year.	Production	Imports.		Year.	Production	Imports.	
	Pounds.	Pounds.	Value.		Pounds.	Pounds.	Value.
1894.....	6,763	24,020	$29,857	1897.....	19,300	24,771	$34,773
1895.....	6,400	26,155	39,889	1898.....	9,640	33,731	49,245
1896.....	12,825	27,189	36,212	1899.....	10,200	46,791	68,847

THE WORLD'S PRODUCTION OF NICKEL. (METRIC TONS.)

Year.	New Caledonia.				Canada.	Norway.	United States.		World's Total.
	Prussia.	France.	England.	Total.			Domestic	Imported.	
1894	522	1,545	355	2,422	2,226	108	4	1,220	4,755
1895	698	1,545	305	2,548	1,764	17	5	1,220	4,334
1896	822	1,545	405	2,972	1,541	16	8	1,685	4,537
1897	898	1,245	715	2,858	1,813	*Nil.*	15	1,859	4,686
1898	1,108	1,540	960	3,608	2,508	(a)	5	3,234	6,116
1899	(a)	(a)	(a)	(a)	2,605	(a)	10	3,651	.....

(a) Statistics not yet reported.

NOTE.—The figures for France and Prussia in the above table are from the official reports; the amount credited to England is obtained by deducting the output in France, officially reported, from the combined output of the French and English works as given by the *Metallgesellschaft*. The actual output of all English works is somewhat greater, since there is a little Canadian ore smelted there. The Prussian figures include a small amount of nickel of domestic origin. The Canadian figures are those of the Geological Survey of the Dominion. The production of Norway is as officially reported. For purposes of comparison the production of metallurgical works in the United States is appended; it is not included in the world's total because all but a few tons of this metal has already been counted as Canadian. The world's production of nickel, as given in the above table, is probably overstated slightly, since Messrs. Basse & Selve, which is the most important nickel smelting firm in Germany, have produced some metal from oxide imported from the United States. Since the latter was derived originally from Canadian ores, its metal contents have been counted twice in the method of statistical computation employed here, the total for New Caledonia, arrived at by adding the Prussian report to that which is smelted in France and England from New Caledonia ore, is correspondingly high. The error is not, however, very important.

Canada.—According to the report of the Dominion Geological Survey, the production of nickel in Ontario in 1899 was 5,744,000 lb. valued at $2,067,840 against 5,517,690 lb. valued at $1,820,838 in 1898. The detailed statistics of Ontario compiled by A. Blue, Esq., director of the Ontario Bureau of Mines, are as follows:

Schedule.	1894.	1895.	1896.	1897.	1898.	1899.
Ore raised..... tons	113,087	75,439	109,097	98,155	123,920	208,118
Ore smelted..... tons	87,916	86,546	78,505	96,098	121,924	171,220
Per cent. nickel.....	2·92	2·67	2·67	2·08	2·26	.....
Per cent. copper.....	3·14	2·73	2·54	2·86	3·43	.....
Per cent. cobalt.....	0·0721	.....	.....	.....	.....	.....
Ordinary matte..... tons	10,410	12,525	9,733	13,706	21,101	19,109
Bessemerized matte..... tons	1,470	103¼	.....	338	.....	106
Nickel contents..... tons	2,570¼	2,315¼	1,948¼	1,999	2,284	2,873
Copper contents..... tons	2,748	2,365¼	1,868	2,750	4,187	2,884
Cobalt contents..... tons	3¼	.....	.....	.....	.....	.....
Value of nickel.....	$612,724	$404,861	$357,000	$359,651	$514,290	$526,104
Value of copper.....	195,750	160,913	180,660	200,007	268,080	176,226
Value of cobalt.....	1,500	.....	.....	.....	.....	.....
Wages paid.....	311,719	209,960	247,151	253,223	315,501	443,879
Men employed.....	655	444	485	535	637	889

(By A. McCharles.)—"The year 1899 was the beginning of a new era in the development of the Sudbury nickel mines. Early in the season Ludwig Mond, of London, England, purchased a large nickel property, comprising six claims, in the township of Denison, and the work of prospecting to ascertain the extent and character of the ore bodies was started in May, and carried on till late in the fall. Two diamond drills were used for this purpose, it is said, with satisfactory results, and from 30 to 70 men were employed at the same time in stripping the mineral outcrops, sinking two test shafts and making roads. The timber is being got out for a smelting plant with five furnaces and a capacity for treating 500 tons of ore every 24 hours. A railway track is to be built from Ranger's siding on the Sault Ste. Marie branch of the Canadian Pacific Railroad to the property, a distance of about three miles. The present intention is to reduce the ore to high grade matte in Canada, and to do the refining in England by the Mond process.

"The Lake Superior Power Co. has been exploiting the Gertrude mine in the township of Creighton throughout the whole year. A vertical shaft has been sunk to a depth of 100 ft. near the east end of the main outcrop, and a short tunnel is being driven into the ore body, which, according to diamond drill exploring is about 160 ft. wide at that point and of unusual length. There are altogether on the property three very large deposits and one auxiliary bed of high grade ore, the mine run being over 5% Ni, though the copper contents of the ore are rather low. Very few mines in the whole district have so far been found to carry more copper than nickel. A 24-drill compressor is being installed at this mine, and smelting and refining works of 1,000 tons capacity a day are being erected by the company, at Sault Ste. Marie.

"The Canadian Nickel Co., organized by some Ohio men, bought a number of mining claims and prospects in the township of Drury, and on one of these some development work is being done. At the end of the year an inclined shaft on the principal claim was driven about 50 ft., mostly through a solid body of high grade ore. The outcrop has a wide quartzite belt on one side, while the mode of occurrence of all the other contact deposits on the whole nickel range is to have syenite for one wall, with granite on the other. This company purposes to erect a smelting plant close to Worthington station.

"The Great Lakes Copper Co. began operations about a year ago on a gold and copper prospect some 15 miles north of Markstay, near Kookagaming Lake, but stopped work on it last summer, and the local agents of the company then bought two nickel properties, one in the township of Trill and the other in Blezard, about five miles to the north of Sudbury. On the latter location, in which there is some very good ore, the company erected an experimental plant to test a new process invented by Prof. Anton Graf, of Boston. This was not successful, and the company is now doing some development work.

"The operations of the Canadian Copper Co. were greatly extended during the past year. A railway track was built from the Copper Cliff mine westward to Clarabel Lake, about two miles, and two fine new mines were opened up along this line, making seven mines in all that this company is now working on a large scale. A new smelting plant, with three furnaces, was also erected a mile or so to the west of the old works and on higher ground. Both plants have a capacity of 800 tons a day. The company now employs above and below ground about 1,100 men, and the pay sheet is said to be over $30,000 a month.

"As a result of a more active demand for nickel mines, a good deal of prospecting for new properties was carried on during the season, and some valuable discoveries were made on what is locally known as the 'north range,' which extends in a sort of irregular curve from the township of Levac to Wahnapitae Lake."

Important developments according to Dr. R. W. Ells, have taken place on Calumet Island, Quebec, where pyrrhotite, carrying nickel and cobalt, occurs in diorites, gneisses, and limestones.

New Caledonia.—(E. A. Weinberg).—"Nickel ore was first discovered in 1873, at Mount d'Or. In 1880 the Société le Nickel acquired the nickel mines at Thio, but did not commence active operations until the beginning of 1887,

since when they produced from their mines some 250,000 tons of nickel ore (about 7%). A few years ago the world's production of nickel was considerably in excess of the consumption, and as there was a large accumulation of stocks, the New Caledonian nickel mining industry was practically at a standstill. During the last two and a half years, however, the demand has increased wonderfully.

"The whole nickel mining industry is now practically in the hands of the Société le Nickel, and the recently formed Nickel Corporation, Ltd. The former owns the nickel mines at Thio, Canala and Kouaoua, while the latter represent an amalgamation of nickel properties at Nepoue, of which L. Bernheim's mines are forming the center of the operations. Bernheim held originally 20,000 hectares (49,420 acres) of nickel concessions, known as the 'Si Reis,' 'Amelie' and 'Surprise,' situated on the spurs of the mountains between the Peone and Nepoue Rivers. He proved the nickeliferous ground by numerous quarries and a systematic method of prospecting, and established improved haulage facilities by building over 20 miles of tramlines to connect the mines with the deep sea port Moueo. His former holding, together with the unexplored concessions owned by the International Nickel Corporation, comprising altogether some 60,000 acres, were recently acquired by the Nickel Corporation, Ltd., a company which was floated on the London market through the agency of the London and Globe Finance Corporation, with a capital of £750,000.

"The ores of New Caledonia are found only in the serpentines, in irregular veins and fillings of the shattered surface rock, forming veritable stockworks. Although no special rules can be laid down for these ore occurrences, it must be observed that the principal nickel mining centers follow specific lines, which can be traced along the spurs of the high mountains. In this way six distinct mining areas have been established.

"(1) The great line, which for more than 100 km. (62 miles) runs N. E. from the Mount d'Or to Nakety, and which includes the mines of Dumbea, Tontouta, Ouenghi, and above all, the celebrated district of Thio, the seat of the Société le Nickel. (2) The line which runs from the bay of La Recontre to Thio, on the East Coast, and which has produced the mines of Ny, Brandy and Port-Bouquet. (3) The line from Nakety to Mount Arembo, which passes through the mines of Canala, Kouaoua and Mere. (4) The line from Mount Boa to Mount Adio, in the center of the island, known through the veins at Mount Krapet. (5) The lines from Mount Poya to Mount Kopeto, with the rich mines belonging to Mr. Bernheim and the International Corporation (now the Nickel Corporation, Ltd.). Almost without exception the mines are difficult of approach. (6) The line from Mount Koniambo to Mount Katepahic.

"There can be no doubt as to the hydrothermal nature of all the nickel veins and deposits, as the ore always consists of the hydrosilicate of nickel and magnesia, sometimes found absolutely pure, but more frequently associated with silicious gangue, red clay, or the fragments of decomposed serpentine.

"The average composition of nickel ore as it is being shipped at present to Europe, is represented by the following analysis (dried at 100°C.):

"SiO_2, 42·00%; NiO, 10·00%; MgO, 22·00%; Fe_2O_3, 10·50%; Al_2O_3, 2·50%; CaO, 1·00%; H_2O, 12·00%; total, 100%.

"Invariably, wherever nickel ore is present, the surface indications are of such a striking nature that no special skill or intelligence is required to prospect for it. The ground is always covered with iron gravel, which itself is slightly nickeliferous, and fragments of serpentine rock are met with, showing often plainly the traces of a saturation with nickel solution.

"All workings are open quarries, the length and depth of which vary with the different deposits, but in every case the life of a particular quarry is limited to a few years, when new ground must be opened out and whatever installation has been made, is shifted.

"The secret of success therefore in nickel mining operations lies not so much in the mining operations *per se*, but in the problem of transporting the ores down from the mountains to the sea board. Aërial rope-ways are used for bringing the bagged ore from the various quarries to stations alongside a tramline, which, as in the case of the Nickel Corporation, Ltd. properties, is operated by locomotives.

"The mining operations consist mainly of pick and shovel work, assisted by an occasional shattering of the serpentine rock by powder. The veins vary from an inch to several feet, but operations are as a rule slow and tedious. With the convict labor employed at present on the nickel mines, the average production does not exceed 2 to 3 tons per month per man, though in exceptional cases it may average 7 to 8 tons per month and would even reach 15 tons, provided the European smelters would lower their limit from 6 to 7% Ni.

"European buyers generally pay about 75 centimes per kilo nickel contained in the ore (with a minimum of 7%) delivered in barges alongside the loading vessel. Freights by sailing vessels vary from 32 to 34s. per ton, to Glasgow, Havre, or German ports.

"Provided the minimum of 7% has to be maintained, I doubt very much whether the production will exceed 60,000 to 70,000 tons per annum, but with a lowering of the nickel contents to say 5·5%, the production could be more than doubled.

"Although it is reported that smelting operations have been carried on successfully in New Caledonia, it is rather significant that the Société le Nickel has for years discontinued smelting operations and shipped their ores to Europe."

PRODUCTION.

Kinds.	1893.	1894.	1895.	1896.	1897.	1898.	1899.
Nickel ore	69,614	61,243	29,623	6,417	26,464	53,200	(a)
Cobalt ore	2,200	4,112	4,277	4,123	3,200	21,000	(a)

EXPORTS.

Nickel ore	45,614	40,089	38,976	37,467	57,439	74,614	101,908
Cobalt ore	520	4,156	5,302	4,823	5,395	2,373	4,997
Cobalt matte	169	7	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*

(a) Statistics not yet available.

Progress in the Metallurgy of Nickel.

The Mond process which is described in The Mineral Industry, Vol. VII., p. 526, has been further developed by the erection of extensive works at Southwick, England. Storer treats ores containing silicate, hydrate or oxide of nickel with ferrous chloride solution. One part of ore with 2·75 parts of solution containing 26·5 parts of the iron salt is placed in a closed vessel heated to 187°C. under pressure of superheated steam. After five to eight hours' digestion double decomposition has occurred, leaving an iron oxide as a precipitate and the nickel in solution from which it may be separated by electrolysis or precipitation. About 750 lb. of ferrous chloride are consumed to the ton of ore.

The American Nickeloid Co., of Peru, Ill., has been formed for the purpose of manufacturing an alloy of zinc and nickel, which is said to have greater tensile strength and to resist corrosion better than metallic zinc, at the same time being much less costly than sheet brass or copper.

From experiments lately conducted at one of the Reichsanstalten in Germany, on nickel-iron alloys, it has been developed that the coefficient of expansion of such alloys varies according to the percentage of the two metals present. Furthermore, it is said to have been experimentally demonstrated that an alloy of iron and nickel containing 45% Ni possesses the same rate of expansion as platinum, and it is proposed to use this alloy to replace platinum in the manufacture of incandescent electric lamps.

A. E. Fulton* gives the results of the investigation on the thermal expansion of pure nickel and cobalt. Nickel with a lower atomic weight expands to a greater extent than cobalt.

The Atomic Weight of Nickel.—T. W. Richards, and A. S. Cushman describe† their methods of determining a new value for the atomic weight of nickel, which differs slightly from those found by Winkler, and Zimmermann. Winkler's results, corrected to oxygen at 16·00 instead of 15·96 as used by him, together with Zimmermann's agree very closely with the author's results, and also to Clark's mathematical selection from the values obtained by various investigators. These results were: Winkler's corrected value, 58·69; Zimmermann's value, 58·694; Clark's selection, 58·687; and Richards' and Cushman's value, 58·706. The general average therefore, is 58·70, which is exact enough for any practical purpose.

Analytical Methods.—Geo. Wm. Sargent‡ describes a method for determination of nickel in nickel-steel which has been adopted in the laboratory of the Carpenter Steel Co., and which has given extremely satisfactory results. The method in detail is as follows: Dissolve 2 g. of the steel in hydrochloric acid (1·1 sp. gr.), add 1 c.c. of strong nitric acid to oxidize the iron, and evaporate to hard dryness. Take up the residue in 20 c.c. hydrochloric acid (1·1 sp. gr.), evaporate to 10 c.c. volume, and transfer the solution to a 250 c.c. separatory funnel. Use warm hydrochloric acid (1·1 sp. gr.) for washing the contents of

* In a paper read before the Royal Society, May 4, 1899.

† *Proceedings of the American Academy of Arts and Sciences*, Vol. XXIV., No. 13, February, 1899.

‡ *Journal of the American Chemical Society*, October, 1899.

the beaker into the funnel, taking care to keep the volume as small as possible. Cool the funnel and its contents under the spigot, then introduce 40 c.c. of ether free from alcohol, which has just been thoroughly shaken with 5 c.c. of strong hydrochloric acid, and shake vigorously for 10 minutes, keeping the temperature from rising by frequently holding the funnel under the running water. Allow the funnel to stand a few minutes, draw off the lower aqueous layer containing the nickel, copper, manganese and other chlorides not held by the ether, wash down the sides of the separatory funnel with hydrochloric acid (1·1 sp. gr.), shake with the ether, and run the washings into the beaker with the chlorides of nickel, copper, etc. Two washings with 5 to 10 c.c. of hydrochloric acid are sufficient to remove completely the nickel and other chlorides from the funnel. The ethereal solution containing the ferric chloride is run into a residue bottle and later the ether recovered by distillation. Boil the liquid containing the nickel to expel completely the ether, and add to the boiling solution, diluted to at least 200 c.c., an excess of ammonia and 10 to 20 drops of bromine water to precipitate any iron or manganese. Filter, wash, re-dissolve in hydrocholic acid, precipitate with ammonia and bromine water, and filter. Combine the filtrates, boil, remove and precipitate, acidulate with hydrochloric acid, boil until the excess of bromine has been expelled, and precipitate the copper as sulphide. The filtrate from the copper sulphide which contains nickel only is evaporated to a volume of 100 c.c., cooled, and 1 c.c. excess of ammonia added. Now introduce into the nickel solution, which is best contained in an Erlenmeyer flask, 5 c.c. of silver nitrate (0·5 g. silver nitrate in 1 liter of water), and the same amount of a 2% solution of potassium iodide. Run into the opalescent solution, which should have a temperature slightly lower than that of the hand, standard potassium cyanide (1 c.c. equal to about 0·001 g. of nickel) until the liquid becomes clear and bright. This titration is best made with a black background, when the end-reaction becomes very sharp and decided after a little practice, half a drop only being sufficient to discharge the opalescence.

The potassium cyanide is standardized by introducing into an Erlenmeyer flask 10 c.c. of a solution containing a known amount of recrystallized nickel nitrate, $Ni(NO_3)_2$, $6H_2O$ 10 c.c. of hydrochloric acid (1·1 sp. gr.), 1 c.c. excess of ammonia, 5 c.c. each of the silver nitrate and potassium iodide, diluting the whole to 100 c.c., and running in the potassium cyanide until the solution clears. It is necessary that a blank be made upon the silver nitrate and potassium iodide used as the indicator, under the same conditions as the standardization, and this amount deducted from each titration.

The presence of varying quantities of ammonium chloride in the nickel solution to be titrated has no effect on the titration.

For the estimation of nickel in ferro-nickel, dissolve 20 g. of the sample in aqua regia, evaporate to hard dryness, take up with hydrochloric acid (1·1 sp. gr.), transfer to a liter flask, and dilute to the mark with water. Remove 50 c.c., to which add 5 c.c. strong hydrochloric acid, and concentrate to one-sixth its volume. Transfer to the separatory funnel, and proceed as with the steel.

Nickel and Iron Alloys.

By Henry Souther.

Nickel steel at this time occupies a very prominent position in the eyes of the engineering profession, offering, as it does, such immense possibilities toward a gain in strength of structural materials, without any serious loss in toughness or ductility. Nickel unites with iron and the ordinary impurities which are always associated with iron, and forms alloys whose physical peculiarities and characteristics differ from any other alloy now known.

In the class of alloys of this metal where the percentages are fairly high, results are obtained that are likely to prove valuable in the arts for the manufacture of fine instruments, and other special uses. The practical facts in regard to the alloys of iron and nickel have been most completely collected by Mr. Hadfield and presented in a paper entitled "Alloys of Iron and Nickel," which was read before the Institution of Civil Engineers in England. And another collection more recent has been presented by Mr. Browne of Cleveland before the meeting of the American Institute of Mining Engineers at Cleveland in 1899.

It is difficult to condense the facts presented by the above writers without losing much that is of vital importance and of the utmost value. There are, however, certain prominent and practical features in regard to this alloy that may be singled out as most important.

Methods of Manufacture.—Alloys of nickel and iron (nickel steel) are commercially made by two methods, by the crucible process and by the open hearth process. In both methods the alloy is cast into ingots and forged as any ordinary carbon steel is. Ingots cast from the open hearth process that are intended for armor plate, gun tubes and large forgings, are of immense size. Krupp, in Germany, makes large ingots for these same purposes, by the crucible process, pouring large numbers of crucibles simultaneously. The latter method is not considered any better practice by American manufacturers (nor perhaps as good) than the making of the steel in open-hearth furnaces, which gives a homogeneous material, the equal of any, if one may judge from the tests of the product. The addition of nickel in crucibles is in the form of nearly pure metallic nickel, running as high as 98 or 99% Ni.

The accompanying table* shows the composition of the most common metallic nickels used for alloying purposes:

COMPOSITION OF METALLIC NICKEL USED FOR ALLOYS.

Components.	Ferro Ni.	Mond Ni.†	Thompson Ni.
	%	%	%
Carbon	0·16	0.05	0·16
Silicon	0·09	None	0·09
Sulphur	0.016	0 ·006	0·016
Copper	0·11	none	0·11
Tin	0·11		0·11
Arsenic	0·11		0·11
Iron	0·60	0·13	0·60
Nickel	98·80	99·80	98·80
Cobalt		None	

*Gathered from Hadfield's paper on alloys of iron and nickel. †Mond nickel not now produced commercially.

For the open hearth process in the United States nickel oxide is mostly used. This contains about 76% Ni with about the same amounts of impurities as the metallic nickel, which is reduced from the oxide. By the use of the oxide the steel maker saves the additional cost of production of the refined nickel (amounting to about 8c. per pound of nickel produced) and as the nickel oxide is reduced by the carbon in the steel bath, this form of nickel seems the most rational one to use. There is some loss of nickel in the slag, but nothing serious. The oxide is added shortly before the heat is tapped from the furnaces, becomes thoroughly mixed with the steel in the furnace itself, and being then run into a ladle produces a mixture that cannot be other than a perfectly homogeneous one.

The increase in cost of nickel steel over carbon steel is in proportion to the per cent. of nickel aimed at. At the present price (about 35c. per pound metallic nickel) the increase is about one-third cent per pound of metal produced,* or one cent per pound steel if a 3% alloy is used.

During rolling and hammering operations there is rather more loss by scaling than with carbon steel, which increases the above figure to a slight extent, but not seriously. The addition of nickel to steel has no quieting or solidifying effect on the cast steel, and the usual doses of silicon or aluminum are therefore added for this purpose. Nickel does not segregate in the ingot; in fact, nickel seems to retard the ordinary segregation of sulphur and phosphorus.

Classification.—Laboratory research has so far very plainly shown that the alloys of nickel divide themselves into three classes, each characterized by special physical qualities.

The result of the tests made and graphically presented by these curves† are most conclusive as to the physical characteristics of iron-nickel alloys being absolutely consistent throughout. The actual composition of the alloys used is given by the following table:

Specimen Mark.	Actual Analysis.						Intended Percentage of Ni.
	C.	Si.	S.	P.	Mn.	Ni.	
	%	%	%	%	%	%	
A	0·19	0·31	0·10	0·09	0·79	0·27	0·25
B	0·14	0·20	.....	.....	0·75	0·51	0·50
C	0·13	0·23	.....	.....	0·72	0·95	1·00
D	0·14	0·21	0·08	0·07	0·73	1·92	2·00
E	0·19	0·20	.....	.....	0·65	3·82	4·00
F	0·18	0·31	.....	.....	0·65	5·81	6·00
G	0·17	0·28	0·11	0·08	0·68	7·65	8·00
H	0·16	0·20	.....	.....	0·86	9·51	10·00
I	0·18	0·22	.....	.....	0·93	11·39	12·00
J	0·23	0·24	0·11	0·05	0·93	15·48	16·00
K	0·19	0·27	.....	.....	0·93	19·64	20·00
L	0·16	0·30	.....	.....	1·00	24·51	25·00
M	0·14	0·38	.....	.....	0·86	29·07	30·00
N	0·16	0·31	0·08	0·06	1·08	49·65	50·00

These steels are unusually high in sulphur and phosphorus but being all about the same are consistently comparable.

I. Alloys of iron low in nickel, containing from 2 to 8% Ni.

* David H. Browne, *Transactions of American Institute of Mining Engineers*, September, 1899.

† R. A. Hadfield, *Proceedings of the Institution of Civil Engineers*, March 28, 1899.

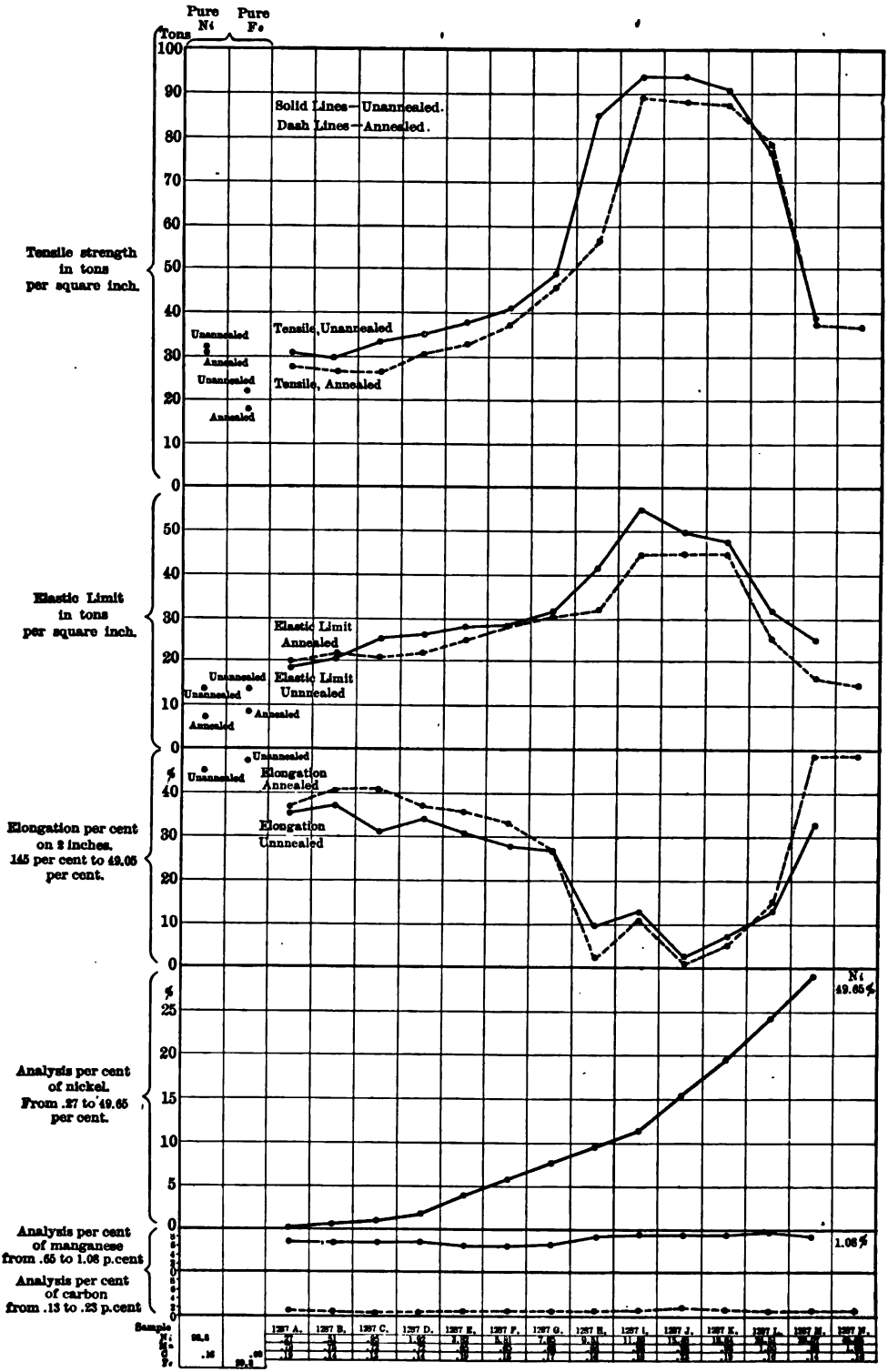


FIG. 1.—CURVES SHOWING THE PHYSICAL CHARACTERISTICS OF IRON-NICKEL ALLOYS.

II. Alloys of iron containing from 8 to 22% Ni, commonly classed as the brittle alloys of nickel.

III. Alloys of iron and nickel containing from 22 to 40% Ni.

The alloys of Class II. require but little comment. They seem to be of such a nature that they will be of little practical value and can serve only in adding to our knowledge of the behavior of metals as alloys in general.

Class III. is of importance and at present in the following lines. This class of nickel alloy resists corrosion better than any other metal, that is as strong and tough. Its color is good and it may be easily worked either hot or cold; 25 to 26% nickel steel becomes quite soft when quenched in water from a low red heat, and works in the machinist's hands very much like German silver; 24 to 28% nickel steel makes what is considered the best of resistance wires in rheostats, as overheating does not change its physical or electrical characteristics, as is the case with German silver. Instruments of precision made of alloys in this class may be light, yet strong and stiff, and of a high degree of non-corrosibility. In some instruments the exceedingly low coefficient of expansion of one of these high alloys is likely to be of extreme value. For example, referring to the table of coefficients of expansion in Mr. Browne's paper, it is found that 36% nickel steel expands only 4·6% as much as brass, 7% as much as hard steel, and only 10% as much as the lowest figure given for any of the common materials. It would seem as though this alloy would be better than anything now in use for our standard bars of length.

COEFFICIENTS OF EXPANSION.

Brass	0·00001878	31·4% nickel steel	0·00000840
Copper	0·00001718	34·6% nickel steel	0·00000137
Soft steel	0·00001078	35·6% nickel steel	0·00000087
Hard steel	0·00001239	37·3% nickel steel	0·00000856
Nickel	0·00001252	39·4% nickel steel	0·00000537
26% nickel steel	0·00001312	44·4% nickel steel	0·00000856
28% nickel steel	0·00001131	Platinum	0·00000884
28·7% nickel steel	0·00001041	Glass	0·00000861
30·4% nickel steel	0·00000458		

Influence of Nickel.—The alloys of Class I. stand out prominently as being the most important class of the nickel iron alloys; 2 to 8% Ni added to good steel produces a material that has no rival among structural materials. What the nature of the influence of the nickel on the steel is, seems much in doubt. Mr. Hadfield's view is expressed by him as follows: "As already pointed out, it is probable that the special advantage of the use of nickel is due not so much to the properties it confers upon iron in carbonless, or nearly carbonless iron alloys, as to its modifying influence upon iron in the presence of carbon when it probably forms a special carbide of nickel."

And again in another place he says: "Experimenters can only leave this point with the surmise that nickel, in its remarkable toughening action upon iron, causes a very intimate combination of the molecular structure. This, perhaps, may be supported by the fact that nickel does not show any disposition to segregate in steel like other elements; in other words, it appears to be more intimately combined."

The writer has several micro-photographs of nickel steel containing about 5% Ni, and all show a more regular, finely divided and homogeneous make-up than any carbon steel of the same composition and treatment. It is a fact that fineness of grain is accompanied by strength and toughness combined, and is the most desirable quality to strive for in our treatment of all steels. Nickel steel seems to possess it inherently.

All the evidence in regard to the effect of nickel on steel seems to point to the fact that it is a happy combination of nickel and carbon, with probably manganese in the usual quantities acting in a beneficial direction. The other elements usually present—phosphorus, sulphur and copper—are certainly detrimental. The effect of silicon is in dispute, some prominent metallurgists strongly favoring it up to a half per cent., others believing that if possible it should be nearly if not entirely absent, except in quantities just sufficient to quiet the steel in the moulds.

It is the opinion also that pure nickel added to pure iron would not give the wonderfully fine results that are obtained when using our commercial steels. This question, however, is only of theoretical importance, as anything other than impure iron is practically unobtainable.

The opinion of Mr. Maunsel White of the Bethlehem Iron Co. in regard to the general effect of the mixtures of nickel and steel is most valuable and is given as follows:

"The tensile strength and elastic limit of nickel-iron alloys and nickel steel rise with increasing proportions of nickel, reaching a maximum at about 20%. Passing this they begin to fall, and elongation increases abnormally up to 30%. The ratio of elastic limit to tensile strength increases with increasing nickel, at first slowly up to about 10%, where it makes a sharp rise, and then continues gradually up to 20%, after which the ratio falls away rapidly.

"The hardening effect of quenching ceases at about 10%, but is quite marked in the lower percentages. In this case the effect is heightened by the manganese, but with 0·06% Mn it is still decided. Between 10 and 20% Ni, neither quenching nor annealing exerts any decided effect, the variations being not more than 1 or 2%, practically within the limits of error of testing. Above 20% quenching produces a softening effect which is decided at 30%. Perhaps it would be better to call it a weakening effect, as the tensile strength and elastic limit are much lowered, the elongation increased, but the cutting properties shown by turning in a lathe are not perceptibly changed.

"Nickel tends to check segregation, due, probably, to the fact that it raises the melting-point of the carbides or cementing material, and causes the whole mass to set more nearly together. This is evidenced by the finer grain of nickel-steel ingots.

"Nickel does not prevent blow-holes, and great care is required in the manufacture to avoid them, as they are less amenable to welding than in other steel.

"There are many difficulties to be overcome in the handling of nickel steel as commercially made, where the carbon ranges from 0·15 to 0·90%, the manganese from 0·30 to 0·90%, and the nickel from 3·0 to 30·0%. It is very susceptible to changes of temperature when containing the usual amounts of

carbon and manganese, requiring considerable care in heating and working to bring out its best qualities, which are certainly worth all the labor bestowed upon it.

"The fact that nickel raises the elastic limit of steel is easily proved, but to give a reason for this effect, which can be definitely proved, would be difficult. The best explanation is that it strengthens the cementing material up to a certain point (20%), and holds in a more rigid matrix the iron crystals which in nickel steel exist in smaller aggregations; beyond this the cementing

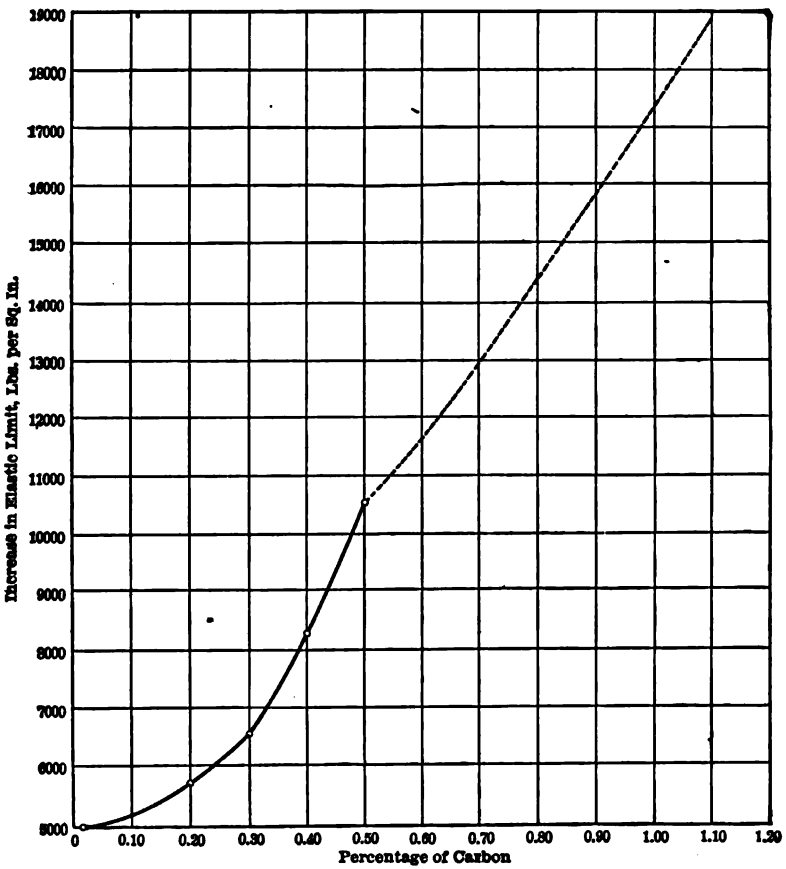


FIG. 2.—APPROXIMATE CURVE OF INCREASE IN ELASTIC LIMIT DUE TO ONE PER CENT OF NICKEL.

material is not so much strengthened as it is toughened, which accounts for the abnormal increase in elongation."

Perhaps the most important and vital point brought out by Mr. White is that in relation to the care in heating and working that is necessary to bring out its best qualities. It must be remembered in using nickel steel that it is far more sensitive than ordinary carbon steel, and may in this respect be compared to the most delicate of tool steels.

Physical Characteristics.—The practical commercial gains in strength that

are obtained with nickel steel are given in the table presented by Mr. Browne in his discussion of Mr. Hadfield's paper, and graphically by Fig. 2.

Character of Steel.	Elastic. Lb.	Tensile. Lb.
Average of all soft low-carbon steels show a gain due to 1% Ni of about	5,000	4,000
Bethlehem Iron Co. oil-tempered, annealed forgings, 0·20% C, 3·5% Ni, gain due to 1% Ni	5,714	8,571
Bethlehem Iron Co. oil-tempered, annealed forgings, 0·20% C, 3·5% Ni, gain due to 1% Ni	6,571	5,714
Bethlehem Iron Co. oil-tempered, annealed forgings, 0·40% C, 3·5% Ni	8,285	7,142
Bethlehem Iron Co. oil-tempered, annealed forgings, 0·50% C, 3·5% Ni	10,570	8,571
Crescent tool steel, 1·10% C, 1·65% Ni	18,957	20,484

This curve is most interesting as it shows so plainly that the strengthening effect of nickel on the elastic limit increases as the carbon increases. The tests have not been carried into the high carbons (tool steels) in sufficient numbers to prove conclusively that the increased effect extends to tool steel. But the one test on record (that of Mr. McDonald of the Crescent Steel Co.) seems to indicate that it does.

Composition of Nickel Tool Steel.	Elastic Limit.	Tensile Strength.
No Ni, 1·10% C	57,700	112,452
1·65% Ni, 1·10% C	88,980	146,250
Gain due to 1·65% Ni	31,280	33,798
Gain due to 1% Ni	18,957	20,484

These results indicate that nickel tool steel should be of an exceedingly strong and good wearing character.

Nickel Steel Forgings.—H. F. J. Porter of the Bethlehem Iron Co. expresses himself as follows: "Although forgings can be made to fill a large variety of specifications, they can, in general, be divided into six classes as follows: (1) Mild steel, annealed; (2) medium hard steel, annealed; (3) medium hard steel, oil-tempered; (4) nickel steel, annealed; (5) nickel steel, oil-tempered, No. 1; (6) nickel steel, oil-tempered, No. 2. Each of these classes is supposed to cover a series of grades of steel, varying in strength several thousand pounds. In selecting the materials for the forgings of an engine and in drawing up the specifications therefor, the premise should not be omitted that 'all forgings shall be made of open-hearth steel,' and that 'they shall be carefully annealed after forging.'

"Large shafts and similar forgings, crank and cross-head pins, should be made of fluid-compressed steel, and should be hydraulic-forged, not hammered. Wherever practicable, an axial hole should be bored through shafts to insure absence of any internal defects. If forgings are oil-tempered, the hole can be made larger in diameter than if they are simply annealed, and where the hole is seven inches in diameter and above, they can be hollow-forged on a mandrel. A hollow-forged oil-tempered forging insures the highest attainable qualities, and can be specially recommended where the maximum strength with the greatest lightness is desired.

"Where it is important that the quality specified should be obtained in the more important parts, physical tests of the forgings as delivered should be

demanded. For such tests prolongations should be left on the ends of forgings for the purpose of having test specimens cut from them after forgings and treatment have been complete. Such prolongations should receive no greater reduction than the forging at its largest part."

A table follows which gives actual obtainable results:

	Carbon Steel.	Tensile Strength. Lb.	Elastic Limit. Lb.	Extension Percentage.	Contraction Percentage.
1	Annealed	58,000	28,000	28	55
2	Annealed	80,000	37,000	23	45
3	Oil-tempered, with axial hole	80,000	45,000	25	50
	Nickel Steel.				
4	Annealed	80,000	45,000	23	45
5	Oil-tempered, with axial hole	80,000	50,000	25	50
6	Oil-tempered, with axial hole	90,000	60,000	22	50

Most noticeable in this table is the fact that an immense increase of strength is possible without loss of toughness, as shown by little or no decrease in the contraction percentage column. The elongation may drop somewhat without important detraction from the good qualities of a structural steel. If the contraction at the break under tension test is maintained at a high point, it is certain that the steel is capable of resisting alternate stress to a high degree.

Effect of Nickel in Elongation and Contraction of Area.—Nickel steel in its annealed condition as shown is stronger than carbon steel of the same composition, with the exception of the nickel, and does not lose in percentage of elongation and contraction of area in the same proportion as it gains in strength. By proper heat treatment (oil quench and oil temper) the reduction of area can be maintained quite as high as in the annealed steel even in the face of a 50% increase of strength. As the reduction of area is the surest indication of the amount of punishment a steel will stand this is most important and is being taken advantage of in the manufacture of engine shafts and gun tubes.

As an illustration of the practical results obtainable, and the great saving in weight possible, the shaft of the United States Steamship Brooklyn is given. As it is, it weighs 36 tons and is made of nickel steel. To have as great strength in carbon steel the weight would have to be 53% greater. Similar reductions in weight are being made in torpedo craft, boilers, automobiles, and other fine pieces of engineering construction.

With our present knowledge of the importance of the critical temperature of steel, it seems unnecessary to heat a steel for annealing or hardening much above it. This being the case it is plain that 1,200°F. is quite high enough to yield the best results for a 5% nickel steel. Practice has borne out theory in this case most perfectly. During the manufacture of nickel steel tubing by cold drawing it was found that greater reductions could be given the tubing after annealing at about the above temperature than if annealed at a higher temperature. Under tension test greater elongation was obtained with a high annealing point but not as great reduction of area at break.

A steel that will not show good reduction of area will not admit of rapid reduction under cold drawing operations even if the elongation is high.

Aluminum at first glance is very attractive, but will not do as much in the saving of weight for a given strength, and, in addition, is positively dangerous because of its extremely bad behavior under fatiguing strains. The following table, gathered from isolated figures, is given to show what may be depended upon as the strength of nickel steel in its various uses:

	Lb. per Sq. In. Elastic Limit	Maximum Strength.	Per cent. Elongation.	Per cent. Reduction of Area.
Nickel steel, boiler plate	61,880	87,380	21·7	52·2
Carbon steel, boiler plate	31,840	60,650	27·0	49·5
Nickel steel propeller shaft	60,000	94,500	24·6	56·5
Torpedo boat propeller shaft*	68,700	101,000	22·12	
Forgings	57,000	91,000	25·05	56·45
Wheel tires	55,000	87,000	28·7	46·0
Quenched 5% nickel steel bicycle handle-bars†	200,000	240,000	4·0	

* Smaller with higher temper. † Can be flattened to one-half diameter.

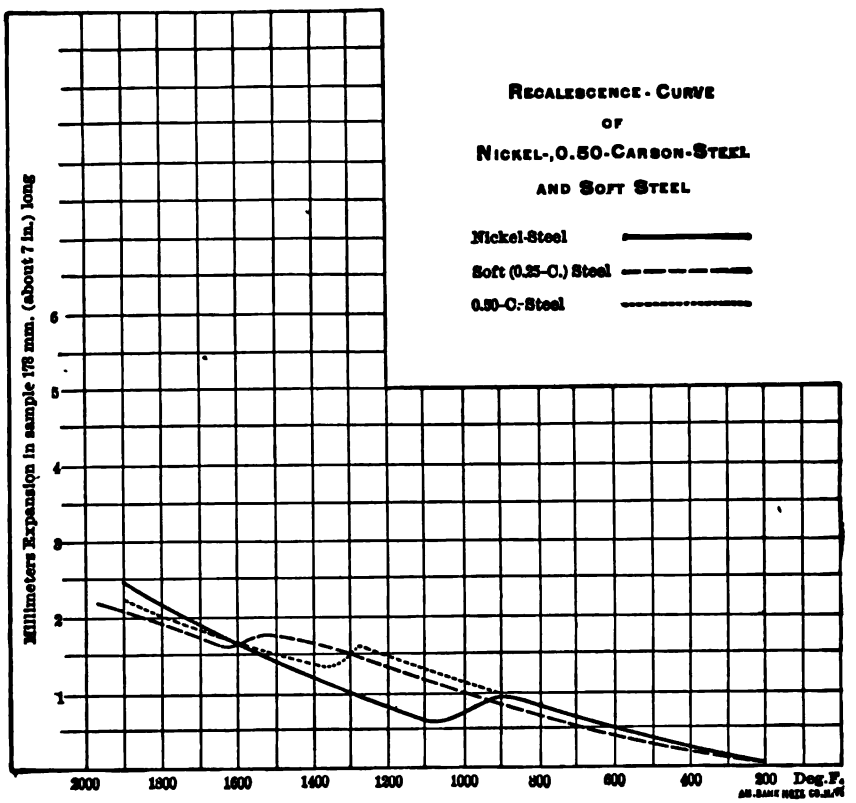


FIG. 3.

Partial annealing after quenching puts nickel steel in its very best condition and is resorted to in gun tubes, hollow shafts, and other forgings. Mr. Hadfield speaks of this, but uses the term "anneal" without qualification, which will probably mislead the general public, inasmuch as the word "anneal"

usually means a treatment which renders a steel as soft as possible. This is certainly not the result aimed at by those treating the highest class forgings.

Nickel steel has caused much difficulty with those using it because of its behavior in machining. Small masses of nickel steel become excessively hard and tough, but a suitable annealing will in every case give a clean, easy-cutting metal. Over-annealing, that is, too high a heat, will give a stringy, bad-cutting metal, which is quite as troublesome as before annealing at all, although not hard.*

Alternate Stress Tests.—It has been thoroughly demonstrated that nickel steel withstands fatiguing strains some four or five times as long as carbon steel of similar composition. The quenched specimens followed by partial annealing show the greatest efficiency in this respect.

The apparatus used for these tests is a lathe with ball joints at head and tail stock. The specimen (33 in. between centers) is gripped by these so that it is free to bend when loaded at the center. The load is hung at the center, resting on rollers, and cushioned by graduated springs to take up what little shock there might be. Enough load is usually added to produce a fiber stress of about two-thirds the tensile elastic limit as determined from a piece cut from the same specimen.

Rotation under this stress is continued to rupture at about 400 revolutions per minute. The comparative results are as follows:

	Revolutions.		Revolutions.
0·10% C ordinary steel	100,000	0·50% C ordinary steel	400,000
0·25% C ordinary steel	200,000	0·25% C. 5% Ni	1,000,000

The results given by Mr. Hadfield as to the use of nickel steel for fire box stays and stay bolts show the steel to be most capable of resisting the alternate stress undoubtedly brought upon such parts, but it also brings up a point in regard to nickel steel which must be carefully watched—that is, the tendency to set up electrolytic action when used with carbon steel.

Electrolytic Action.—The writer begs to call attention to the following results of a laboratory experiment made at a time when the possibility of using a 5% nickel steel for boiler tubes was presented.

A 4·5% nickel tube was put into the same vessel with a tube of 0·45% carbon steel, in pure water. Evaporation to dryness took place once a day. At the end of 15 days the specimens were carefully washed. A distinct difference in the surface of the two metals where not covered with oxide was noted, the carbon steel being very bright and clean, the nickel steel dull. Electrolytic action was set up between the carbon and the nickel steel; the carbon losing to the nickel. No weights were possible on account of the thick scale in spots. Surface inspection, however, showed plainly a pitting of the carbon tube. The

*The most important point in connection with the use of nickel steel is the degree of annealing necessary to get the best results. The writer notes that Mr. Hadfield says his annealing has been done in the usual way, at about 1,800° F., and that the steel has been left in the furnace three or four days. It is the opinion of the writer that 1,800° F. is too high to get the best results, and that several days is unnecessarily long a period for cooling. It has been the experience of the writer that from 1,200 to 1,300° F., just above the point of recalescence; heating, long enough continued to warm the mass throughout, and cooling just slow enough to avoid chilling, gives the best combination of strength and ductility to the nickel steel.

same experiment was tried in other solutions; salt water, dilute stearic acid and dilute tannic acid. The results were the same, the salt water leaving the cleanest surface.

These results show a decided danger and that nickel steel should not be used with carbon steel in the same structure unless some means are taken to prevent or neutralize the electrolytic action. Rivets of nickel steel are stronger than ordinary rivets, but for the above reason are of doubtful utility in carbon steel plates.

There is little evidence that nickel steel included in Class I. corrodes less than ordinary carbon steel. Lack of non-corrosibility should certainly not be reckoned on in connection with ship plates, boilers, or similar uses.

Electrical Properties.—The electric permeability of Class I. nickel steel is better than wrought iron, and it has been used in instances where great strength was needed in connection with electric machinery, the illustration quoted being that of the electric generators at Niagara Falls.

The conductivity of nickel alloys of all classes is shown by the following diagram of curves, both for annealed and unannealed specimens.

(To anneal the specimens were raised to about 1,000°C, (1,832°F.) and

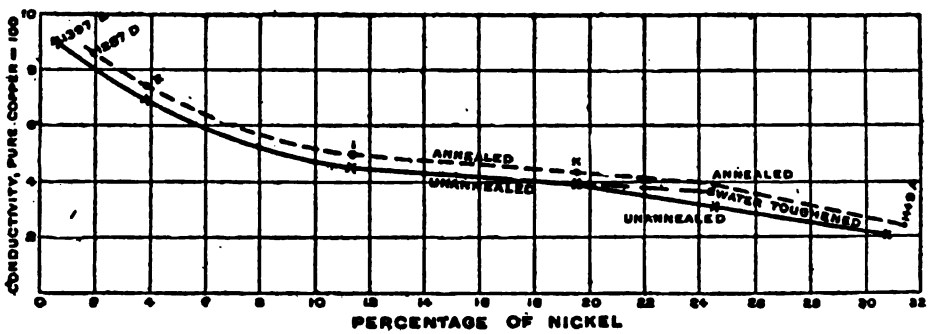


FIG. 4.—CURVE SHOWING THE CONDUCTIVITY OF NICKEL-IRON ALLOYS.

allowed to cool four days and four nights.) It is seen that the annealed specimens were always the higher in conductivity.

Welding Properties.—Nickel steel certainly welds less readily than carbon steel, but the steels of Class I. may be made to "stick" in such a way that the use of nickel steel for any purpose where welding is necessary need not be abandoned for that reason only.

Effect of Nickel on Hardness.—Nickel added to low carbon steel makes it harder, yet not harder to the file in large masses, even if quenched. But to machine nickel steel easily it must be carefully annealed after forging. In thin objects, like a bicycle tube, it becomes hard even to a file if quenched in oil or water and often in cold air.

Nickel seems to accentuate any hardening tendency a steel may naturally have. (This refers to nickel alloys of Class I.) It is sensitive to cold and heat and must be treated accordingly.

The use of nickel steel in bicycle tubing was abandoned after thorough

trial, as it exhibited a tendency to tear while heated in the brazing operation and, in addition, it chilled rapidly on removal from the brazing fire and prevented further mechanical operations.

Could bicycle frames be joined without brazing there is no question but what nickel steel would make the best material known at the present time.

Uses of Nickel Steel.—As has been indicated in the foregoing pages, nickel steel is being used for a wide variety of high class work and is daily being tried for new purposes. Perhaps most important is its use in armor and heavy ordnance. Its great strength and toughness make it invaluable for these purposes. It is also used for our modern high power rifle barrels in large quantities and resists the erosion of the modern powder better than other steel.

For engine and propeller shafts it has no rival as it withstands the alternate stress and shocks to which these objects are exposed, so much better than any other material. These parts are all submitted to heat treatment, to increase their strength or to obtain a resulting decrease in weight, so vital to torpedo craft and high speed steamships and engines in general. Piston rods, crank pins, light-forged engine frames, bolts for extreme hydraulic pressures, hydraulic forged cylinders, railway axles, are some of the parts into which nickel steel is put to the utmost advantage because of increase in strength or decrease of weight.

Its use for boilers, hull plates, boiler tubes, rivets, has not been practically demonstrated a success as yet in cases where carbon and nickel steel are used in the same object. Some boilers are now in use of composite structure and promise good results. The increase in strength is considerable, although not as great as in parts where heat treatment may be resorted to.

For bridge work little or no nickel steel has been used; probably because of increase of cost, which is not compensated for by any valuable or very desirable decrease in weight.

Nickel steel has proved a great success for steam hammer piston rods and for rock drill piston rods. This is because it resists intense vibration so well.

For certain tubular parts of a bicycle it is invaluable as a saver of weight and as furnishing a means to an end hitherto impossible.

The use of nickel in tool steel is in somewhat of an experimental stage. Certain lots of it have been manufactured that are not excelled by any tool steel, but absolute uniformity is in doubt.

An alloy of about 0·80% C and 3 to 4% Ni makes the best tool steel for all-around work. Higher machined increases the strength but makes the steel so sensitive as to be difficult to work.

Nickel steel of Class III. enters the market most successfully as resistance wire.

OCHER AND OXIDE OF IRON PIGMENTS.

WHILE the production of all mineral paints (including the previous year's products of ocher, umber, sienna and iron oxide) increased in 1899 from 41,950 short tons to 42,286 short tons, the importation of all kinds increased from 7,601,885 lb. to 12,284,708 lb. The United States product in 1899 was valued at $465,146, against $461,450. The production of Venetian and vermillion red almost doubled from 3,270 tons ($64,570) in 1898 to 6,575 tons ($125,519) in 1899. As in former years ocher and oxide of iron were obtained mainly from New York, Pennsylvania, Tennessee, Vermont and Georgia. A small amount of iron oxide was produced in Nevada County, Cal., and it is stated that in Oregon new bodies of mineral suitable for the manufacture of paint have been discovered. These as well as those recently found in Texas have not as yet become producers.

IMPORTS OF OCHER, UMBER AND SIENNA INTO THE UNITED STATES.

Year.	Ocher of All Kinds.						Umber. (b)		Sienna.			
	Dry.		Ground in Oil.		Total.				Dry.		Ground in Oil.	
	Pounds.	Value.	Pounds	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds	Value.
1895	7,107,987	$56,020	41,153	$2,239	7,149,140	$58,259	1,560,786	$13,075	456,861	$11,021	6,576	$501
1896	8,954,252	68,196	27,023	1,561	8,981,275	69,757	689,075	8,360	(c)	(c)	(c)	(c)
1897	a 7,720,075	59,272	20,123	1,000	7,740,198	60,272	1,447,889	14,479	580,468	12,340	7,058	481
1898	5,898,720	46,571	31,460	1,546	5,930,185	48,117	1,123,079	9,051	544,713	11,451	4,008	230
1899	9,765,616	72,825	14,881	756	9,780,497	73,581	1,739,036	13,336	758,691	14,242	6,484	492

(a) Includes 241,452 lb. entered as crude and 1,416,587 lb. as powdered. (b) In 1895 includes 6,137 lb. ground in oil and 1,544,649 lb. dry; in 1896, 5,292 lb. ground in oil and 683,783 lb. dry; in 1897, 14,471 lb. ground in oil and 1,433,418 lb. dry, crude or powdered; in 1898, 4,608 lb. ($823) ground in oil and 1,118,471 lb. ($8,728) crude, powdered, washed or pulverized; in 1899, 4,849 lb. ($300) ground in oil and 1,734,187 lb. ($13,036) dry crude, powdered washed or pulverized. (c) None reported.

Increased production has doubtless been due to increased prosperity. Simultaneously with the increase of painting with lead paints, hematite and other iron oxide paints have been in demand for less pretentious structures, notably railway buildings, grain elevators and works where plumbiferous paints are likely to be attacked by fumes.

Our classification separates ocher, umber and sienna, but the Venetian red returned outside the table, includes Indian red. It is difficult to obtain an exact valuation from returns of these products, but as in the previous year we have adopted the nominal figure of $11 per short ton.

PETROLEUM.

THE year 1899 witnessed a marked improvement in the production of petroleum to which the Appalachian field made the largest contribution, although important gains were made in other fields, particularly in California and Texas. The output is still considerably below that of 1897, but with the increase in both price and production the past year is to be considered one of the most satisfactory and prosperous in the history of the American industry. It is safe to say that under the stimulus of present prices which is leading to the development of new fields and the drilling of new wells in the old, the production in 1900 will approximate the output of 1897.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES. (BARRELS OF 42 GAL.)

Year.	Appalachian Field. (a)	California. (b)	Colorado.	Indiana. (Lima Field.) (c)	Kansas.	Ohio. (Lima Field.)	Texas. (e)	Wyoming.	Other States.	Total.
1895.....	30,931,698	1,245,339	530,000	4,386,132	44,000	15,850,000	None.	3,500	1,800	52,992,464
1896.....	33,980,571	1,257,780	400,000	4,659,290	113,000	20,354,000	None.	2,878	3,600	60,771,119
1897.....	35,249,700	1,911,569	650,000	4,353,138	(c)90,000	18,507,000	65,000	15,000	(d) 9,000	60,849,407
1898.....	31,625,360	2,249,068	650,000	3,751,307	88,000	16,578,000	544,620	3,500	(d)10,000	55,499,875
1899.....	32,870,689	2,677,875	600,000	3,818,713	(c) 69,556	16,565,092	601,308	6,071	(d)25,000	57,234,304

(a) Includes New York, Pennsylvania, West Virginia, and part of Ohio. The statistics of production in this field and the Lima field in Ohio as given in previous volumes of THE MINERAL INDUSTRY were based on the pipe-line receipts reported by the *Oil City Derrick*. They have been changed in the present volume so as to represent the actual production. (b) Statistics of California State Mining Bureau. (c) Statistics of State Geological Survey. (d) Estimated. (e) The statistics for 1899 were furnished by Mr. C. F. Z. Caracristi.

The total net stocks of oil in the hands of the various pipe line companies reporting, in barrels of 42 gal., December 31st, were as follows:

Appalachian.				Ohio, Indiana.			
1896.	1897.	1898.	1899.	1896.	1897.	1898.	1899.
9,550,583	10,789,652	11,541,753	13,451,191	23,302,779	22,762,779	15,180,892	10,345,927

It is gratifying to note that the United States continues in the lead as a producer of light and illuminating oils, notwithstanding the fact that the Russian output of crude oil was greater in 1899 by 2,748,569 bbl. The yield in the United States for 1899 was capable of distilling over 73% of all light, illuminating and lubricating oils refined in the world. This condition is

due to the superior quality of American petroleum for purposes of distillation. The total amount of light oil produced in the United States in 1899 was approximately 28,534,556 bbl.

Competent authority during past years has placed the composition of American petroleum at: illuminating oil, 76%; light oils, 11%; lubricating oil, 3%; residuum, etc., 10%. Since this determination, however, there has been a considerable change in the quality of petroleum produced; the output of the Lima heavy oil increasing, while that of the light oils in the Appalachian field has diminished. A more correct estimate would be: Illuminating oil, 75%; naphtha and light oils, 10%; lubricating oils, 6%; fixed carbon, gas loss, etc., 9%.

The average cost of producing a barrel of petroleum in the United States is 65c., the several items being about as follows: Labor, including superintendency, 32c.; management and office expenses, 8c.; repairs to machinery, etc., 5c.; interest on investment, 10c.; depreciation, 10c. This includes loss from "dry holes" and "wild cat" ventures. On this basis, the actual cost of the output in 1899 at the wells was $37,493,561·95. There is of course considerable variance in cost in the different fields. The average monthly prices of crude oil are shown in the subjoined tables. The market in 1899 was much higher than for several years past, the average for January in the Appalachian field being $1·17 per bbl., against $0·65 in the previous year and for December $1·65⅛ against $1·17½. In the Lima field prices ranged 30 to 35c. higher than in 1898.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

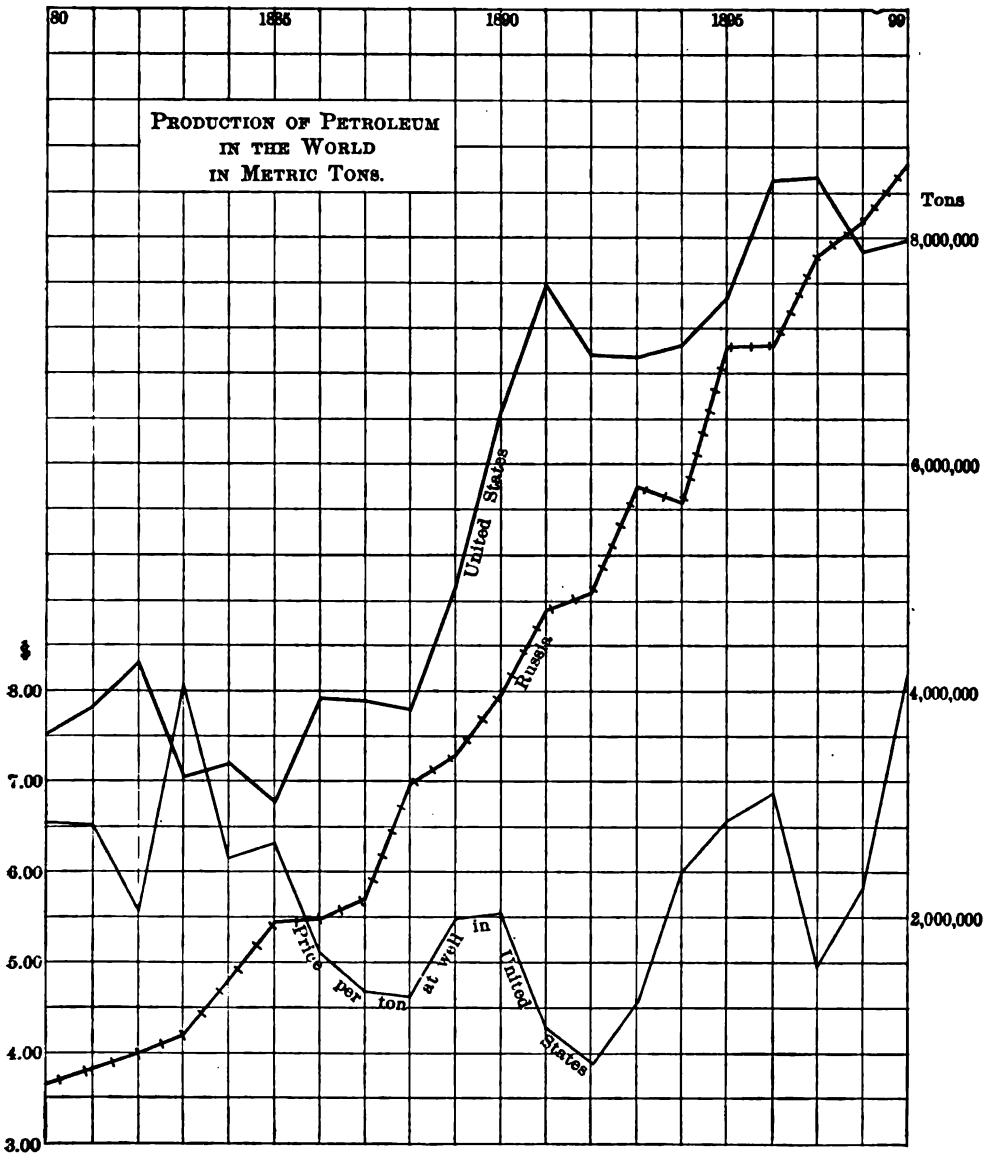
Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Yearly Aver'ge
1894...	$0·79½	$0·80½	$0·82	$0·84½	$0·86	$0·89½	$0·88½	$0·81	$0·83	$0·83	$0·83	$0·91½	$0·83½
1895...	0·99	1·04½	1·09½	1·79	1·74½	1·53½	1·46⅝	1·26½	1·22⅝	1·24½	1·48½	1·42	1·35½
1896...	1·45⅝	1·39	1·33½	1·22⅛	1·10⅛	1·14⅛	1·09½	1·05	1·12⅛	1·15½	1·15½	0·97½	1·19
1897...	0·88	0·90½	0·92⅛	0·85½	0·85½	0·86½	0·76½	0·71	0·69½	0·67⅛	0·65	0·65	0·78½
1898...	0·65	0·67⅛	0·78½	0·73⅛	0·82½	0·87½	0·93½	0·97½	1·01⅛	1·13½	1·16½	1·17½	0·91½
1899...	1·17	1·15	1·13	1·13	1·13	1·13½	1·22½	1·27½	1·44½	1·50⅛	1·57⅛	1·65½	1·29⅛

AVERAGE MONTHLY PRICES OF CRUDE OIL IN THE LIMA FIELDS. (a)

Month.	1896.			1897.			1898.			1899.		
	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana	North Lima.	South Lima.	Indiana
January...	$0·69	$0·79	$0·76½	$0·58	$0·58	$0·53	$0·46	$0·41	$0·41	$0·80	$0·75	$0·75
February.	0·85	0·75	0·75	.....	.....	.....	0·50	0·45	0·45	0·80	0·75	0·75
March.....	0·83½	0·73½	0·73½	0·57	0·52	0·52	0·58	0·53	0·53	0·79	0·74	0·74
April.....	0·73½	0·65½	0·65½	0·51½	0·47½	0·47½	0·54½	0·49½	0·49½	0·79	0·74	0·74
May.....	0·66½	0·61½	0·61½	0·51	0·46	0·46	0·59	0·54	0·54	0·81½	0·76½	0·76½
June.....	0·64	0·59	0·59	0·49	0·44	0·44	0·64½	0·59½	0·59½	0·85½	0·80½	0·80½
July.....	0·60½	9·55½	0·55½	0·48	0·43	0·43	0·67½	0·62½	0·62½	0·90½	0·85½	0·85½
August....	0·60	0·55	0·55	0·47	0·42	0·42	0·71½	0·66½	0·66½	0·93½	0·88½	0·88½
September	0·63	0·58	0·58	.....	.....	.....	.....	.....	.....	1·03½	0·98½	0·98½
October...	0·65	0·60	0·60	0·46	0·41	0·41	0·76	0·71	0·71	1·07½	1·02½	1·02½
November.	0·64½	0·59½	0·59½	.....	.....	.....	0·77	0·72	0·72	1·11½	1·06½	1·06½
December.	0·61	0·56	0·56	.....	.....	.....	0·79	0·74	0·74	1·16½	1·11½	1·11½
Average...	$0·69½	$0·63½	$0·62½	$0·51½	$0·46½	$0·46½	$0·63½	$0·58½	$0·58½	$0·92½	$0·87½	$0·87½

(a) Based on reports of the *Oil City Derrick*, except the figures for 1899, which are taken from *Stowell's Petroleum Reporter*.

Exports.—The exports of liquid petroleum products and crude petroleum in 1899 were $12,619,000 more than in 1898, and $6,035,000 more than in 1897. The quantity was less, however, than the amount exported in 1897. It will



be observed also that the value reported is low, averaging 5·93c. per gal. in 1897, 6·25c. per gal. in 1898, and 6·83c. per gal. in 1899, which is due to the fact that the exports consist in large proportion of crude petroleum and residuum.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (IN GALLONS.)

(1 = 1000 in quantities and values.) (a)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffin.		Residuum.		Totals.	
1895.....	116,108	$6,286	12,922	$1,000	686,006	$43,540	47,876	$6,289	170	$15	863,082	$57,131
1896.....	118,133	6,032	13,641	1,123	758,076	49,704	51,705	6,770	521	28	942,076	63,657
1897.....	121,864	5,044	13,704	1,020	804,446	46,876	52,659	6,782	12,947	835	1,004,920	60,007
1898.....	120,436	5,016	17,257	1,071	764,823	38,895	65,526	7,626	30,436	815	998,478	53,423
1899.....	117,690	5,958	18,210	1,597	733,393	49,173	71,105	8,656	21,609	658	962,007	66,042

(a) In addition to the above, the following quantities of paraffin and paraffin wax were exported: 1895, 114,231 lb. ($4,505); 1896, 112,517 lb. ($4,563); 1897, 136,069 lb. ($5,284); 1898, 166,317 lb. ($6,363); 1899, 181,861 lb. ($7,650).

It is worthy of remark that the amount received from petroleum and petroleum products exported in 1899 exceeded the sum paid for crude petroleum to the producers by more than $7,000,000. From this one may get a measure of the enormous profits made by the refiners, pipe lines and exporters.

Of the total exports, the United Kingdom and the British colonies took about $38,000,000 or over 53%. This is remarkable considering that British capital is largely interested in the Russian oil fields and that the British press (in many cases subsidized by Russian interests) is constantly decrying the so-called "low flash" American illuminating oil and as constantly maintaining the virtue of the Russian product. The results of various analyses made some months ago showed that at least 90% of the so-called American "low flash" oil was in reality Russian kerosene bearing an American label. The "low flash" question which has been brought so prominently before the English public is largely grounded on imagination. In most cases accidents occurring from American illuminating oils in the United Kingdom and Europe are due either to carelessness or to the use of unsuitable burners.

OUTPUT OF THE PRINCIPAL PETROLEUM-PRODUCING COUNTRIES. (a) (IN METRIC TONS.)

Year.	Austria.	Hungary.	Canada.	Germany	India.	Italy.	Japan.	Russia.	United States.
1894.....	111,930	2,061	116,000	17,232	44,506	2,853	23,837	5,161,552	6,926,361
1895.....	188,634	2,085	101,587	17,051	52,082	3,594	(b)22,463	7,056,330	7,413,914
1896.....	262,356	2,168	101,682	20,395	60,228	2,524	31,414	7,105,768	8,502,187
1897.....	275,204	2,299	99,310	23,308	76,834	1,982	(c)	7,831 254	8,513,140
1898.....	.....	2,471	98,044	25,789	71,627	2,015	(c)	(c)	7,764,713

(a) From the official reports of the respective countries. This table is only partially complete since it does not include the production of Sumatra, Borneo, Java, South Africa, Peru, and some other countries. (b) Estimated from hektoliters. (c) Not yet reported.

THE WORLD'S PRODUCTION OF PETROLEUM.

BY C. F. Z. CARACRISTI.

THE growth of the world's petroleum and allied industries have doubtlessly kept pace, more than any other natural resource, with the march of human progress, adding wealth and comfort to every section of the civilized globe. During the past few years search for petroleum has been carried on constantly in every region, and surveys and test drillings of new territory are being made wherever surface indications warrant the expenditure.

The subjoined table gives the production of illuminating light oils in 1899, in all countries except India, for which statistics are not yet available. It

also shows the relative percentage of each country's output, and the percentage of the average yield in distillate:

THE WORLD'S PRODUCTION OF ILLUMINATING OIL AND THE PERCENTAGE OF DISTILLATE PRODUCED.

Country.	Gallons.	Percentage Total Crude Petroleum Output.	Percentage Distillate.	Country.	Gallons.	Percentage Total Crude Petroleum Output.	Percentage Distillate.
Borneo.....	188,000	0·01	45	Roumania....	23,120,806	0·70	70
Canada.....	28,286,158	0·70	75	Russia (a).....	579,437,789	48·00	23
Galicia.....	46,176,880	1·60	55	Sumatra.....	42,501,711	1·27	60
Germany.....	2,217,327	0·10	28½	United States.	2,059,261,787	46·10	85
Italy.....	294,462	0·01	60				
Japan.....	5,974,260	0·22	55				
Java.....	12,893,328	0·49	50		2,810,511,427	100·00	
Peru.....	3,559,080	0·10	40				

(a) Col. R. B. Smolianinoff, of the Russian Engineers, supplied this information.

The world's refined petroleum supply has not been equal to the vast and growing demand made upon it, as is clearly shown by the prices at different ports. On December 29, 1899, the price of illuminating oil, f.o.b., was 3½c. per gal. at Batoum; 9·9c. per gal. at New York; and 20·0c. per gal. at Galician refineries.

The high price of Galician refined oil is due to an internal revenue tax of 8½c. per gal. Export and other charges should be added to the Russian oil.

AMERICAN VS. RUSSIAN CONDITIONS.—It is quite evident from the world's production that Russia is more than a successful competitor to the United States as to quantity of crude petroleum produced, but an analysis of the value of the component parts shows the fallacy of the statement that Russia will displace the United States in the world's oil markets. Russia being the greatest oil producer should, under normal conditions, be taken as the basis upon which to establish the unit of value for crude petroleum. In recent years the price of Russian petroleum has increased from 29 to 60c. per bbl.

Taking the present price as a standard of value we have:

Russian petroleum, 1 bbl. contains 9·66 gal. illuminating and other light distillates, and American petroleum, 1 bbl. contains 35·70 gal. illuminating and other light distillates. According to the value of the relative amounts of light distillates, this would make the American petroleum worth $2·21 per bbl. as compared to 60c. per bbl. for the Russian petroleum, so that the Russian product is higher than the American even at its apparent low price. The by-products are not considered in either case.

The cost of producing a barrel of petroleum in Russia is much less than in the United States owing to the large output of the wells. Reliable authorities place the price at from 14 to 18c. per bbl. for labor and repairs, and with interest on capital invested, management and incidental expenses the total cost of production should not be more than 23c. per bbl. (This figure is based upon the total known production and expenditures.)

While Russia has been assigned only 23% of the production of illuminating oil, benzine and lubricating oils, a member of the Refiners' Association states that "apart from the estimated percentage of valuable distillates produced from the total production of Russian petroleum, our refineries average 12 gal. of refined oils for every 42 gal. of crude oil treated. The residuum sold as fuel gen-

erally pays for the crude petroleum and we look upon the distillates, less their cost of refining, as net profit." Many producers ship their crude petroleum abroad to be refined, and others sell it for use as fuel in Russia.

While the percentage of distillates is placed somewhat low, the price value per bbl. is high, as most of the larger refineries own their individual production and make, in addition to their refining profits, the profits of producers or the difference between 23 and 60c. per bbl. = 37c. per bbl. The price of 60c. per bbl. is the amount paid for crude petroleum by refineries not producers, shippers to foreign refineries and for petroleum used as fuel.

Russian production 59,983,208 bbl., value	$35,989,924
Cost of production at 23c. per bbl	13,806,137
Net profit	$22,183,787

Over 68% of the Russian petroleum is produced by foreign corporations operating with English, French, Belgian and German capital. From a business standpoint the question that has disturbed foreign investors is the safety of their titles. A recent investigation of this matter showed that the Russian law forbids the tenure of petroleum lands by foreigners and Jews, and no law of trusteeship exists. This naturally makes the titles held by foreign corporations rather doubtful, although the Imperial Government, in order to induce foreign investment, has not enforced the letter of the law or want of law. But as Russia may at any time be precipitated into a foreign war and need resources upon which to realize ready cash, the whole of the foreign petroleum interests in that country might be confiscated, and while this has been denied by the fiscal agent in London, acting under the instruction of the Russian Minister of Finance, nevertheless the danger exists and is realized by the stockholders operating petroleum lands in that country. In London, stocks in Russian oil companies paying enormous dividends can be readily bought at a small advance above par.

North America.—The increase in price of petroleum and the general activity of the large producers during the past year has stimulated speculators to unnecessary energy in leasing supposed oil territory in "wild cat" regions, thereby retarding legitimate development by powerful companies who refuse to pay exorbitant bonuses and advanced royalties. In any oil field the odds are greatly against the man who invests his money in drilling for petroleum, unless he has ample means. This risk does not apply to rich corporations who set aside a certain percentage of their net earnings for new developments. The losses in "dry holes" are calculated to a nicety on the same principle that insurance companies estimate their bad risks. Add to this the experience and ability of their capable engineers and drilling masters we can readily see that their losses are reduced to a minimum.

Alaska.—Several samples of petroleum obtained during the past year from Cape Yacutat gave the analysis quoted below. From tests made the petroleum is probably of Tertiary origin: Benzine, etc., 4·50%; kerosene, 24·75%; lubricating oil, 6·25%; residuum, 64·50%; total, 100%.

The sulphur content was 0·91%. The residuum was of an asphaltic character, but no effort was made to determine the solid asphaltum contained. The sp. gr. of the crude oil was 38°B.

California.—(By A. E. Heighway.)—The production of 1898 while far in excess of all previous years has in turn been exceeded by that of 1899. Great activity has been manifested, and numerous wells were started in extensions of old districts or in entirely new districts, and many old wells were deepened. The area of petroleum bearing land is much greater than was at first supposed and the good demand and increased value for oil has stimulated prospecting in new territory with very satisfactory results. There is now a greater demand for crude oil for use as fuel than ever before existed. The high price of coal and the convenience and economy of oil (estimated to save from 30 to 50% on existing prices) has caused it to replace coal in many factories, railroads, private residences, etc. This increased demand has resulted in a production far beyond that of any past year, and has advanced prices to $1·05 to $1·15 per bbl.

In the Los Angeles district 1,150 wells have been drilled, 663 of these were producing in July, August and September. The depth of the wells average about 1,000 ft. During the past year, about 75 wells have been abandoned and 100 new ones have been drilled. The entire production of the Los Angeles district for 1898 was 1,168,000 bbl., and in 1899 the production was 1,125,000 bbl., showing a decrease in production. This loss has been more than made up in other districts. There has been only one new district discovered, which is at Bakersfield in Kern County.

The following districts are the most important:

District.	Number of Wells.	Abandoned Wells.	Daily Production in Bbls.	District.	Number of Wells.	Abandoned Wells.	Daily Production in Bbls.
Bakersfield	15	1	800	Newhall	100	40	400
Brea Cañon	2	.....	100	Puente	55	18	400
Coalinza	30	10	1,400	Summerland	364	59	400
Fullerton	50	2	500	Sunset	20	3	100
Los Angeles	1,150	487	2,652	Ventura	225	125	2,000
McKittrick	9	.....	250	Whittier	35	11	300

Colorado.—The Florence oil wells have not been able to supply the demand for petroleum distillate at the various reduction plants in the vicinity of Cripple Creek. Most of the plants are now installing coal burning furnaces in place of the petroleum burners.

Indian Territory.—Some activity was looked for in Creek County, owing to the purchase of this field by the Bird Syndicate, Ltd., of London, England, but so far only experimental work has been carried on. Prof. Boverton Redwood has reported favorably on the property and it is safe to infer that the field possesses real merit in lubricating and fuel oil.

Pennsylvania.—In the Pennsylvania field during 1899 were drilled 8,752 new wells, 6,842 being producers and 1,910 being either dry holes or giving off petroleum in quantities too small to warrant the expense of pumping. Some 140 of these wells gave gas in considerable quantity and should not be classed as dry holes, although they produced no oil.

Texas.—This State continues to show marked activity in development and production. The production in 1895 was only 52 bbl., and in 1899 it has increased to 601,308 bbl. Not only are the wells reliable producers but the quality of the petroleum is closer to that of the Pennsylvania field than any other pool so far tested. The proximity of the field to Central America, Mexico, South America, and the West Indies is destined to make Texas one of the most important oil producing States in the Union. The California field owing to its isolated position and the gravity of the petroleum and its distillates, while of great local importance, can in no way effect either the American or foreign markets; that is, unless a better grade of petroleum is encountered than has heretofore been produced. However, the discovery and development of this field has closed the American ports to Peruvian petroleum.

West Virginia.—Harrison and Doddridge counties added considerably to the production of this State during 1899. At Wolf Summit, Lynchburg, Salem, and Wallace important strikes have been made, and the territory is being actively and satisfactorily developed for both petroleum and gas. The geological formation of the region indicates that an extensive pool has been struck which extends into Taylor and Barbour counties to the south and east. The oil and gas in this region is found in the Dunkard, Gordon and Fifth sands. A test well is now being drilled some 10 miles west of Grafton, and it is reported that an abundance of gas was struck in the "Big Injun" sand, and that the lands in that region are being taken up by the Flemmington Coal and Coke Co., the South Pennsylvania Oil Co. and others.

Wyoming.—Considerable exploitation work has been done in the Powder River and Salt Creek regions as well as in the territory between Casper and New Castle. The oils that have been analyzed vary from high-grade oils to those producing from 40 to 50% kerosene, others only from 24 to 38%.

Canada.—The production of petroleum in 1899 was 808,570 bbl. valued at $1,202,020. These returns are calculated from the inspection returns at 100 gal. crude oil equalling 42 gal. refined oil, the value being computed at $1·48⅔ per bbl. of 35 imperial gallons (closely equivalent to 42 United States standard gallons). The barrel of refined oil is assumed to contain 42 imperial gallons. In the Gaspe oil district, Quebec, a pipe line has been installed and a refinery is being built at Gaspe Basin.

Cuba.—Petroleum was struck at a depth of 285 ft. in the province of Santa Clara as well as at Lagomatos in Matanzas. Extensive asphaltum deposits exist throughout Cuba, evidently the residuum from petroleum seeps. The deposit at Cana Tomasita is reported to be over 100 ft. thick and covers 12 acres.

Naphtha was found on the Molembo estate in the Province of Santa Clara at a depth of 312 ft. The naphtha seems to be a distillate forced through the underlying strata by interior heat, and has a sp. gr. of 0·752. Its purity is remarkable and the quantity is said to be abundant. A similar well was struck at Lagumillas, in the Province of Matanzas, at a depth of 82 ft. The wells were both flowing but gave a small production, and were finally abandoned. The supply, however, would have been larger had they been pumped.

Mexico.—Petroleum was discovered in the State of Vera Cruz many years ago, although for many years no attempt was made to develop the property. The Mexican Government granted a concession of some 10 square miles of petroleum fields to a few promoters, who sold their rights to an English syndicate. Considerable money had been invested in exploiting the territory near the town of Papantla and excellent results were obtained. Ten wells were drilled, each giving a production of 150 bbl. of petroleum per day. They were "gushers" and for some reason were allowed to flow. Had they been "shut in" much valuable product could have been saved. The petroleum is of asphaltic base, dark in color and of heavy specific gravity. Although the owners claim an ability to produce 48% illuminating and lubricating oils from the product it is doubtful if more than 31% can be realized, yet even with this assumption, it would be highly profitable in view of the fact that Mexico imported 678,667 gal. of crude and refined oil during 1899, and furthermore, the industry would be protected by import duties and other assistance given by the Government.

The petroleum occurs in the Tertiary formation, and it is said that where the petroleum from the wells overflowed the land a deposit of asphaltic residuum is now found. A report made by Robert Evans states that the field is very extensive and capable of a large production; and that a very small portion only is held by the English syndicate.

Newfoundland.—The Canadian Petroleum Trust, Ltd., and the London & Dublin Finance Corporation, Ltd., of London, have concluded arrangements to take over the newly developed petroleum fields of St. George's Bay and Bonne Bay, Newfoundland. The Newfoundland product is said to resemble that of the Gaspe field, but the wells so far drilled are much more productive. The oil has a sp. gr. of from 35 to 37°B.

Philippine Islands.—In 1897 Don Roberto Barranca de Arbelaez, a Spanish engineer who is interested in the Panay and Leyte petroleum lands furnished the following analysis of the Philippine product: Sp. gr. 34° B.

Kerosene	45·31	Sulphur	0·26
Light oils	8·69	Gases	2·24
Lubricating oils	32·50		
Fixed carbon (coke)	11·00	Total	100·00

If this petroleum exists in large quantities its high commercial value is unquestioned. Heavy oil also is reported to occur on Cebu, and other islands of the archipelago. The Philippine petroleums are identical to those of Japan both geologically and chemically.

South America.—*Argentina.*—Petroleum is reported in the provinces of Jujuy and Mendoza, but although its analysis proves its good quality, its inaccessibility makes it of little commercial importance for the present. A report by the Argentina Government chemist published by the *La Revista de Minas,* gives the following analysis of the Jujuy petroleum: Light oil sp. gr. 0·740, 5%; kerosene sp. gr. 0·814, 30%; lubricating oil, 52%; fixed carbon (coke), 11%; gases, 2%; total, 100%.

Of this, 0·25% was sulphur. The coke on being burned gave 0·50% ash. Paraffin was not determined.

Barbadoes.—Important developments have been made by the West Indies Petroleum Co., Ltd., of London, during 1899. The product is of a heavy asphaltic character and is found in the Eocene formation. The three leading wells have yielded an average of 12 bbl. of petroleum per day. The company contemplates extensive drilling, and the establishment of a pipe line and a refinery. The petroleum is said to contain 42% kerosene and 6% lubricating oil. The geographical location of this field, with the proper expenditure of capital will doubtless make the refined product a factor in the future West Indian markets. The property will be fully developed and orders have already been placed in the United States for oil well supplies, etc.

Colombia.—An extensive gas, petroleum and salt water seep exists on the eastern coast of the Gulf of Darien, between the Rivers Turbo and Guaralito, some 12 miles from the shore, and other seeps were found in the surrounding country; but the whole of the territory between the shores of the Gulf of Darien on the west, the Caribbean Sea on the north, and the River Mulato on the east has been so thoroughly effected by volcanic action that it would be a losing venture to drill, as the loss from dry holes would be too great to make the enterprise profitable. To the east of Mulato River petroleum and natural gas are found along the coast and rivers the entire distance to the Magdalena River.

On the Rivers San Juan and Vulcan several oozes occur giving off large quantities of heavy asphaltic oil. At the juncture of the Vulcan River and the Caribbean Sea the ebullition of salt water and petroleum has thrown up a mound 110 ft. in height, having in the center a crater which is generally filled with petroleum and salt water. Some years ago it gave off larger quantities of petroleum of good quality but now exudes only a small quantity of maltha or brea. The color of the brea is a dark green when it first issues from the ground, but rapidly turns a dark lignite brown. A seep of light oil is found in the hills, some six miles south of this point.

Petroleum is also found on the Caribbean coast at Playa del Viento on the peninsula formed by the delta of the Sinu River. This is a heavy, highly sulphurized petroleum. On the Sinu River at Sitio Neuvo the natives have dug a 12 ft. pit, which gives about a barrel of brea per day. It is used for caulking canoes and brings a good price. To the east of the Sinu River an excellent light paraffin oil is found near Cienega del Oro; and heavy oil on the territorial lands of the tribe of San Andres.

South of Cartagena, at Turbaco gas seeps are found that were observed by Baron von Humbolt, and 10 miles west of Barranquilla is a natural gas seep on Brincon plantation. A well 95 ft. deep gave 40 lb. pressure. In the Department of Tolima, between the Upper Magdalena and the City of Ibague a lake of asphaltum and petroleum is found. Petroleum and asphaltum are also found on the Opon River, a tributary of the Lower Magdalena, and petroleum is also found in abundance and of good quality on the Meta River, a tributary of the Orinoco.

An analysis of Cienaga del Oro petroleum gave the following: Naphtha, 2·92%; kerosene, 31·00%; lubricating oil, 36·00%; paraffin, 3·00%; gas,

11·50%; coke, water, etc., including 0·66, 16·54%; sp. gr., 33°B.; viscosity, 0·98; color from red to light brown; flash point, 101°F.

Another sample collected during the night when the sun did not affect the petroleum gave 52% kerosene. There are few new fields that offer the opportunities of Colombia; but until a change of policy is taken up by the Government, capital will not develop the territory.

Ecuador.—In many of the indentations formed by the river valleys of the coast from the Bay of Caraquez, on the north to the Tumbes River, on the south petroleum is found, and from the Gulf of Guayaquil it is found as far as the Peruvian frontier along the littoral. This territory is locally known as the Southern field. The Northern field includes the Island of Puña and the Canton of Santa Elena.

The most important indications of the existence of petroleum in Ecuador exist in the region from Santa Elena to the coast at Puntilla. The natives have dug several shallow pits in the alluvial deposits of sulphur, salt, clay and gypsum, and have collected the exudations of maltha (brea) and petroleum that seeps through the crevices of the underlying Tertiary shale, limestone and sandstone. The petroleum is brought to the surface by the water. In the foot hills west of Santa Elena several oozes are found and the petroleum seeps out associated with salt water. This petroleum, however, is very heavy and exudes from the bituminous shale which owing to the tilted condition of the formation reaches the surface. In 1874 an unsuccessful attempt was made by the Ecuadorian Government to drill a test well.

A number of salt water springs and gas seeps are found about 30 miles from Santa Elena on the road to Guayaquil. These springs are thermal during the dry season, the water having a temperature of 123°F. At times some petroleum is given up.

A selected sample of the petroleum from the Canton of Santa Elena gave the following analysis: Light oil, 3·35%; kerosene, 24·45%; lubricating oil, 11·35%; residuum, including 0·86 sulphur, 60·85%; total, 100%.

This petroleum came from the upper shale, and was exposed to the effect of tropical heat. This field if developed will doubtlessly be an important rival to the Peruvian product.

Peru.—Considerable activity continues in the Talarra, Zorrito and Grau fields of Peru, especially in the latter where the Compagnie Français is operating. If the suggestion to consolidate the three companies now operating in Peru is carried out, the production of that country will doubtless be greatly increased. This field, while not a competitor to the United States, may be an important factor in the local oil trade of the Pacific Coast of South and Central America.

Venezuela.—Asphaltum is found on the western slope of the Motilones Mountains that separate Colombia and Venezuela on "Oroyo Sucio"; and deposits of asphaltum and petroleum are found in the valley which is drained by the Catacumbo River. A large number of petroleum oozes occur on the lands of Don Silvestre Martinez on the eastern bank of the Catacumbo River, similar to those existing on the Vulcan River in Colombia. Petroleum is also found

for a distance of some 40 miles up the Catacumbo River. After seismic convulsions, the salt water and lighter oils flow quite freely for some months after which the emanations become less marked and the petroleum very heavy and in small proportions compared to the quantity of water emitted.

These terrestrial movements act upon the petroleum flowing fissures similar to the effect of nitroglycerine explosions in artificial wells, only to a greater extent. This note is worthy of record as it accounts for the varied flow in the same district at different times. Good indications of petroleum exist along many branches of the Zulia River, and in the region traversed by the Rivers Tara and Sardinarte. In fact they appear in the entire State of Tachira, in Venezuela and the N.E. section of the Department of Santander, Colombia, in the valley leading from the Lake of Maracaibo to the City of Cucuta.

Light petroleum is reported to exist in the region traversed by the upper Orinoco adjoining the Department of Cauca, Colombia, and the samples received from the Indians trading on the Meta River indicate that the product is of good commercial quality. Near Casigua, in the State of Falcon, petroleum is also found in more or less abundance.

From a commercial point of view the Maracaibo district seems to possess the best outlook of the oil fields of Venezuela.

Samples of petroleum obtained some 6 kilo. from Las Palmas gave the following analysis: Sp. gr., 34·5°B., light oils, 8·35%; kerosene, 41·50%; lubricating oil, 18·30%.

The formation in which the heavy oils are found is the Eocene in the fourth, eighth and tenth series of stratification. The strata consist of bituminous shale, limestone and lastly very porous sandstone, filled with minute remains of Tertiary aquatic life. These have been named from the "Uraba sands," their pertusion and well defined condition in the mountains of Cerro del Aguila at the entrance of the Gulf of Darien or Uraba.

The light petroleum is found in the Jurassic sandstone (secondary) in the third series of stratification called the "Santa Marta sand" from the fact that the formation, although dry at this point, is best examined in the Sierra Nevada de Santa Marta, Colombia. Here it retains the paraffin, however, which is the best indication to determine where that class of petroleum is found. The Uraba sands cross the Gulf of Darien and follow the western Andes through Colombia along the Pacific Coast into Ecuador and Peru, where the formation has been developed.

EUROPE-ASIA.—*Algeria.*—Dr. Dunikowski, at the University of Limburg, having examined the Tliouanet region, states that this part of the Algeria atlas indicates a relatively abundant supply of oil; the development of which will in the future be one of the most important industries of that colony.

Austria-Hungary.—The following report has been made on the petroleum discovered at Mahrne, in the district of Czaesa. The oil is of a light brown color and contains no benzine, but yields nearly 70% of burning oil, with a boiling point of from 150 to 300°C. It contains very little paraffin.

The importation of petroleum products into Austria-Hungary during 1898, with the corresponding figures for 1897, was as follows: Russian crude oil,

319,077 metric centners (405,038); crude mineral oil, light 30,756 (36,759); Roumanian crude oil, 195,087 (188,574), refined mineral oil heavy, dark and light, 83,164 (79,558); lubricating oil, 99,344 (80,470); paraffin, impure, 16,937 (26,582); refined, 53,729 (44,230) ceresin, 41 (48). The exports for the same period in centners were: Lubricating oil, 30,617 (130,824); crude mineral oil, 10,761 (16,004); refined petroleum, 30,617 (130,824); benzine, 226,840 (210,535); unrefined paraffin, 81 (37); refined, 130 (235); ceresin, 14,224 (13,305).

Burma.—The principal wells are located at Yenangyaung and Yenangyat, the former locality being the more important. The cheapness and excellence of the imported American and Russian oils—which in spite of import duty are still in most demand in the East—have a limiting effect on the production of the Burma oil. If the latter could be refined to a quality equal to that of the imported oils, the demand would be greatly increased. The statistics of production in 1898 are not yet available. The output in 1897 was 80,904,710 gal.

Dutch East Indies.—A refinery has been built on the British North Borneo territory by the Bombay-Burmah Trading Co. Boring has been successfully carried on in Sarawak. At Moaara Djawa on one of the delta branches of the Mahakam River eight flowing wells have been struck. Refineries are also being built at Balik-Papan to treat oil brought by pipe line from Sanga-Sanga, and at Koetei, one claimed by Sir Marcus Samuel, to be the largest in the world, is about completed. It will have a capacity of 3,000 tons a day, and will be controlled by the Shell Co.

The petroleum industry of Sumatra is passing through a critical period. The original good yield has been curtailed due to the lowering of the level at which the oil could be tapped. A further handicap to progress is the recent determination of the Holland Government to grant no concessions to foreigners, thus barring out the Standard Oil Co., which had planned to take over the works of the Holland Monwara Enim Co. in South Sumatra. A large refinery is now being built under Russian direction in Palembang, about 150 km. from the oil fields, and is to have modern equipment. This concern has received a grant of 1,122 sq. km. of land. It has a capital of 16,870,000 marks.

Egypt.—The Egyptian Government, through the intervention of Lord Cromer, has granted L. Frederichi, of London, a concession to operate the petroleum field lately tested on the southwestern shores of the Red Sea. Two flowing wells have been drilled and petroleum was struck at a depth of 780 ft. A company is now being organized in London to drill additional wells and construct a pipe line. The product is similar to the Russian. The bad climate will probably cause the failure of the enterprise.

Germany.—The production of Germany in 1899 was 25,789 metric tons, an increase of 2,486 tons over the production in 1898. The output is derived mainly from Alsace and near Hanover, Prussia.

India.—Boverton Redwood gives the following description of the petroleum lands now being worked by the Assam Railways and Trading Co., Ltd. The fields consist of 4 sq. miles at Digboi and 30 sq. miles at Makum. The development of these fields was commenced by the company in 1888, with the result

that at Digboi, the district is now a highly productive developed oil field, yielding large quantities of crude oil of excellent quality. Recently renewed attention has been paid to the Makum field, which yields a valuable natural lubricating oil, and borings have been very satisfactory.

The structure of the part of the southern flank of the great Brahmapootra anticlinal on which the oil fields lie is somewhat complicated by faults parallel to, or but slightly divergent from, the strike, and by the occurrence of two or more subordinate anticlinals with intervening synclinals, and of many slight deviations from the general S.S.E. dip. Further westward the overlying Miocene sandstones conceal the coal series, except along the strip known as the Jeypore coal field, probably an anticlinal, whose northwest side is buried under alluvium.

Several trial borings at Beloochistan have been put down by the Indian Government, which have proved the presence of mineral oil in an extended area. As there is no coal in the district, the Government has adopted petroleum for working locomotives on the Quettah Railway with some success.

Italy.—The production of petroleum in 1899 was small, amounting to 2,015 metric tons, a slight increase over the production in 1898. Prospecting is still being done but with no great success. During the past year a company has been organized by Prince Colonna and Count Cicardi to develop the Italian oil fields.

Japan.—While Japan will never be a serious competitor in foreign markets, it nevertheless may supplant a considerable importation of refined petroleum at the present time by the refined product of its own wells. To this end development work is proceeding energetically in Nagaoka and at Nagamiena, which is looked upon as the most promising oil producing center. Nitroglycerine shooting has never been tried in the Japanese oil field. The main objection lies in the character of the rock, which is considered too soft and loose for successful shooting. A law has lately been passed in Japan which allows foreigners to work the oil fields, thus insuring speedy development.

Persia.—At Daliki, about 50 miles from Bushire, oil is collected on the surface of a hot sulphur spring. An attempt was made to sink several bore holes, but the task had to be abandoned when driven to a depth of 631 ft. Oil was struck at 187 ft. occurring as large drops in the water, while at 467 to 470 ft. deposits of bitumen with very thick oil were struck, and a similar layer at 559 to 561 ft. This deeper oil could not be pumped, as it formed a tarry mass. Oil is also found in the Minab district, and on the Island of Kishim in the Persian Gulf.

Roumania.—The most important company is that of the Seana with a capital of 10,000,000 lei. It has a refinery at Campana which refines its own oil as well as that of other producers. The production of many of its wells is falling off, but it has an extensive undeveloped territory. A pipe line is proposed to Constanza, which will effect great economy in transportation. The Government has shown considerable unfriendliness toward foreigners, and there has been much conflict over titles.

Russia.—(For the general discussion of the Russian industry see the article

contrasting it with the industry of the United States which appears in the first portion of this chapter.) The main object of the Russian industry is the production of fuel oil in residuum to the partial neglect of the illuminating grades, a condition naturally discouraging to refining. Three years ago there were 120 refineries at Baku; in 1897, however, only 63 operated, and of these 20 produced over 90% of the total output. In September, 1898, only 8 refineries were working. The total production of refined oil in 1898 was but little in excess of 1897, notwithstanding that the production of crude was nearly 8,000,000 bbl. larger. The price of crude oil showed a further advance in 1898 over the high price at the end of 1897. At the close of the Volga navigation season, in 1898, the price of crude was 55c. per bbl., against 42c. per bbl. at the opening of the year. This advance in the price of crude undoubtedly explains the great increase and energy in drilling new wells, which was a feature of 1898. Nearly 25% more wells were producing in that year than in 1897. The increased number of wells, however, did not result in a proportionate gain in production. It has been pointed out that unless richer oil-bearing strata are found in the Russian oil district, or other productive territory be found, more drilling will be required in the future if the Baku output is to be materially increased.

The higher price of residuum or fuel oil is apparently due to the increasing demand for such fuel in Russia's manufacturing industry. This is specially interesting to the American oil producer, as the more the Baku refiner gets for his residuum the cheaper he can sell his refined product. The American consul at Moscow is quoted as saying that, owing to the price of petroleum being relatively so low, coal was only used for domestic heating purposes and by blacksmiths, the manufacturers using petroleum fuel almost exclusively. The value of petroleum fuel for steam raising has been firmly established. According to one report, 1 pood (36 lb.) of oil fuel is equivalent to 2 poods of coal for steam raising; taking into consideration the saving of expense in burning oil and the relative prices of the two fuels. It has been stated that with crude oil as high as 85c. per bbl. coal cannot successfully compete with it in the Moscow district.

Important oil fields on the east shore of the Caspian Sea are now being thoroughly explored in connection with the Siberian Railway. The product differs from the Baku oil in that it contains paraffin, and hence may be suitable for fuel only. Distillation tests of oil from Naphtha Mountain give the following results: Benzine, 5%; kerosene, 18%; solar oil, 8%; residue, 69%. The residue yielded 36% vaseline, and contains about 10% bitumen.

Petroleum also occurs in another part of Russia in the Sagis-Emba Delta in the Ural Steppes. Asphalt and bitumen are of frequent occurrence at the mud Lake Possor about 100 km. east of Gerjew; a Belgium company is developing the resources. The oil here has a sp. gr. 0·916, and its quality coupled with a considerable outflow of gas points the presence of a large deposit.

The Baku Russian Petroleum Co., Ltd., took over its property from the Russian owners on the 13th of July, 1898. The report for the 15 months ended on the 30th of September last showed a gross production of 26,310,029

poods, from which must be deducted royalties amounting to 3,660,269 poods, making a net production of 22,649,760 poods. The production came from 54 wells and showed a falling off from the average of the preceding three years amounting to 3,200,000 poods. This falling off is attributed to engineering difficulty; 2,851,046 poods were used as fuel. The average price obtained per pood was 11·19 kopecs and for astakki, 11·87 kopecs per pood. The amount of profit for the period to the 12th of October, 1899, after setting aside £42,000 for depreciation and £8,500 provision for taxes leaves £101,010. Up to the 12th of October, 1899, £31,084 dividends had been paid.

Turkey.—Oil springs exist near Broussa, in old Servia, and also in Armenia (outside the Russian Baku field); but it is said that no attempt has been made to sink wells or otherwise develop these deposits.

SHALE OIL.

*France.**—The French Government, in order to encourage the development of the production of oil from shale, decided on June 22, 1899, to offer a special premium to the refiners for the next six years, on the following conditions: That in the course of the first year the total sum expended on premiums shall not exceed 300,000 fr.; this sum is to be divided *pro rata* between the refiners, according to their production. At the same time, these premiums shall not represent more than 3 fr. per hectoliter of unrefined oil. During the third and fourth years the total sum paid in premiums shall not exceed 200,000 fr., and not more than 2 fr. per hectoliter. In the fifth and sixth year the total amount is not to exceed 100,000 fr., or more than 1 fr. per hectoliter.

New Zealand.—J. Carroll has published a notice† setting forth the terms of the bonus offered by the Government for the production of mineral oil from shale. The conditions are:

1. A bonus of 1s. per gal. (£5,000) will be paid on the first 100,000 gal. of mineral oil produced from shale obtained in the Colony of New Zealand, the oil to be of a quality approved of by Government, and to be sold at a price not exceeding 1s. a gal. at any port of shipment in the colony.

2. Notice of intention to claim the aforesaid bonus must be given in writing to the Colonial Secretary not later than December 31, 1899.

3. The claim must be made before June 30, 1900.

4. The first claimant who proves to the satisfaction of the Government that he has fulfilled all the conditions to be the recipient of the bonus.

5. The other conditions—as to quantity, priority, quality, and value—to be fulfilled to the satisfaction of an officer appointed for the purpose by the Government.

United Kingdom.—The Scotch shale oil industry has been very prosperous during 1899. As a rule, the companies that were on a paying basis increased their dividends, while those that had suspended the payment of dividends were enabled to resume them. The subsidy offered by the French Government to encourage the development of the shale oil industry in France will probably have some effect upon the Scotch industry.

* *Petroleum Industry and Technical Review*, Aug. 26, 1899, p. 460.
† *Journal of the Society of Chemical Industry*, Sept. 30, 1899.

PHOSPHATE ROCK AND PHOSPHORUS.

THE phosphate industry in the United States has been prosperous in 1899; production has been enlarged by the opening of new deposits, consumption increased by superphosphate manufacturers, and prices have risen materially. The statistics of production in the different States and shipments are given in the following tables:

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (a) (IN TONS OF 2,240 LB.)

Year.	South Carolina.	Florida.	North Carolina.	Tennessee.	Other States.	Totals.	Value.	
							Total.	Per Ton.
1895.....	515,734	530,356	7,500	45,078	.....	1,098,668	$3,296,004	$3·00
1896.....	392,068	498,400	7,418	49,047	432	947,373	2,812,116	3·00
1897.....	333,626	543,490	7,000	121,351	2,000	1,007,367	3,022,101	3·00
1898.....	434,273	546,881	2,200	272,191	2,100	1,257,645	4,355,025	3·46
1899.....	686,153	706,677	15,000	462,561	3,000	1,823,391	7,031,785	3·86

(a) The figures for 1899 are based on railway and export shipments, except those for Tennessee, which were furnished by the Commissioner of Labor and Inspector of Mines. In 1898, 1897 and 1896 the statistics were compiled partly from shipments and partly from direct reports of the producers. Those for 1895 represent shipments, chiefly from points of export.

SHIPMENTS OF PHOSPHATE ROCK. (IN TONS OF 2,240 LB.)

State.	1898.			1899.		
	Foreign.	Domestic.	Total.	Foreign.	Domestic.	Total.
South Carolina.....	64,175	364,881	429,056	94,931	432,187	527,108
Florida.....	430,998	123,225	554,218	583,133	88,321	671,453
Tennessee.....	70,000	202,191	272,191	163,114	277,447	440,561
North Carolina.....	.....	2,200	2,200	.....	15,000	15,000
Pennsylvania.....	.....	2,100	2,100	.....	3,000	3,000

Prices.—The price of high grade Florida rock advanced from $8·50 to $11 per long ton, f. o. b. Fernandina, owing to the good export demand. Florida land pebble rose from $5·50 to $7·50 per ton, delivered in New York, while Peace River sold up to $4·50, f. o. b. Punta Gorda. Tennessee high grade rock rose from $2·30 to $4·50 per ton, f. o. b. Mount Pleasant, and low grade material advanced from $1·75 to $2·75, f. o. b. at the mines. South Carolina kiln-dried land rock sold at $4·50@$6, f. o. b. Fetteressa. Tennessee rock for export is sold on a basis of 78% bone phosphate of lime, containing 3 to 4% iron oxide and 2% alumina. For domestic consumption the 78% rock carries from 4 to 5% iron oxide and 3% alumina; 75% rock from 5 to 5·5% iron oxide and 3% alumina, and 72% rock from 5·5 to 6·5% iron oxide and 3% alumina. River rock was variously quoted at $3@3·25 per ton f. o. b. Ashley River, and $4·25@$5 f. o. b. Fetteressa.

Acid phosphate sold in the South in 1899 at $6·50@$6·75 per ton f. o. b. cars or vessels at factory of sellers, for shipments in bulk containing 13·5 to 14% available phosphoric acid; bagged and tagged goods 13% brought $7·75@$8, f. o. b. cars or vessels. Several combinations in the fertilizer industry were formed in 1899, the principle ones being the American Agricultural Chemical Co. in the North and the Virginia-Carolina Chemical Co. in the South. These companies have secured large interests in phosphate mining lands, and have made large shipments to the fertilizer factories.

The European superphosphate manufacturers have been large buyers of American phosphates in 1899, notably of Florida and Tennessee high grade rock. In fact, these phosphates are rapidly replacing the Algerian and the Continental apatites. The United Kingdom manufacturers also have been heavy purchasers. Florida high grade rock, 75 to 80%, sold from $13·13@$15·80 per long ton c. i. f. United Kingdom or North Sea ports, while the 78 to 80% Tennessee rock brought $11·31@12·48, for the same delivery; 68 to 73% Florida land pebble sold at $10·15@$11·25, and 58 to 63% Peace River at $8·40@$9 per ton, c. i. f. United Kingdom or North Sea ports. South Carolina rock sold around $6 per ton, but the demand was small. Algerian 58 to 63% rock sold at the close of the year around $10·05 per ton, and 63 to 70% rock at $8·40, same delivery. The selling prices of phosphates is so much per unit of phosphate of lime contents, but in order to compare these prices with those that ruled in the United States we have converted them into similar quotations.

The following table gives the exports of phosphates from the United States in the last two years, in long tons of 2,240 lb.:

UNITED STATES PHOSPHATE EXPORTS IN 1899.

Month.	Florida High Grade Rock.	Fla. Pebble, Peace River; Tenn.; So. Car. Rock.	Total.	
			Quantity.	Value.
January	27,643	26,573	54,216	$356,773
February	32,690	18,798	51,423	408,970
March	43,051	30,424	73,475	595,123
April	59,001	32,187	91,188	738,396
May	47,078	51,557	98,635	782,545
June	23,051	34,046	57,097	446,181
July	48,747	52,452	101,199	785,389
August	41,155	43,934	85,089	665,665
September	35,728	33,392	69,120	535,348
October	36,694	39,759	76,453	616,255
November	28,947	26,445	55,392	407,719
December	18,527	36,659	55,186	431,788
Total, 1899	442,252	426,171	868,423	$6,770,102
Total, 1898	358,569	222,379	580,948	4,672,463

By far the greater part of these exports has been on account of German superphosphate manufacturers.

Florida.—Messrs. Auchincloss Brothers in their annual circular state that during 1899 no high grade deposits of importance have been discovered, nor have improvements in the processes of mining or handling rock been introduced. The ever increasing difficulties of obtaining rock from lower levels, the higher prices of mining machinery and materials, and the advance in the cost of labor have increased the cost of production to an extent that may be estimated at

33% greater than that of 1898. The improved prices have influenced production, and previously abandoned mines have been reopened. Notwithstanding this, miners have been hard pushed to meet their contracts, and on December 31, 1899, fully 90% of the stock above ground, ready for market, had been sold for future delivery. An examination of all the high grade mines at present operating in this State indicates that the present rate of production has reached its maximum. Concerning the shipments of high grade rock to Europe, it is alleged that the increase of 23% over 1898 is due to the fulfilment of delayed 1898 contracts, postponed by the excessive freight rates ruling from April to August of that year, during the Spanish-American war. It is also noted, however, that during the last five months of 1899 the range of ocean freight rates has been equally excessive, so it may be inferred that considerable shipments for the fall of 1899 have likewise been postponed for delivery in 1900.

The shipments of high grade phosphate rock from Florida in 1896-99, according to Messrs. Auchincloss Brothers, are given in the following table:

YEARLY SHIPMENTS OF HIGH-GRADE FLORIDA PHOSPHATE ROCK.

Countries.	1896.	1897.	1898.	1899.	Countries.	1896.	1897.	1898.	1899.
	Tons.	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.	Tons.
Austria	2,494	4,505	4,946	(c)	Italy.....	32,999	16,931	11,040	4,546
Belgium....	27,214	22,964	38,908	37,103	Norw'y & Sweden	12,534	7,442	9,378	11,988
Denmark ..	9,594	11,019	8,287	5,475	Russia	1,607	3,613	.....	1,700
England....	20,533	24,163	23,849	31,789	Scotland.....	1,038	5,957	6,000	9,545
France.....	6,986	13,931	.....	3,165	All other (*b*).....	8,663	2,415	3,642	8,360
Germany ..	151,461	181,355	186,731	243,887					
Holland (*a*).	47,235	53,039	64,309	87,167	Total, long tons.	322,871	350,277	360,505	444,675
Ireland.....	518	2,953	3,420	(c)					

(*a*) A large proportion of the shipments to Rotterdam is forwarded to the interior of Germany. (*b*) Included in these shipments are those made to the United States, Australasia, Japan, and the West Indies. (*c*) Not reported.

Of the total shipments in 1899 the United Kingdom ports received 41,334 tons; Baltic, 118,229 tons; Continental, 269,041 tons; Mediterranean, 7,711 tons; Australasia and Japan, 5,937 tons, and United States ports, 2,423 tons, making a grand total of 444,675 tons, as against 306,505 tons in 1898.

North Carolina.—The shipments in the fiscal year ending September 1st, 1899 amounted to 13,750 tons, against 12,500 tons in 1897-98.

Pennsylvania.—The production of marketable phosphate rock was limited, and was consumed locally.

South Carolina.—The industry in this State was materially better than for several years past, and the production of phosphate rock showed an increase of about 200,000 tons over 1898. The total amount of river rock mined in 1899 was given by the State Phosphate Inspector at 121,073 tons, against 99,315 tons in the previous year. The stock on hand on December 31st amounted to 19,766 tons, which compares with 33,015 tons at the same date in 1898. The total shipments in 1899 aggregated 134,094 tons, showing an increase of 40,086 tons over the previous year. Of these shipments 94,921 tons went abroad (64,174 tons in 1898) and 39,173 tons were consumed at home (29,923 tons in 1898). The royalties due the State for 1899 amounted to $34,929, as against $23,523 in 1898. According to the Inspector the price of dried rock advanced to $3·38 per ton f. o. b. at mines.

Tennessee.—The demand for phosphate rock has increased and prices are

higher. Phosphate land has consequently risen greatly in value. The high grade material has been derived chiefly from Mount Pleasant and vicinity, while the lower grade deposits exist in Hickman, Giles and Sumner counties. For the former rock there is a good export market, but the low grade rock is mostly consumed at home. The foreign shipments in 1899 were made chiefly through Pensacola, and amounted to 134,441 tons. The domestic shipments from this port aggregated 11,212 tons. The exports from Norfolk in 1899 were 27,359 tons, of which 15,054 tons went to England, and the balance to Germany, Holland and Belgium. The shipments from Hickman County amounted to 42,202 tons, being the largest on record, and from the Gallatin district, where only one concern was producing in 1899, 550 tons were shipped.

The Mount Pleasant district has been very active. The Tennessee Phosphate Co. is credited with having invested $220,000 in phosphate land, not including $100,000 in the blue rock field of Hickman and Lewis counties. This company owns 887 acres in fee simple, and has a complete washing, drying, crushing and grinding plant at Attilla, one mile from Mount Pleasant, and a dry shed at Mount Pleasant.

The Central Phosphate Co., incorporated in South Carolina, has a large investment at Mount Pleasant, besides owning property at Beaufort, S. C. It has now a small plant and is erecting a new and complete one. The Blue Grass Co. operates under a lease of 1,000 acres, and has a drying and crushing plant at Mount Pleasant. The Columbia Phosphate Co. and the Petrified Bone Mining Co. also operate in this district. The International Phosphate Co. has about $125,000 invested and owns the Solita and the Arrow mines, together comprising 808 acres. This company has a spur track and complete washing, drying, crushing and grinding plant at Arrow. Several other companies have invested heavily in this district. In April, 1900, an agreement was entered into by the Bluegrass, American, Columbian, Howard, International, Jackson, Wilson, and Carpenter Phosphate companies, and the Petrified Bone Mining Co., to regulate the phosphate industry in Tennessee. These concerns formed the Tennessee Phosphate Association, of which G. W. Killebrew, the State Geologist, is president.

The present cost of producing and preparing a ton of phosphate rock from the Mount Pleasant field in Tennessee for market is summarized by a correspondent in the *American Fertilizer* for March, 1900, as follows: Royalty, $1; stripping, 46c.; mining, 50c.; hauling, 40c.; burning, 20c.; loading, 19c.; total, $2·75. This is without allowance for extra cost of work in rainy or cold weather, for rehandling of rock necessitated by failure of kilns to burn properly or for the expense of storing, which is frequently necessary through the lack of cars.

THE WORLD'S PRODUCTION OF PHOSPHATE ROCK. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Algeria.		Belgium. Cubic Meters.		Canada.		France.		Norway.	
1894.....	64,260	$220,888	371,776	$879,300	6,224	$ 41,166	539,000	$3,811,142	2,086	$26,460
1895.....	157,886	634,823	506,730	683,140	1,653	9,565	526,784	3,160,600	1,601	23,760
1896.....	165,738	500,905	297,470	537,320	517	3,420	582,667	3,502,027	1,106	17,280
1897.....	222,141	912,564	350,056	436,762	824	3,964	535,390	2,852,887	*c* 872	12,960
1898.....	269,500	1,078,000	156,920	303,230	665	3,665	568,558	3,115,958	(*e*)	(*e*)

Year.	Redonda. Br. W. Indies. (d)		Russia. (b)		Spain. (b)		United Kingdom.		United States.	
1894.....	579	$5,525	14,072	$31,680	40	$80	711	$3,385	1,098,147	$3,296,602
1895.....	5,778	24,880	6,327	18,682	1,040	2,081	2,540	21,875	1,116,247	3,296,004
1896.....	(e)	(e)	3,776	11,065	770	3,080	3,048	26,250	952,370	2,812,116
1897.....	(e)	(e)	5,917	22,132	2,084	16,672	2,032	17,500	1,023,485	3,022,101
1898.....	(e)	(e)	(e)	(e)	4,500	46,008	1,575	13,565	1,277,717	4,355,052

(a) From the official reports of the respective countries and *Annual General Reports on the Mineral Industry of the United Kingdom*, by C. Le Neve Foster. (b) Phosphorites exported. (c) Apatite exported. (d) Phosphate of alumina exported. (e) Statistics not yet published.

Algeria.—The phosphate industry is now well under control of a few companies, these including the Compagnie des Phosphates du Dyr, which has acquired the properties of Crookston Brothers. The properties and plant of the Constantine Phosphate Co. were sold to the French syndicate that controls La Société Française des Phosphates de Toqueville and La Société des Phosphates de Tébessá, the reported purchase price being $750,000. With the inauguration of the Sfax railway, the Tébessá phosphate mines are now in direct communication with the coast, and are thereby materially benefited. It has been proposed to convert the phosphates into superphosphates by sulphuric acid, to be manufactured at the numerous and recently discovered deposits of iron pyrites. The total shipments of phosphates from Bonê, Algeria, in 1899, amounted to over 246,406 tons, as against 223,348 metric tons in 1898. These phosphates carry from 58 to 70% phosphate of lime, and are sent chiefly to England, France, Germany and Italy, where they command a price nearly equal to the Florida land pebble. The exports in metric tons of phosphate rock from Bonê in 1898 and 1899 are summarized from the *Phosphate Industry* as follows:

Month.	1898.	1899.	Month.	1898.	1899.	Month.	1898.	1899.
	Tons.	Tons.		Tons.	Tons.		Tons.	Tons.
January.....	18,060	24,931	May	20,314	23,958	September.....	15,277	13,154
February	18,010	19,586	June.....	20,935	17,250	October.....	12,590	10,292
March.....	18,895	22,798	July.....	20,844	23,330	November	11,887	27,240
April	18,482	20,565	August.....	19,105	20,679	December.....	27,990	22,482
						Total	222,348	246,406

Belgium.—The phosphate rock mined in this country averages about 50% phosphate of lime, and occurs in the Department of Ariége in the form of black rounded masses, in a bed from 2 to 11 m. thick. Some of the rock has been shipped to the United States, where it was used as a filler for fertilizers.

Brazil.—Phosphate deposits are said to occur on the Fernando de Noronha Islands containing 86·53% lime phosphate, 5·61% calcium carbonate, 0·03% calcium sulphate, 0·22% calcium fluoride, 0·06% calcium chloride and 2% iron and aluminum sesquioxide. The product is white in color, and is tender and almost friable.*

Canada.—The production of apatite is very small, amounting in 1899 to 3,000 tons, valued at $18,000.

Tunis.—The railway from Sfax to the mines, about 250 km. is completed. The Société des Phosphates de Gafsa has increased its shipments of high grade rock. It is reported that new companies are to be formed to exploit other districts in this country.

* *The American Fertilizer*, April, 1900.

West Indies.—A company is operating phosphate deposits at Santa Barbara on the Island of Curaçoa, and is shipping principally to Great Britain. This company has received its grant from the Dutch Colonial Government, which extends over 1901. In the fiscal year ending September 30, 1899, the company reports that no profit accrued from the industry during this time. It paid to the Colonial Treasury 270 fl. 22c., while the export duties amounted to 4,046 fl. 70c. According to a British Consular report, a Holland syndicate contemplates entering into some arrangement with this company for extending exploration work on the phosphate deposits.

The Electrolytic Manufacture of Phosphorus.

By William Borchers.

Billaudot, of the Compagnie Electrique du Phosphore, Paris, published a full description of an electric phosphorus plant. As originally described the apparatus and the method of working is as follows: The substances to be treated are introduced into the electric furnace *F* at its upper portion (see Figs. 1, 2 and 3); under the influence of the electric arc the reaction takes place and the phosphorus vapors mixed with carbon monoxide are liberated. A portion of the various solid substances in this mixture is deposited at the upper part of the furnace, and a portion is deposited in the exhaust tube *a*. To prevent the obstruction of the latter the weighted scraper *b'* is used from time to time. This is lowered and raised by means of an external hand wheel attached to the prolongation of the axis of the drum *b*.

The phosphorus vapors and the gases pass through the open door *c* into the pipe *D*, and then become distributed in the vertical pipes *T*, reuniting in the pipe *E* and then proceeding to the open door *g*. During this passage, the phosphorus vapor is condensed by cooling and drops of greater or smaller size, mixed with the solid matter present, are deposited upon the walls of the condenser and remain there in a liquid state, as these walls are maintained at a temperature exceeding 50°C. This more or less liquid mixture, under the influence of gravity, flows toward the door *h;* and as its consistency is sometimes thick the flow may be assisted from time to time by means of blasts of carbon monoxide introduced through the nozzles *t* of the pipes.

The gases which have been deprived to a great extent of the phosphorus vapors and of the solid matter consist almost exclusively of carbon monoxide with infinitely small particles of phosphorus in suspension. They pass through the open door *g* into the towers *G*, where they come in contact with a solution of copper salt spread over a very large surface, which retains almost completely the last traces of phosphorus, forming copper phosphide.

Upon leaving the towers *G* the gases, consisting almost entirely of carbon monoxide, are either conducted toward the exhaust chimney *C* or, by means of the suction device *A*, are conveyed to various heating appliances and burned, or they may be compressed by means of the compressor *B* in a vessel *R* in communication with the pipes *k* to produce the blasts mentioned in connection with the condensation apparatus.

As stated above the condensed phosphorus mixed with the foreign substances

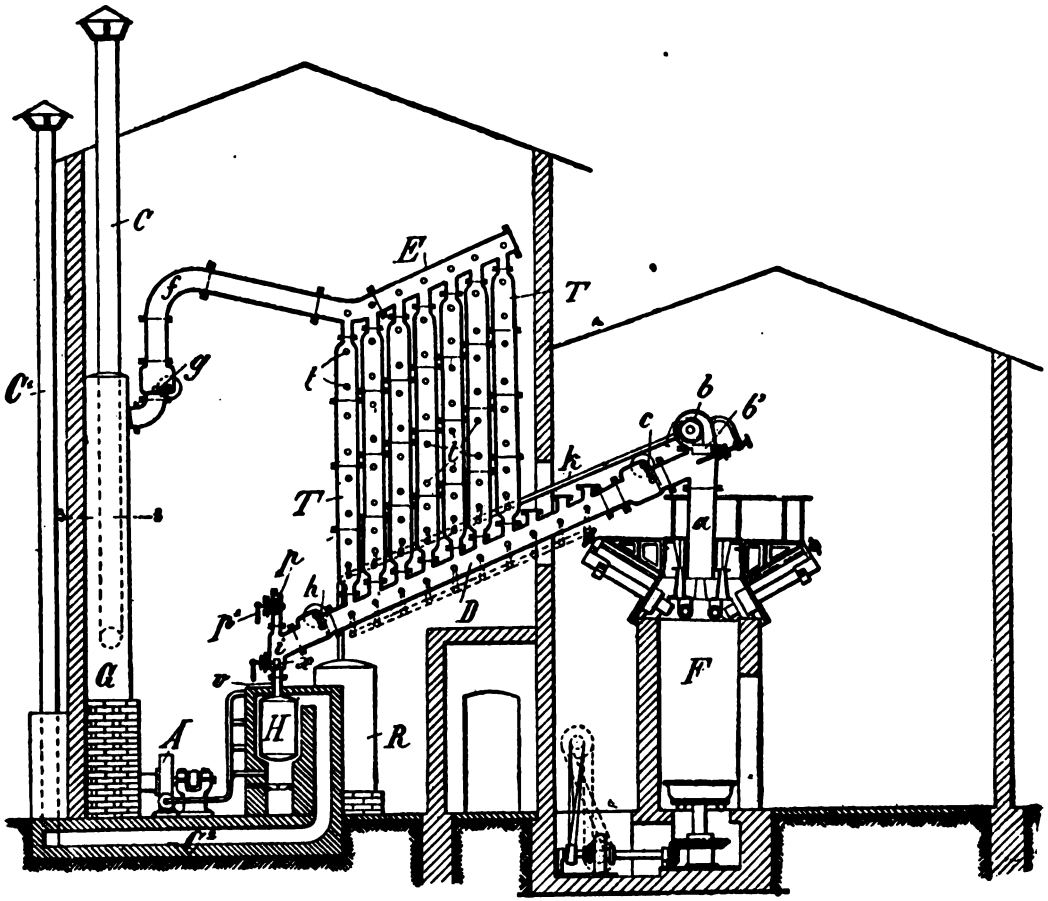


FIG. 1.—Cross Section.

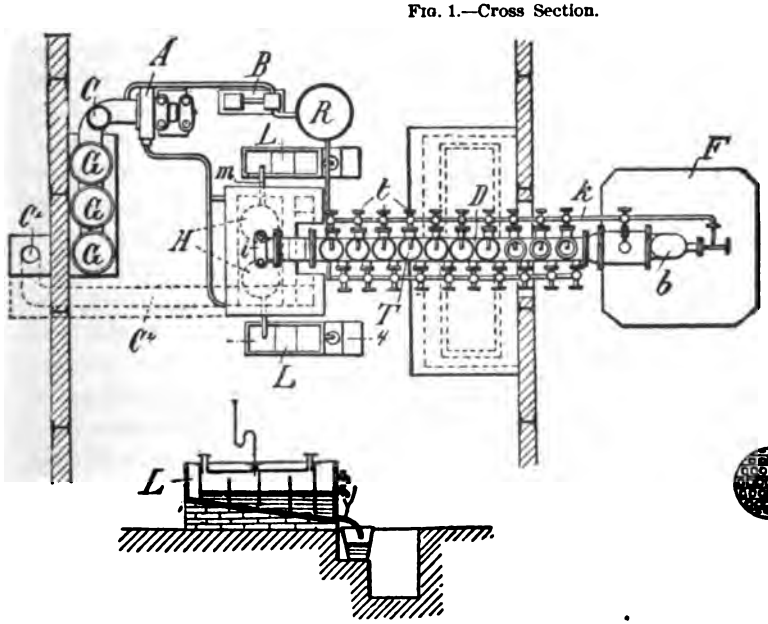


FIG. 2.—Plan.

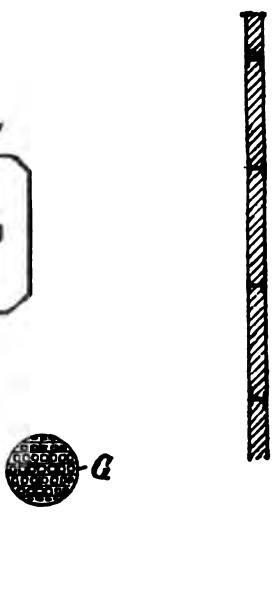


FIG. 3.—Details of Condensation Tower.

THE BILLAUDOT ELECTRIC PLANT FOR PHOSPHORUS.

flows along the pipe *D* and reaches the door *h;* this being open, allows the phosphorus to pass through and enter the distillation box *i*. From there the mixture is directed into one of the two distillation vessels *H* by opening the corresponding door *x*. In order to facilitate the filling of the vessel *H*, a pointed-cylindrical weight is displaced from time to time by means of a device *p* arranged in the box *i* and similar in action to the device *b* of the pipe *a* at the outlet from the furnace *F*, above described. The displacement of the weight is effected by operating the handle *p'*.

When the first vessel *H* is judged to be filled sufficiently, the corresponding door *x* is closed and the condensed mixture is directed into the second vessel *H*. While this second vessel is filling, the contents of the first are distilling. To effect this, the carbon monoxide burners beneath this vessel are ignited; the combustion gases circulate around the vessel and raise the temperature sufficiently to effect the distillation of the phosphorus. The gases of combustion finally escape through the flues C^2 of the chimney C^1.

The issue of the phosphorus vapors from the vessel is aided by means of a slight current of carbon monoxide, or of carbon dioxide.

The distilled phosphorus is collected in the trough *L*, beneath water, the temperature of which is maintained above the fusion point of phosphorus, and is removed and passed through a filter if necessary, for the purpose of separating the small quantity of red phosphorus which may have been formed; it is then ready for the moulding and packing room. The solid matter which remains in the vessel, after distillation, is withdrawn through the upper manhole *H*.

The operation is continued by distilling the product contained in the second vessel *H*, proceeding in the same manner; the phosphorus coming from the condenser is then directed to the other vessel. The operation is interrupted only during the time necessary for clearing the electric furnace. When this is done the doors *c g h* should remain closed.

The installation as described above may be variously modified to suit special conditions. The source of heat necessary for the distillation may be derived from gas- or coal-fired furnaces, the condenser may consist of various groups of vessels or of a single vessel either simple or composite (tubular). The temperature of the condenser may be maintained above the fusion point of phosphorus by utilizing the heat of the gases which pass into the condenser, and, if this is not sufficient, the loss of heat by radiation may be prevented by surrounding the walls with insulating material or by employing an external source of heat. The special device for preventing obstruction and for facilitating the flow of the products within the condenser may be replaced by other appliances as scrapers, screws, etc. The condensation towers should afford the best contact possible between the gases and the absorbent liquid, and for this purpose they may be filled with hollow bricks, coke, etc. The towers may be replaced by boxes having horizontal stages or baffles, or by scrubbers. The absorbent may be a salt of copper, tin, or iron, either in a solid form or in solution, or it may be a caustic soda or potash solution. The collecting apparatus may be of different forms and the product collected therefrom may be distilled directly from these vessels or removed and treated elsewhere.

PLATINUM AND IRIDIUM.

ALTHOUGH there have been rumors that platinum has been found in matrix, in Arizona, recent investigations by Prof. Wm. P. Blake have shown that these were unreliable. The reported discoveries near Leavenworth, Wash., have not been confirmed nor have those at Entiat Mountain in the Mad River district of the same State.

Platinum is refined in the United States by the Mint; the Welsbach Incandescent Light Co., Gloucester, N. J.; Baker & Co., Newark, N. J.; C. F. Croselmire, Newark, N. J.; the S. S. White Dental Manufacturing Co., and the Orford Copper Co.*

The Mint handles platinum incidentally with gold, but the platinum and palladium produced by the Orford Copper Co. comes from the matte produced at Sudbury, Ont. Here the mineral, sperrylite, an arsenide of platinum has been known to exist for a number of years, but it has only become of recent knowledge that palladium is found in the ore, although in what mineralogical form it has not yet been announced. According to J. A. Edman, platinum and iridosmium are found in Trinity County, chiefly near Spanish Creek, in thin scales, rounded granules, or irregular thicker grains up to 0·25 in. in diameter. The largest piece found weighed 28 grains; microscopic particles, however, predominate. Grains of pure iridium are rare; iridosmium, ruthenium and palladium are found as alloys. The quantity of platinum and allied metals found in hydraulic mining may be estimated roughly at from 0·1 to 0·2% of the weight of the gold obtained. It is improbable that the platinum metals will ever be a source of revenue here except in isolated cases, and in no case without taking the gold into consideration.

The metal opened the year at $15 to $15·50 an ounce, rose as high in New York as $18 in small lots. Manufacturers of chemical appliances state that there has been a large demand for platinum ware.

Algeria.—It is stated that crusts and crystallized nodules of certain clay beds contain iridium, 64·85%; platinum, 9·64%, and palladium, 0·89%.

New South Wales.—The yield of platinum in 1898 was 1,250 oz. valued at £2,062, a falling off from previous years. It all came from Fifield, where it is found with gold in alluvial ground. The production for 1899 is estimated at 625 oz.

* For production in the United States see introduction.

New Zealand.—Platinum was found in several localities in New Zealand, in beach sand and auriferous gravel. According to Mr. George Lee, manager of the Round Hill Gold Mining Co., the faces of the claims are unusually sandy, covered with clay and heavy timber, carrying small leads of inferior and low-grade wash-dirt in places; on the lower portion of the field the main rich wash-dirt being on the false sandy bottoms, and the hard, fine clay bands constituting the false bottom in places. On the higher levels the rich wash-dirt is on a main bottom of igneous formation, which generally has been composed of various rocks, feldspathic diorites predominating on the upper field, and apparently merging into decomposed granites, etc. The wash on both bottoms generally is formed of portions of these rocks, mixed with black sands carrying gold and small quantities of platinum, and possibly other precious and rare metals.

Northwest Territory.—A company was formed early in 1900 to work the platiniferous sands of the Hootalinqua River in the Northwest Territory which were mentioned in THE MINERAL INDUSTRY, Vol. VII. It is proposed to build a steam dredger and to excavate and wash sand for gold. The platinum is found in the magnetite concentrates which accumulate in the sluice boxes. This material can again be concentrated by removal of the non-auriferous or platiniferous magnetic iron oxide, by means of a magnetic separator.

Russia.—Notwithstanding the complaints and uneasiness in Russia, owing to the platinum deposits passing into the control of foreigners, a combination was formed in the early part of the year to affect a consolidation of the properties of the two largest owners of platinum mines in the Urals, Count P. Schuwaloff, and the heirs of Prince Demidoff. The capital supplied by Brussels, St. Petersburg and Paris capitalists is said to amount to 20,000,000 fr. It is proposed to erect reduction works in the vicinity of the mines.

Six years after platinum was found in the Ural, the Goroblagodad area was opened up, but it was not until after 1879 that the Iss district attained the first rank in production. The entire region is one of great denudation and as there has been no removal or transportation of residual material by glaciation, the rocks are everywhere capped with layers of subangular fragments, nearly or quite in place in turn covered by loam and vegetation locally called turf. Peridotites, olivine, gabbros and the serpentinous product of these rocks compose the area of the western portion of the field. Platinum is found principally as an original constituent of the peridotite in particular, and the gravels resulting from the decomposition of this rock have concentrated on the limestone formation, which offered a corrugated surface forming a natural riffle for saving the platinum and gold. It is on this contact that the richest gravel is found, though the limestone itself contains no platinum whatever. The thickness of the bed does not average more than 4 ft. and the average grade is about 40c. per cu. yd. It can be worked only for five months in the year, from May 1st to October 1st, although stripping of the turf is possible in April and October. The gravel from the beds which have been stripped by open cutting is hauled in carts to the washing machine, which consists of an upright cylindrical tank about 7 ft. in depth and 8 ft. in diameter, with a

floor formed by a circular punched iron plate with 0·75 in. holes. An agitator driven by a central vertical shaft disintegrates the clayey mass. Water is fed into the machine, which has a capacity of about 500 cu. yd. in 24 hours. The gravel, bearing the values passes through the holes in the bottom directly to sluices, which are covered with coarse matting. It is stated that the cost of hauling and washing the gravel is about 15c. per cu. yd. Where the gravel is not accompanied with large quantities of clay a horizontal revolving screen is used from which very good and economical results have been obtained. These methods are of course available when the ground has a minimum amount of water, but where the gravels are wet, a system of spoon dredging is employed, the gravel raised by the spoon being washed in a hand rocker. The river bed is 300 ft. wide and the water is 7 ft. deep. Many old channels exist and the entire width of its flood plain valley varies from 1,500 to 2,000 ft. Tests of the gravel from all parts of the valley show platinum. It would seem to afford an excellent opportunity for carrying out dredging operations on a large scale.

Occurrence of Platinum in Meteorites.—An analysis by John N. Davidson showed the presence of platinum and iridium in meteoric iron, found in Coahuila and in Toluca, Mexico. From 608·6 g. of Coahuila meteoric iron, 0·014 g. of platinum and 0·0015 g. of a black powder considered to be iridium were obtained. In a specimen of meteoric iron examined by J. C. H. Mingaye, analyst to the Mines Department of New South Wales, platinum was found. The occurrence seemed novel but a repetition of the analysis confirmed the result. The specimen contained nickel and cobalt as well.

F. Mylius and R. Dietz investigated commercial platinum, palladium, iridium, rhodium and ruthenium produced by Heraeus at Hanau.* Platinum and iridium were found to be relatively pure, but the results of the rhodium analyses were less satisfactory, the sample containing but 98·4% pure metal, with 1·5% iridium, 0·1% ruthenium and a trace of platinum. This metal (sp. gr. 12·6) had been prepared from the well-defined pentamine-trichloride $Rh(NH_3)_5Cl_3$. This salt was examined by the authors, and purified by treating with hydrochloric acid containing free chlorine, and then dissolving in warm, concentrated, sulphuric acid. A second, purified, specimen of rhodium, supplied by the same firm, contained 0·4% of iridium, and a third, with the exception of a trace of platinum, was found to be quite pure. The tenacity with which the iridium clings to the rhodium is due to the fact that it also forms a pentamine-trichloride which is isomorphous with the rhodium compound. These two bodies are separated by repeated crystallization from dilute ammonia. The pure rhodium compound is soluble in 182 parts of water at 18°C., and is nearly colorless; the impure salt is more soluble, and has a dark yellow color.

Osmium and ruthenium are easily purified by taking advantage of the fact that their oxides are volatile. Commercial osmium tetroxide is nearly pure; on reduction it gives the pure metal. On account of its instability, per-ruthenic acid is not on the market; instead of this, the dioxide, RuO_2, was examined. After igniting with potash and niter and dissolving in water, the brown solution of potassium ruthenate was saturated with chlorine, and then distilled.

* *Journal of the Society of Chemical Industry*, Feb. 28, 1899.

The tetroxide obtained was pure, melting at 25°C. Ruthenium can be melted with the oxyhydrogen flame; osmium, which has hitherto been regarded as infusible, melts in the electric furnace. Finally, directions are given for the detection of the members of this group of metals, in presence of one another, and of other metals.

Refining Platinum.—A method of refining gold scrap containing platinum which is not suitable for the ordinary method of refining gold is described.* Sifted filings are digested with nitric acid of sp. gr. 1·199 as long as any silver dissolves, a trifling amount of platinum dissolving also, as silver-platino-nitrate. The remaining metal is washed and digested with aqua-regia until the silver chloride forming on the surface of the metal prevents further action. The gold solution is then decanted and the silver chloride is dissolved in ammonia, then the metal is ready for further treatment by aqua-regia. After six such treatments the residue consisted of pure platinum. If the metals are really alloyed it is advisable to fuse the alloy with three parts of zinc. On dissolving out the zinc by means of sulphuric acid, the metals are left in a suitable condition for separation.

Volumetric Determination of Platinum.—A method for the volumetric determination of platinum and gold by H. Peterson,† is based on the fact that potassium iodide effects a partial reduction of the metals in many metallic solutions. In this reaction free iodine is separated and is titrated with sodium thiosulphate. It is important that a sufficient amount of potassium iodide be used. With platinum the reactions are as follows:

$$PtCl_4+4KI=PtI_2+I_2+4KCl.$$
$$I_2+2Na_2S_2O_3=2NaI+Na_2S_4O_6.$$

With gold the reactions are rendered more complicated by the reaction of iodide of gold with sodium thiosulphate.

$$AuCl_3+3KI=AuI+2I+3KCl.$$
$$AuI+Na_2S_2O_3=NaI+NaAuS_2O_3.$$
$$I_2+2Na_2S_2O_3=2NaI+Na_2S_4O_6.$$

* *Oesterreiches Zeitschrift für Berg- und Hüttenwesen*, XLVII., 356.
† *Zeitschrift für Angewandte Chemie*, Jan. 17, 1899.

POTASH AND POTASSIUM.

THE world's supply of potash salts, with the exception of saltpeter, continues to be derived from Germany, where the production is still controlled by the Kali-Syndicate, which regulates the output of each mine and fixes the prices for all markets. With December 1, 1898, the agreement between the mines, under which the syndicate operated for 10 years previous, came to an end, and a new contract, extending for three years, was entered into to govern the potash industry. According to the new contract each manufacturer is limited as to the amount of potash salts which he can put on the market, while under the old contract the limitations were placed on the amounts of crude salts mined. The different products are now grouped in four classes according to their percentage of potassium, as follows: (1) Products with more than 48% K, an equivalent of 76·1% KCl or 88·9% K_2SO_4. (2) Products with not more than 48% K, nor less than 18%, equivalents of 76·1 to 28·5% KCl or 88·9 to 33·3% K_2SO_4. (3) Crude salts (not carnallite) with 12·4 to 18% K, equivalents of 19·7 to 28·5% KCl or 23·0% to 33·3% K_2SO_4. (4) Carnallite salts with less than 12·4% K. The percentage of potash is made the basis for determining the amount of salts of each kind which any manufacturer may offer for sale, but no restrictions are put on the output of the crude material so long as it does not come into the market. Class 1 includes mostly refined products; class 2, both refined and crude; classes 3 and 4, only crude. A method for separating classes 3 and 4 was adopted, and will prevent carnallite salts which contain a high percentage of potash being sold under class 3. It is based on the relative solubilities of carnallite and kainite in 96% alcohol, the magnesium chloride in the kainite being insoluble, while it is soluble when occurring in carnallite. Salts which contain more than 6% soluble chlorine are put in class 4.

Imports.—The imports and exports of potassium products of the United States will be found in the tables at the end of this volume.

The statistics of the Stassfurt salt industry are shown in the following tables, which we owe to the courtesy of the Kali-Syndicate:

OUTPUT OF CRUDE SALTS OF ALL KINDS. (IN METRIC TONS.)

Year.	Rock Salt.	Carnallite.	Kieserite.	Sylvinite.	Hartsalz, Schoenite and Kainite	Boracite.	Total.
1857	12,798	.....	.....	.....	.....	.....	12,798
1858	25,568	.....	.....	.....	.....	.....	25,568
1859	15,387	.....	.....	.....	.....	.....	15,387
1860	31,863	.....	.....	.....	.....	.....	31,863
1861	40,315	2,293	.....	.....	.....	.....	42,608
1862	47,045	19,727	20	.....	.....	.....	66,792
1863	42,403	58,303	68	.....	.....	.....	100,774
1864	46,511	115,408	89	.....	.....	5	162,013
1865	45,027	87,671	75	.....	1,314	8	134,095
1866	49,128	135,554	413	.....	5,808	14	190,918
1867	56,154	141,604	1,143	.....	8,976	10	207,888
1868	71,946	167,337	1,418	.....	10,772	18	251,490
1869	65,201	211,384	226	.....	.....	27	294,196
1870	52,018	268,226	71	.....	20,301	16	340,632
1871	50,154	335,945	47	.....	36,582	16	422,744
1872	55,335	468,537	22	.....	18,067	26	541,988
1873	64,342	441,079	7	.....	6,101	25	511,554
1874	71,072	414,961	16	.....	9,753	18	495,815
1875	77,705	498,737	5	.....	24,124	12	600,583
1876	76,656	563,669	145	.....	17,988	24	658,483
1877	80,526	771,819	151	.....	35,477	44	888,018
1878	101,694	735,750	520	.....	34,004	98	872,066
1879	107,472	610,427	761	.....	50,206	105	768,971
1880	118,171	528,212	893	.....	139,491	107	786,874
1881	149,258	744,726	2,082	.....	158,330	117	1,054,513
1882	141,338	1,059,300	4,658	.....	148,477	126	1,353,899
1883	152,747	950,205	11,790	.....	228,817	205	1,343,762
1884	180,818	739,959	12,389	.....	217,107	159	1,150,432
1885	212,082	644,710	11,970	.....	272,369	142	1,141,273
1886	238,544	698,229	13,917	.....	247,327	149	1,198,167
1887	201,962	840,207	14,186	.....	237,629	150	1,294,134
1888	191,595	849,602	10,754	2,220	375,574	169	1,429,915
1889	259,286	798,721	9,354	28,329	362,611	139	1,458,441
1890	302,205	898,526	6,951	31,917	401,871	165	1,581,635
1891	365,910	818,862	5,816	32,661	512,494	180	1,735,923
1892	292,964	736,751	5,782	32,669	585,775	166	1,654,127
1893	264,411	794,660	4,807	49,140	689,994	187	1,803,199
1894	281,247	851,338	3,865	63,495	729,301	169	1,929,415
1895	259,424	782,944	3,012	76,097	669,592	145	1,791,164
1896	277,884	856,223	2,841	90,389	833,025	195	2,060,557
1897	288,035	851,272	2,619	84,105	1,012,186	184	2,238,400
1898	291,591	990,998	2,444	94,270	1,120,616	252	2,500,171
1899	310,377	1,317,947	2,066	100,658	1,063,195	155	2,794,395

UTILIZATION OF THE CRUDE POTASH SALTS. (IN METRIC TONS.)

Year.	Carnallite and Rock Kieserite.				Kainite and Sylvinite (a) (including Hartsalz and Schoenite).			
	For Agricultural Purposes.		For Manufacturing Concentrated Salts	Total.	For Agricultural Purposes.		For Manufacturing Concentrated Salts	Total.
	Germany.	Elsewhere.			Germany.	Elsewhere.		
1880	4,137		524,968	529,105	23,769	103,749	11,973	139,491
1881	6,902		739,905	746,808	20,372	119,491	18,466	158,330
1882	10,249		1,053,709	1,063,958	30,413	95,263	22,800	148,477
1883	17,434		944,560	961,994	48,138	153,200	27,479	228,817
1884	18,654		733,694	752,348	48,643	109,656	58,807	217,107
1885	18,988		637,691	656,679	50,870	143,518	77,981	272,369
1886	22,729		689,418	712,147	65,835	105,050	76,441	247,327
1887	30,892		823,500	854,393	84,493	89,203	63,842	237,629
1888	31,776		828,580	860,356	105,237	142,171	130,386	377,794
1889	37,746	381	769,947	808,076	150,342	113,109	127,489	390,940
1890	34,574	373	810,530	845,477	178,031	126,984	128,772	433,787
1891	38,893	551	785,234	824,678	240,001	173,508	131,646	545,155
1892	45,367	1,253	695,913	742,533	366,661	131,912	119,871	618,444
1893	59,464	3,483	732,233	795,181	428,891	184,358	125,852	739,134
1894	60,893	4,117	790,193	855,203	466,208	200,240	126,348	792,796
1895	50,528	3,886	731,582	785,946	436,922	190,732	117,975	745,629
1896	56,541	3,964	798,559	859,064	557,527	245,060	120,828	923,415
1897	58,544	5,157	790,190	853,891	668,340	295,765	132,185	1,096,290
1898	60,793	7,189	925,461	993,443	722,115	334,111	158,660	1,214,886
1899	58,677	4,611	1,256,730	1,320,017	717,637	314,869	131,342	1,163,848

(a) Quantities of sylvinite containing more than 13% potash.

PRODUCTION OF CONCENTRATED SALTS. (IN METRIC TONS.)

Year.	Potassium Chloride. 80%	Potassium Sulphate. 90%	Sulphate. Potassium—Magnesium		Kieserite in Blocks.	Potash Manure Salt. (a)	Kieserite; Ground and Calcined.
			Crystallis'd 40%	Calcined. 48%			
1884	106,330	3,000	400	8,000	17,800	9,500	.....
1885	104,500	4,000	450	9,000	18,500	8,400	.....
1886	110,200	3,639	472	10,111	19,500	8,161	.....
1887	130,000	10,528	500	6,285	24,018	8,163	.....
1888	132,000	10,916	522	11,380	28,325	13,918	.....
1889	131,593	7,321	671	9,215	31,824	17,285	.....
1890	134,760	13,339	907	10,880	32,005	17,620	.....
1891	143,487	18,981	1,053	11,400	21,559	16,045	.....
1892	121,028	15,466	706	11,842	23,855	16,895	11
1893	132,528	16,361	739	12,643	24,386	17,344	105
1894	147,986	15,242	1,780	12,718	26,440	19,727	216
1895	145,027	13,403	898	8,249	25,115	19,724	142
1896	155,805	13,889	1,051	4,622	24,987	19,253	211
1897	158,963	15,403	922	7,415	25,669	23,042	214
1898	174,379	17,781	914	10,585	19,394	24,284	728
1899	180,672	24,656	579	8,459	28,216	70,916	260

(a) Quantities containing 38% potash.

Prices.—The prices established by the Kali-Syndicate for the year beginning March 1, 1900, for New York, Boston and Philadelphia, per 100 lb., are as follows: March 1, 1900—for New York, Boston and Philadelphia per 100 lb. Muriate of potash, 80 to 85%, basis 80%, $1·80; muriate of potash, minimum 95%, basis 80%, $1·83; sulphate of potash, 90%, basis 90%, $2·025; 96%, basis 90%, $2·055; double manure salt, 48 to 53%, basis 48%, $1·04; manure salt minimum 20% potash, 62c. For Norfolk, Va., prices, add 2c. per hundred to the foregoing quotations except for double manure salt, to which add 1c. For Charleston, Savannah, Wilmington, N. C., and New Orleans, add 1·5c. to New York prices. For bulk salts, Kainit, testing 12·4% potash, is quoted at $8·80 per ton of 2,240 lb., invoice weight at shipping port, or $9·05 actual weight at receiving port; sylvinite, 38c. per unit of sulphate of potash, invoice weight at shipping port, or 39c. per unit, actual weight at receiving point. Kainit prices for Norfolk are 30c. per ton higher than New York, and at Charleston and other Southern ports 20c. higher, while sylvinite is 1c. per unit higher. These prices are for not less than 500 tons of bulk salts, or 50 tons of concentrated salts, and are based upon shipment by river from the mines to seaport. For shipments by rail 40 pfennigs per 100 kg., or 5c. per 100 lb. extra is required.

Saltpeter.—Potassium nitrate is produced exclusively in India, and is refined at Calcutta for export. Owing to a decline in the use of saltpeter in the manufacture of gunpowder and for preserving food, the production has been restricted in recent years.

The output of saltpeter in India in the five years, 1894-1898, is reported officially as below, in long tons:

District.	1894.	1895.	1896.	1897.	1898.
Bengal	11,852	10,928	14,848	11,692	9,344
Madras	535	307	821	474	1,729
Panjab	90	180	216	451	544
Totals	12,477	11,415	15,885	12,617	11,517

The value of the production in 1898 is placed at 161,025 rupees, which is equal to $32,044, showing a decrease as compared with 1897 of 15,228 rupees, equal to $5,497.

The United States showed an increased consumption in 1899, and the imports in that year were 18,961,270 lb., as against 11,461,823 lb. in 1898 and 16,276,352 lb. in 1896. A small export business was done in 1899, amounting to 111,417 lb., against nothing in 1898. Prices fluctuated but averaged $3·82 per 100 lb. for crude, and $4·68 for the refined saltpeter in New York.

Potassium Chlorate.—Production in the United States is increasing, and imports in 1899 show a material falling off as compared with 1898. In 1899 the combination of foreign makers resulted in maintaining market values on a stable basis. In January potassium chlorate sold in New York at $8·875 to $9·50 per 100 lb. At the end of April the French product sold at $9 to $9·125, and the English at $9·50 to $9·75, while orders for the domestic article were booked at $9. In July limited offerings by importers caused prices to advance to $9·50 to $10·50, while domestic potassium chlorate sold at $9·25 to $10·25. In October the demand improved, but the expected dissolution of the foreign combination broke prices. Large orders were then booked for 1900 at $8 in New York, while the open market was quoted at $9 to $9·75 for the foreign article, and $8 to $9·50 for the domestic. Late in the year domestic contracts were taken for 1900 delivery at $8 to $8·25. In December quotations were $9 to $9·25 for foreign, and $8·25 to $9·375 for domestic potassium chlorate.

Manufacture of Alkaline Cyanides.—The United Alkali Co. at its Old-Quay Works at Runcorn, England, has adopted a system, starting with ammonia, as the nitrogenous material, and combining the nitrogen with carbon forming sulphocyanide as the intermediate product. The final reaction in making alkaline cyanides may be represented by the following equation:

$$RCNS+3O=RCN+SO_3.$$

This shows the thiocyanate converted to cyanide by the oxidation of its sulphur to sulphuric anhydride.

At these works sodium sulphocyanide is the salt employed, and the reactions taking place may be represented as

$$(1)\ NaCNS+2HNO_3=HCN+NaHSO_4+2NO.$$
$$(2)\ 2NO+3O+H_2O=2HNO_3.$$

The first reaction is carried out in a solution maintained acid by sulphuric acid. The nitric acid required for the start and that consumed is made up by additions of sodium nitrate. Sodium sulphocyanide is used instead of calcium salt, as an insoluble precipitate of calcium sulphate would occur if the latter were used. The process may be divided into two stages. (1) The decomposition of the sulphocyanide. (2) Recovery of nitric acid. The operation takes place in a series of earthenware jars connected with each other and each fitted with an exit pipe for gases and an inlet pipe for the steam which heats and agitates the contents. The first jar of the series is furnished with inlets for the sulphocyanide and niter solutions, for vitriol, and for the return of the recovered nitric acids and also for

the incompletely reduced nitrogen oxides washed from the scrubber. The second jar is fitted to receive the water and dissolve hydrocyanic acid returned from the condensing apparatus. To begin the operation the jars are first filled with dilute acid and this heated until near boiling point by means of steam. The sulphocyanide solution with a strength of 10 lb. of potassium sulphocyanide per cubic foot is run in together with the nitrate solution, and the supply of liquor and steam so regulated that while the temperature is maintained, the liquor running away from the first jar is free from sulphocyanide and that leaving the last jar of the series is free from hydrocyanic acid. Thus the first jar acts as the decomposer, while the other jars boil out the various gases produced in the reaction and the hydrocyanic acids returned from the condenser. Although the thermal equations of the reactions show a balance of 57 calories per molecule of hydrocyanic acid to the good, it is necessary to use steam. This use is apparent when we consider the steam carried over with the gases and the large volume of spent liquor running away at practically boiling point, since the reactions take place only at this temperature and this heat is necessary to expel the hydrocyanic acid. The spent liquor, practically a solution of sodium bisulphate, which runs away is tested for sulphocyanide, hydrocyanic acids and also for nitric acid, thus forming a check on the working of the process. It usually is free from sulphocyanide but contains under 1% of hydrocyanic acid.

The gases are so generated that the products of the decomposition of the hydrosulphocyanide consist almost wholly of hydrocyanic acid and nitric oxide in the proportion of one volume of the former to two volumes of the latter; small proportions of CO and NO_2 are also contained. These gases are then passed through a scrubber built of earthenware pipes packed with flints, through which a small quantity of water is constantly flowing. The water removes the absorbable nitrogen oxides without lowering the temperature materially and consequently without absorbing the hydrocyanic acid. The gases then pass to a condensing apparatus consisting of cast iron pipes externally cooled by water. The steam is condensed here, and the water naturally contains a fair percentage of hydrocyanic acid. The liquor is run back by a special pipe to the before mentioned second jar of the series. About 30% of the hydrocyanic gas passing through the condenser is thus returned as a saturated aqueous solution. The gases are now passed to the cyanide absorbers. These are of cast iron cooled by a water jacket to prevent decomposition of the cyanide by the caustic alkali due to the rise of temperature resulting from the reaction. The caustic liquor has a density of 75° Tw. The gas is allowed to bubble through this solution, the hydrocyanic acid being absorbed, while the nitric oxide passes on. Over 90% of the hydrocyanic acid is eliminated in the first absorber, which considering the weak nature of the acid seems remarkable. This stage of the operation has to be carefully watched to prevent an excess of HCN, as then the cyanide becomes brown and is considered spoiled.

The nitric oxide liquors which have passed through the absorber are now ready for reconversion to nitric acid. This is brought about by passing the gas mixed with an excess of air through two stoneware towers packed with flints over which water is allowed to flow. Air is admitted at the bottom of

the first recovery tower and the quantity of it as well as the quantity of water has to be regulated carefully in order to produce nitric acid of uniform strength. It is necessary to use an excess of air above the theoretical amount required. The resultant liquor has a density of about 25° Tw. and is returned to the first decomposing jar. The cyanide has been so far recovered in the form of a liquor from which the recovery of the solid salt is more difficult than would first seem, as there is a considerable loss in the evaporation owing to the formation of ammonia due to the action of superheated steam imprisoned in the cyanide mass. The process was the invention of Dr. Raschen and is described in full by James P. Conroy.*

Progress in the Electrolytic Chlorate Industry in 1899.

By John B. C. Kershaw.

Production.—The year 1899 has been marked by considerable developments in the electrolytic chlorate industry. The new chlorate factory of the North American Chemical Co., at Bay City, Mich., has been completed and put into full work; a new factory is now in course of erection by the Wenlock Electrolytic Co., at Little Wenlock, in England; and a very large factory is projected at the Ljungan Waterfall in Sweden. There are therefore now 10 factories producing chlorates of potash or soda, by the electrolytic method. Details of these are given in Table I. The total amount of power available for electrolytic chlorate manufacture is now considerably over 33,000 h. p., and taking the figures obtained from the Société d'Electrochimie, for the production at Vallorbes in the early days of the industry (900 tons per annum for 3,000 h. p.), it can be calculated that were the whole of the available power in these 10 factories constantly devoted to the production of potassium chlorate, the output of the electrolytic works would amount to about 10,000 tons per annum. This is in excess of the whole world's consumption of chlorates, which has

ELECTROLYTIC CHLORATE FACTORIES IN AMERICA AND EUROPE.

Name of Company.	Locality of Works.	Horse-Power.			Process Used.	Capital of Company.	Date Start'd.	Rema'ks
		Type.	Available.	In Use.				
Société d'Electrochimie	Vallorbes, Switzerland	W.	3,000	3,000	Gall & Montlaur.	$336,400	1890	
Société d'Electrochimie	St. Michel, Savoy	W.	6,900	4,000			1891	
Elektrochem. Werke, Bitterfeld	Bitterfeld	S.					1894	Use brown coal.
Elektrochem. Werke, Rheinfelden	Rheinfelden	W.	8,000			720,000	1898	
Superphos. Akti'n Gesellsch. zu. Stockh'm	Mansboe	W.	5,000	4,000	Carlsen.		1894	
Consorti'm f. Elektrochemische Industr'n	Golling, Austria	W.	4,500		Kellner.	1,440,000	1896	
M. M. Corbin & Cie	Chedde, Savoy	W.	10,000		Corbin.		1897	
Chemical Construction Co.	Niagara Falls, N. Y.	W.		500	Bl'm'nb'g		1897	
National Electrolytic Co.	Niagara Falls, N. Y.	W.		1,100	Gibbs.		1898	
North American Chemical Co.	Bay City, Mich.	S.			Hurter.	600,000	1898	
Wenlock Electrolytic Co.	Little Wenlock, Engl'd	S.				576,000	not yet started.	

* *Journal of the Society of Chemical Industry*, May 31, 1899.

been estimated at 9,000 tons, though doubtless the consumption is rapidly increasing.

But many of the companies named in Table I. carry on other manufactures besides that of potassium or sodium chlorate, and the low price to which these products had sunk at the end of 1898, has caused the manufacturers to be ready to devote their electric power and plant to more profitable products. Thus the factories at St. Michel, at Bitterfeld, at Golling, and at Rheinfelden, are known to be using a portion of their available power for the production of other chemicals, and possibly others of the factories named are following in their footsteps. The writer, therefore, estimates the present output of chlorate by the electrolytic factories in America and Europe, to be between 5,000 and 6,000 tons per annum, or about two-thirds of the total supply; and if any change in the margin of profit should occur, he believes that this output will undergo considerable increase.

Price.—The European prices of chlorates during 1899 have remained fairly steady, a result due to the agreement made in the autumn of 1898 between the chief English, French and German manufacturers.

The English prices, which had, prior to the agreement, fallen to 3d. per lb. for the potassium salt and 3½d. per lb. for the sodium salt, were raised to 3¾d. per lb. and 4½d. per lb., respectively; and they have maintained that level throughout 1899. It is interesting to note that before the advent of the electrolytic process, potassium chlorate was selling for between 5½d. and 6d. per lb., and that a fall of 25% in values has occurred within the last few years.

Process and Works Details.—Illustrated descriptions of the chlorate works at St. Michel, and at Chedde, have appeared in the papers named below.* Abstracts of these descriptions appeared in Vol. VII. of THE MINERAL INDUSTRY.

The Kellner process, which is reported to be at work at Golling, is similar in principle to the Gall & Montlaur, and to the Corbin processes; but special forms of anode are used, and a small percentage of calcium hydrate is added to the salt solution contained in the cell. This addition is supposed to lessen the loss by reduction at the cathode, and thus to increase the efficiency of the process.

The writer has been unable to obtain any details of the Carlsen process and cell, which are in use on a large scale at Mansboe, in Sweden. The Swedish patent is No. 3,614 of 1890, and according to Dr. F. Peters is likely to invalidate many of the later patents.† Electrolytic chlorate manufacturers appear, however, to have determined to avoid litigation, and up to the present the industry has provided no work for the law courts. It has been more than once remarked that Charles Watt, the pioneer in all electrolytic processes for the decomposition of salt solutions, in British Patent No. 13,755 of 1851, clearly forestalled all later inventors as regards the chief features of the process and cell for chlorate production; and it is doubtless a knowledge of this fact that renders the owners of more recent chlorate patents somewhat chary about enforcing their claims upon those who may be infringing them. The

* *Electrical Review*, London, Nov. 25, 1898; *Engineering and Mining Journal*, June 10, 1899.
† *Dingler's Polyt. Journal*, Vol. CCCIX., p. 215.

Hargreaves chlorate process is in operation at Messrs. Bowman Thompson & Co.'s works in England, and at the St. Gobain works, near Paris. This process has been wrongly described as electrolytic; but is in fact merely an improved method of bringing about the chemical reaction between the chlorine and the sodium salt. The chlorine may be produced either by electrolysis, or by the older chemical methods from hydrochloric acid. By the older method of manufacturing sodium chlorate, great loss occurs on crystallization, owing to the excessive solubility of the salt in water, and Hargreaves has sought to avoid this loss by allowing the chlorine to act on semi-dry sodium-carbonate in a tower, and by so regulating the supply of steam to this tower that only the more soluble sodium chlorate should be dissolved. A full description of this method of manufacturing sodium chlorate is given by the inventor in the paper named below.* For potassium chlorate production, the method is less advantageous, since the potassium salt is much less soluble, and its separation from the mother liquors by crystallization is attended by less serious loss.

With regard to the factory of the North American Chemical Co., at Bay City, it was stated by the chairman at the 1899 annual meeting of the United Alkali Co., in Liverpool, that the works were in full operation and that good results were being obtained. The plant was designed by the late Dr. Hurter and by Mr. Duff, the company's engineer, and it is probable that some modified form of the cell patented by Dr. Hurter, in 1893 (No. 15,396, English Patent), is used. The chief feature of this cell is, that the wall serves as cathode, and that a cement lining to this wall, acts as the diaphragm. The cells are arranged one above the other in tiers, and the electrolyte flows through the vertical series of four or more cells. The weak point in the original form of cell, is the absence of any provision for dealing with the hydrogen liberated at the cathodes; and this detail has no doubt been dealt with by altering the construction of the cell. The Bay City works has a capacity of 28 tons potassium chlorate per week; but the writer is unable to say whether it is in full operation. Two large dynamos supply the current and 672 decomposing cells are stated to be installed.

The generating machinery for the supply of current to the works of the National Electrolytic Co., at Niagara Falls, N. Y., was described in the *Electrical World* early in 1899.† The dynamo yields a continuous current of 5,000 ampères at 175 volts pressure, and this is transmitted to the factory by means of an aluminum conductor. Allowing for a 10% loss in conductors and electrical connections, the number of decomposing vats worked in series by this current is probably about 26. This factory has been twice enlarged, and according to Dunlap a third enlargement is now in progress. Some details of the processes used by the two factories at Niagara, and of the plant at the Chemical Construction Co.'s works, will be found in the papers named below.‡

The writer has been unable to obtain any details of the process which is to be used at the Ljungan Waterfall, at Olby, in Sweden. According to a writer in *Engineering*,§ the company who are exploiting this fall, have a capital of

* *Electrochem. Zeitschrift*, April, 1899.
† *Electrical World*, Jan 14, 1899.
‡ *Electrician*, Feb. 19, 1897; *Cassier's Magazine*, July, 1897.
§ *Engineering*, Oct. 27, 1899.

$513,000, and intend to use one-half of the available power (10,000 h. p.) for carbide production, and the other half for the manufacture of chlorates. Their present capital will certainly require considerable increase, before these schemes are completed.

Theory.—The results of several important investigations relating to the theory of the electrolytic chlorate cell, have been published during 1899. The researches of Oettel upon this subject, were referred to by Dr. Borchers, in one of the early issues of this Annual.* It is interesting to note that Schuckert & Cie have patented the addition of alkaline bicarbonates to the chlorate cell, and that Kellner has patented a similar addition of calcium hydrate. In view of Oettel's published results, it is questionable whether these patents are valid. These improvements of the original process are intended to be worked in a cell without a diaphragm, but Waubel, in the paper named below,† has stated that—"by use of a diaphragm, and of an anode solution of sodium bicarbonate, the whole of the chlorine carried into the anode chamber can be converted into chlorate. The electrolyte should be kept at 60 to 70° C., the current density should be 5 to 10 ampères per sq. dm., and the E.M.F. should be 4 to 5 volts. The solutions of bicarbonate for the anode chamber, and of chloride for the cathode chamber should be saturated. In order to maintain the former in this state, additions of fresh amounts of bicarbonate are necessary from time to time. In the anode chamber 6·5 g. of chlorate are obtained for every 7·5 g. of bicarbonate used, while in the cathode chamber a 10% solution of sodium hydrate can be obtained.

"Comparing the electrical energy necessary to produce chlorate by this process with that required in the experiments of Oettel and of Gall & Montlaur, the following comparative figures are obtained for the production of 88.14 g. potassium chlorate:

"Oettel, alkaline electrolyte; without a diaphragm, 736 watt-hours. Gall & Montlaur, alkaline electrolyte; with a diaphragm, 1,003 watt-hours. Waubel, bicarbonate in anode chamber; with a diaphragm, 951 watt-hours.

"The latter process has the advantage over the other two, that sodium hydrate is concurrently produced, and can be recovered from the cathode solution. Bromates and iodates can be obtained by a similar method of procedure."‡

"Müller, in a contribution published still more recently,§ has recorded the results of an investigation which opens the door to still more striking improvement in the efficiency of the electrolytic chlorate process. The following is an abstract of Müller's paper:‖

"The author has found that the addition of a small amount of a chromate salt to the electrolyte, enables a very high current efficiency to be obtained, without the use of diaphragms between anode and cathode compartments of the cell. Full details are given of the electrolytic apparatus used in these experiments, and of the chemical methods used for determining the electrical efficiency of the process.

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* The Mineral Industry, Vol. IV., 1895.

† *Industrie Electrochemique*, 1899, 8, pp. 14-15.

‡ *Science Abstracts*, No. 1239. Aug., 1899.

§ *Zeitschrift f. Elektrochemie*, Vol. V., pp. 469-473.

‖ *Science Abstracts*, No. 1,408, 1899.

"The following figures were obtained in experiments, each of which lasted 21 hrs.: Sodium chlorate, current efficiency, 67·9%; potassium iodate, current efficiency, 97·0%; potassium bromate, current efficiency, 89·8%. It was found that platinum formed the best anode material.

"The author concludes his paper by presenting two theories to account for the remarkable action of the chromium salt, in accelerating the oxidation of the electrolyte. According to the one, a thin film of chromic hydrate is formed over the surface of the cathode, and acts as a diaphragm to protect the oxygen salts in solution from the reducing action of the hydrogen liberated at the true surface of the cathode. According to the second, a film of an alloy of chromium and platinum forms upon the surface of the cathode, and acts in a similar manner.

"It was found that chromates gave better results than any other oxygen salts, and that manganates have little or no effect in accelerating oxidation of the electrolyte."

Prof. Foerster, in an address delivered before the sixth annual meeting of the German Society of Electro-chemists, has given a very useful summary of the investigations carried out by Oettel, Wohlwill, Winteler, Müller and himself in relation to the electrolytic decomposition of chloride solutions.*

At the conclusion of this address Prof. Foerster set forth the conditions which the latest researches had shown to be the most favorable for the production of chlorate in the cell. These were as follows:

1. Presence of calcium hydrate, or chromium hydrate, at the cathode. 2. A slightly *acid* electrolyte. 3. A large volume of electrolyte in rapid circulation. 4. A temperature of at least 40° C.

Prof. Foerster also stated that the ideal electrolytic chlorate process would be one in which pure calcium chloride was used, in place of the sodium or potassium salt. With such an electrolyte, a current efficiency of 87% can be attained.

The importance of these researches relating to the chemistry of the electrolytic chlorate cell, cannot be too much emphasized. The current efficiency of the electrolytic chlorate process, as originally carried out, was only 35%, and the energy efficiency only 18%; and as the cost of electrical energy represents a very large proportion (30%) of the total cost of production, there is evidently a wide margin for effecting improvements in the process, and reductions in its cost.

Properties.—The year 1899 was marked by a most disastrous explosion at one of the English chlorate factories, owned by the United Alkali Co., and as a result of the official investigation into the cause of this explosion, a previously unknown property of potassium chlorate has been brought to light.† The experts who have prepared the official report have arrived at the opinion that chlorate, when subjected to intense heat, liberates its oxygen with explosive violence, and they attribute the St. Helen's disaster to such an explosion of about 5 tons of the chlorate; this explosion being produced by the rapid and intense heating of the chlorate in the storehouse.

* *Zeits. f. Elektrochemie*, Vol. VI., pp. 11-23. † *Engineering and Mining Journal*, July 1, 1899.

QUICKSILVER.

ALTHOUGH there was a greater demand for quicksilver in 1899 than in the previous years, the production in Spain and in California fell below the returns of previous years. Demand was due to increased activity in gold and silver mining, as well as to developments in the manufacture of chlorine and caustic soda electrolytically. The Rhodin cell used in this industry, has been the cause of a considerable consumption of mercury. It is estimated that one ton of mercury is required for each ton of caustic soda produced weekly. This demand is not likely to be continuous, as there is a very slight loss of quicksilver in the operation, and the number of plants operating under this process will probably be limited.

California.—While the price of quicksilver rose nearly $10 a flask during the year, yet the production fell from 30,116 flasks to 28,618 flasks. The following table shows the production of the leading mines during 1899, as well as a comparison of the totals of previous years:

Month.	New Almaden.	Napa Con.	Mirabel.	Ætna.	Great Western.	Great Eastern.	New Idria.	Altoona.	Abbott.	Reddington.	Knox.	Total.
January	460	600	.....	300	89	165	400	215	.....	90	76	2,395
February	425	600	.....	300	90	225	400	225	47	85	90	2,487
March	425	500	.....	300	100	210	400	223	21	70	38	2,287
April	475	435	.....	260	100	127	400	211	6	132	.....	2,146
May	400	465	.....	340	150	170	400	190	.....	181	22	2,318
June	250	550	.....	300	182	76	400	313	40	40	32	2,283
July	350	405	.....	300	160	104	400	294	18	.....	53	2,084
August	350	495	.....	300	119	230	400	231	42	.....	27	2,194
September	300	415	200	260	137	228	400	305	24	30	43	2,342
October	300	485	247	440	141	194	400	300	13	85	.....	2,605
November	300	450	61	400	157	205	380	317	22	75	9	2,376
December	300	450	20	300	100	201	400	320	69	94	35	2,289
Total, 1899	4,495	5,850	528	3,800	1,525	2,135	4,780	3,134	302	882	435	*a* 28,618
Total, 1898	5,875	6,850	108	3,450	1,128	1,704	5,000	4,032	189	990	790	30,116
Total, 1897	4,700	6,200	391	3,600	2,445	1,591	3,605	686	773	1,488	*Nil.*	25,479
Total, 1896	6,200	5,000	1,125	3,800	2,165	1,126	1,335	4,203	1,189	1,443	*Nil.*	*b* 29,271

(*a*) Includes 312 flasks, the output of the Sulphur Bank mine, and 500 flasks for various other mines not included in the list. (*b*) Includes 1,685 flasks produced by the Sulphur Bank mine, which made no output in 1897. or 1898.

The Ætna, New Idria and Napa Mining companies, all controlled in Boston, paid dividends during the year, though the New Almaden mine of the Quicksilver Mining Co., which in former years was the leading producer, was able to give only $22,000 profit to the shareholders.

The Napa Consolidated Quicksilver Mining Co., the largest producer, marketed 5,850 flasks in 1899 against 6,850 in the previous year, although the tonnage treated increased from 32,480 tons to 36,021. A considerable amount of development work was done with promising results. Stocks of quicksilver have been reduced, and the savings for the year show net earnings of $91,117·25, from which dividends amounting to $90,000 were paid. The receipts last year were $6·59 per ton of ore treated, the expenses $4·00 and the profits $2·53.

The New Idria Quicksilver Mining Co. produced 4,780 flasks, against 5,000 flasks in the previous year, although 20,629 tons were treated against 18,627 tons in the previous year. The net earnings were $101,135·63, and dividends to the amount of $90,000 were paid. Development work has placed a large amount of ore in sight. The San Carlos claim has been developed, and it is anticipated that it will be the source of a considerable ore supply. The receipts were $9·62 per ton furnaced, the expenses $4·72, and the profit $4·90.

The Ætna Quicksilver Mining Co. produced 2,800 flasks from 18,084 tons of ore against 3,450 flasks from 18,394 tons in the previous year. Unlike its sister companies it shows an increased production from a decreased tonnage. Net earnings amounted to $69,057·65, and the dividends to $50,000. Ore is practically exhausted in the upper levels, and prospecting is being earnestly conducted to locate a possible continuation of the ore chute.

Some development work was being done on the Oakland mine at the end of the year. The record of the property shows a production of 6,000 flasks and profits somewhat in excess of $200,000, although under the ownership of the late Thomas Bell mining was conducted in a crude manner. The Guadaloupe mine, about six miles from Los Gatos, which was closed down in 1880, was reopened in the latter part of the year. The old miners, mainly Mexicans, have traditions that a considerable quantity of ore is left in the stopes. The mine, however, is extremely wet and difficult to work. The St. John mine, near Vallejo was reopened. It had a record from 1873 to 1880 of producing 11,528 flasks, valued at $504,810. In Sonoma County the Boston quicksilver mine was reopened by a company formed and controlled by parties interested in the Napa Consolidated and other California mines.

Texas.—The mines in Brewster County produced a small amount of quicksilver, although the methods of extraction have been extremely crude. The total production of Texas amounted to 261 flasks.

QUICKSILVER STATISTICS OF THE UNITED STATES.

Year.	Production.			Exports.			Year.	Production.			Exports.		
	Flasks.	Met. Tons.	Value.	Flasks.	Met. Tons.	Value.		Flasks.	Met. Tons.	Value.	Flasks	Met. Tons.	Value.
1894..	30,440	1,056	$1,095,840	14,408	500	$397,528	1897....	26,079	965	$910,418	13,173	475	$394,549
1895..	33,978	1,179	1,313,589	15,542	539	482,085	1898....	30,493	1,058	1,109,945	12,880	445	440,897
1896..	29,863	1,036	1,104,997	19,944	692	618,437	1899....	28,879	998	1,155,160	16,518	573	609,686

RANGE OF QUICKSILVER PRICES PER FLASK AT SAN FRANCISCO IN 1899.

Month.	Domestic.		Export.		Month.	Domestic.		Export.	
	Highest.	Lowest.	Highest.	Lowest.		Highest.	Lowest.	Highest.	Lowest.
January.....	$42·00	$41·00	$39·00	$37·50	July.....	$43·50	$43·00	$40·00	$39·00
February.....	42·00	42·00	39·50	38·50	August.....	43·50	43·50	40·00	39·50
March.....	43·00	42·00	39·50	39·00	September..	46·00	45·00	41·00	40·50
April.....	43·00	42·50	39·50	38·00	October.....	47·00	46·00	43·50	41·00
May	42·00	42·00	39·00	38·50	November..	50·00	47·00	46·00	45·00
June.....	43·00	42·00	39·00	38·50	December..	52·00	50·00	47·00	45·50

QUICKSILVER PRODUCTION OF THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Austria.	Canada.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1895.....	535	2·5	199	213	434	1,506	1,179	4,069
1896.....	564	2·0	186	218	491	1,594	1,086	4,021
1897.....	532	0·3	192	294	616	1,728	965	4,327
1898.....	491	*Nil.*	178	353	862	1,691	1,058	4,128
1899.....	504	*Nil.*	201	324	360	1,357	993	3,739

(*a*) From official reports of the respective governments and direct reports of the producers to THE MINERAL INDUSTRY. The figures for Austria and Italy for 1898 are due to V. Spirek, that for Russia to the sole producer, and that for Spain to the *Revista Minera*.

GENERAL QUICKSILVER STATISTICS. (*a*)

	1892.	1893.	1894.	1895.	1896.	1897.	1898.	1899.
	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)
Shipments from Spain to London.....	47,321	44,670	42,414	40,409	40,949	46,577	46,367	45,739
Shipments from Italy to London.	6,765	6,680	8,700	5,775	3,900	4,450	5,650	6,206
Total.....	54,086	51,350	51,114	46,184	44,799	51,027	52,017	55,935
Shipments from London....	46,055	42,265	41,046	37,937	31,273	31,734	32,268	33,101
Maximum price of Spanish.	£7 15s.	£6 17s. 6d	£6 15s.	£7 7s. 6d.	£7 5s.	£7 7s. 6d	£7 15s.	£9 12s. 6d
Minimum price of Spanish..	6 1s.	6 2s. 6d	5 10s.	6 7s. 6d.	6 8s. 6d	6 12s. 6d	7 0s.	7 15s.

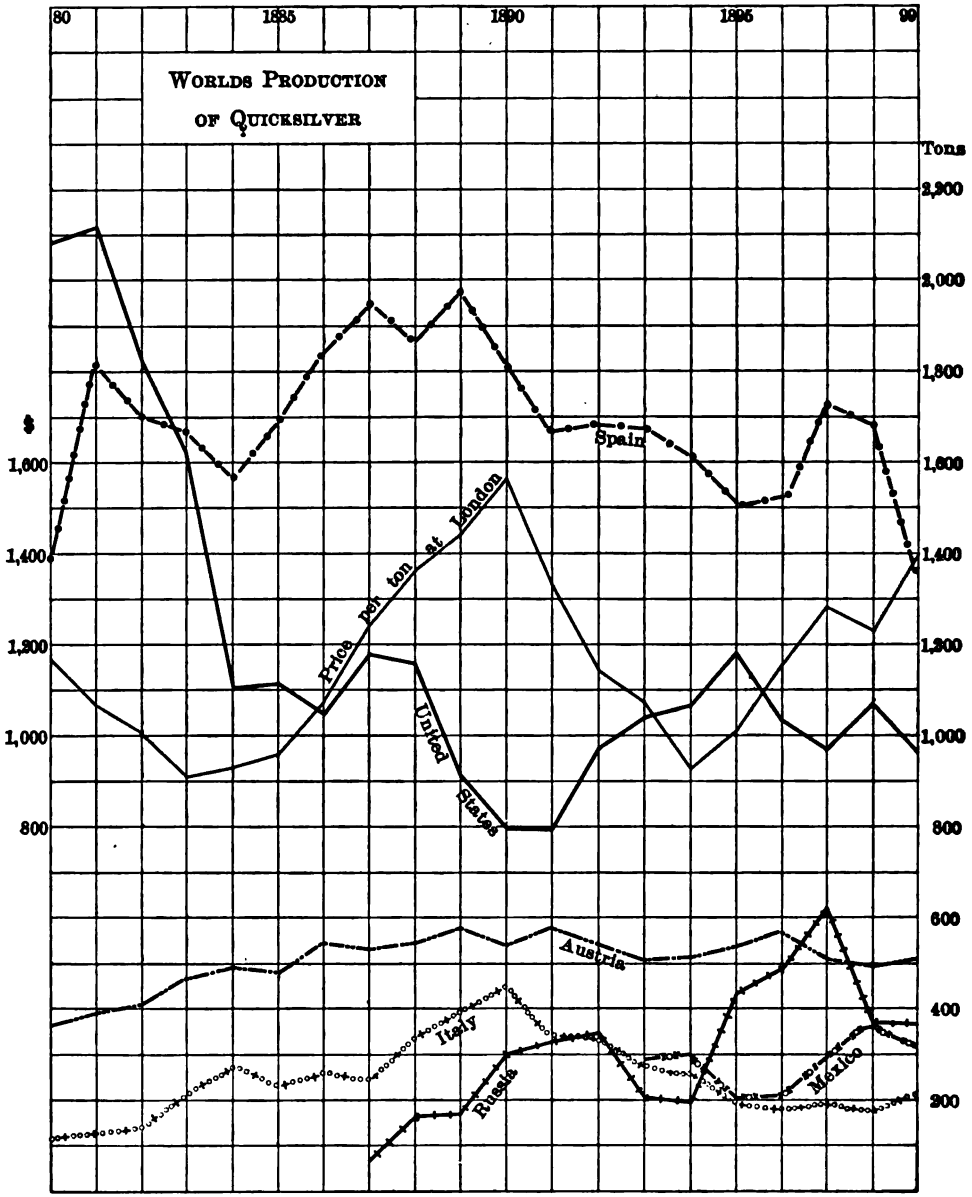
(*a*) From W. Sargant & Co.'s Annual Metal Circular. (*b*) American, Russian and Italian flask, 76·5 lb. (34·7 kg.); Mexican flask, 75 lb. (34·03 kg.); Spanish flask, 76 lb. (34·5 kg.).

Australia.—Quicksilver has been found at Yulgibar on the Clarence River. According to John Plumber, the ore contains from 3 to 5% of quicksilver. An English company was formed to work it. A small amount of quicksilver was produced at Kilkivan, in Queensland, from which the Gympie mines are supplied for amalgamating purposes.

Austria.—The production of quicksilver from the mines of Idria in Saint Anne remained at about the same figure as in the preceding year. A few hundred flasks are annually produced as a by-product at Kolterbach in Upper Hungary.* The mines here are of great antiquity and have been worked to the depth of 1,500 ft. for irregular bunches of tetrahedrite both argentiferous and mercurial. The vein stuff itself is spathic iron which at present is the sole source of profit. The ore is roasted in closed kilns, of which there are 36 in number, varying from 980 to 1,120 cu. ft. capacity and holding from 56 to 62 tons of ore each. These have hinged covers which can be lifted on charging and have a central outlet pipe with an automatic valve which closes the pipe when the cover is open. These pipes run to condensing towers

* H. Wedding, in *Verhandlungen des Vereins zur befoerderung des Gewerbfleisses*, May, 1899, p. 185.

filled with broken limestone on grids, there being one tower for each 12 kilns. A Körting ejector draws the gases from the kilns and with steam of 75 lb. pressure delivers about 3,600 cu. ft. per minute to the towers. An analysis of the



raw ore showed FeO, 43·24%; Fe_2O_3, 0·60%; MnO, 2·01%; BaO, 0·47%; CaO, 0·30%; MgO, 5·07%; Ca, 0·4%; Sb, 0·06%; Hg, 0·05%; S, 0·78%; CO_2, 30·39%. The output for the year was 55,000 tons which went to the blast

furnaces at Wittkowitz in Moravia. There are two different products containing mercury obtained from the plant: (1) Flue dust from the ejector with 78·1% of mercury; (2) slimes from condenser with 49 to 76%. The products are distilled in a Cermak furnace (which was described in THE MINERAL INDUSTRY, Vol. VI.) with low picked ore that runs 1·85% mercury, and washings carrying about 0·5%.

China.—The Anglo-French Quicksilver Mining Concession, Ltd., was formed with a capital of £310,000, to work a concession in the Province Kivei Chau. These properties have been worked in a small way by the natives. Forty men latterly were employed against 2,000 previously engaged. The ore is said to occur in limestone. Clay retorts are used, from which the sulphur and moisture passes out through the side, and the mercury collects in a bell placed over the top.

Italy.—The mines at Siele and Cornacchino were operated throughout the year, and the furnaces at Abbadia, San Salvadore, were in operation since March. The mines of Montebuono started their furnaces in December. The total production of the Monte Amiata district was 201 metric tons of quicksilver. Although many new companies have been formed owing to the high price of quicksilver, only one, the Santa Fiora, has commenced to develop its mines.

Madagascar.—Cinnabar has been discovered about 15 kilometers northeast of Miadribaco.

New Zealand.—Cinnabar has been discovered at Mawngakirikiri. The formation of decomposed andesite is said to be 60 ft. wide, through which veins 3 to 4 in. wide, carrying cinnabar, pass frequently. The mines in the Waitahuna range have been developed satisfactorily during the year. The vein averages about 18 in. wide, but the ore occurrence is irregular. The Thames Quicksilver Mines, Ltd., is a company formed to work the deposit on Kaweranga Creek in the Thames district. Alexander McKay, the Government Geologist, reported favorably on the property.

Peru.—The San Andres mine, in the Department of Juancavelica, has a vein 0·5 m. wide. The ore is low grade, running slightly above 1%. The production does not exceed 1 cwt. of mercury daily. The Poderosa, San Julian and Carmen mines contain ore running from 1 to 5%. The production from these is about 360 cwt. monthly.

Russia.—Although it was anticipated that the production of Auerbach & Co. would show a substantial increase in 1899 above the previous year, there was a little difference, 360 metric tons being produced as against 362 in 1898.

Spain.—The production in 1899 was less than that in the previous year, it being the purpose of the owners and lessees to decrease stocks. The Almaden mine produced about 1,300 metric tons, and other properties 57 metric tons.

THE METALLURGY OF QUICKSILVER.

(Communication from Vincente Spirek.)—In THE MINERAL INDUSTRY, Vol. VII., pp. 585 and 587, Mr. Symington makes the following observations on "The Metallurgy of Quicksilver in California," which need correction: (page 585) "The furnace (Newcombs) treated 50 tons every 24 hours, and when full,

holds about 40 tons, so that the ore remains under treatment about 20 hours. This is in very marked contrast with the two and three hours stated as the practice in Italy by Spirek in THE MINERAL INDUSTRY, Vol. VI., p. 577." This extract is a mis-translation and should be: "The furnace is drawn and charged every two or three hours and treats in 24 hours 16 to 26 metric tons. Its capacity when full is 45 metric tons; therefore the ore is under treatment from 41 to 68 hours." It is roasted until the last trace of quicksilver is driven out, the furnace being maintained at a bright red heat. If it were operated at a dark red heat leaving small traces of quicksilver as in California, it would treat about 55 metric tons in 24 hours. The high temperature is needed in order to warm the fresh ore and the necessary amount of air. The gases leave the furnace at about 260 to 300°C., and the spent ore at about 100°C.

(Page 587.) "This differs from the practice in Italy, where half of the condensers are shut off by valves, and the charge in the furnace is lessened during the cleaning up. . . . About 30% of the total yield is got from the soot, this, too, differing from the 70 to 80% reported to be obtained in Italy from a similar grade of ore." The operation of the condensers invented by Cermak is entirely different from the condensing chambers employed in California. The Cermak condenser is patterned after the Gjers air generator, but is employed for just the opposite purpose—that is, for cooling. The gases as they leave the furnace are carried into a coil of pipe of 10 turns which is water-cooled on the outside. The result is very favorable, almost 90% of the entire production of quicksilver and soot is collected in the lower chamber of the condenser over a length of not more than 4 m. Each furnace has two of these condensers which are cleaned monthly, one after the other, so that the furnace remains in operation continuously. The whole plan of condensation is so laid out that the quicksilver and furnace gases pass through the condenser very slowly, there being a pressure of 0·25 mm. (water column). If one wishes to hurry the gases the furnace could be maintained in full operation. This process lasts only about four hours for both condensers and this time can be employed to advantage in heating the furnace, in transferring the quicksilver from the furnace to the condensers, and in inspecting all the pipes and canals of the furnace. By the effective operation of the condenser almost all the product is assembled in the collecting chamber. The condensed quicksilver is run off and the rich soots are squeezed in a soot-press. It will be observed that the process is not so rapid as one might believe. The Cermak condensers have been substituted for chamber condensers both in Idria and in Italy, and Nikikowka has built a smelting works in Russia which is patterned after the principle of Spirek and Cermak.

RARE ELEMENTS.

By Victor Lenher.

During the past year, the newly discovered elements have been subjected to the criticism which invariably follows the announcement of a discovery.

Argon has been prepared in a pure state, and there appears to be much evidence as to the existence of the new radio-active elements. The rare earths have received a due amount of attention, although the decisions in favor of the Welsbach patents have to some extent caused less work than usual in the line of investigation. Metallic cæsium has been prepared by the method of Winkler, and the preparation of metallic lithium (see the separate caption on "Lithium") is worthy of note. Tellurium has also appeared in larger quantities than in previous years, although as yet there appears to be no demand for any of these metals.

It should be remembered that while a use has been found for many of the rare and high-priced elements, there are many more that owe their present market price only to the fact that the demand is limited to specimens for museums or to the small amounts that are needed for research work. For example, in 1895, thorium nitrate sold at $200 a pound while to-day it can be bought in Germany for $3·33 per pound. On the other hand, a small quantity of any one of the fancy priced elements would overstock the market, and although manufactured at a mere fraction of the present selling price, nobody would buy it at any figure.

Argon.—Ramsay has prepared the gas in a pure condition by passing atmospheric nitrogen twice through a heated tube containing magnesium and lime, and then over red-hot oxide of copper. The crude argon was liquefied by means of boiling liquid air. The liquid was distilled yielding first neon and helium, then argon, and finally metargon. The middle portion was redistilled and gave pure argon. The density of the pure gas is 19·957, while that of the crude was 19·94. Its refractivity is 0·9665, while that of the crude was 0·961.

Hempel finds that lithium is five times as efficient as magnesium for removing the nitrogen from argon, magnesium with lime eight times, while a mixture of one part magnesium powder, five parts of lime and one-fourth part sodium is twenty times as rapid in its action as magnesium. Berthelot has repeated his work on the absorption of argon by organic bodies and finds it is best absorbed by benzine, while other organic substances take up small quantities. As before, his experiments were made with the aid of the electric current.

Cæsium.—The method which has heretofore served for the preparation of cæsium has been the electrolysis of the double cyanide of cæsium and barium

using aluminum poles. This method, which is due to Setterberg, has been supplanted by the application of the Winkler method. Owing to the great affinity which the element possesses for oxygen, great difficulty has been hitherto experienced in its preparation. Two applications of the Winkler method now appear.

Erdmann and Menke* prepare metallic cæsium by ignition of the hydrate with metallic magnesium in a current of hydrogen. Cæsium alum, being the most available cæsium compound, is treated with barium hydrate which precipitates the alumina as well as the sulphuric acid. The hydrate of cæsium is mixed with one-half its weight of magnesium powder (the magnesium powder must be previously heated in hydrogen to get rid of the fat with which it is always mixed). The mixture is introduced into a malleable iron tube, and a current of hydrogen passed through. The metal distils and is collected in a paraffin bath in the form of silvery drops. The yield is almost quantitative. The sp. gr. was determined to be 2·4. Graefe and Eckardt† prepare the metal by heating the carbonate with magnesium in hydrogen; a lower temperature is used than for potassium or rubidium. The metal was prepared so rapidly that it fell in a silver-white stream into the receiving vessel. The color of the metal is silver-white with a yellow shade, and is retained under paraffin oil. It ignites spontaneously in the air, floats on water, gives hydrogen rapidly and burns with a red violet flame. The yield of the metal is quantitative.

Calcium.—By means of the electric furnace, Moissan has been able to prepare the arsenide and the phosphide of calcium. The arsenide is made by heating 100 parts of calcium arsenate with 30 parts of petroleum coke, using a current of 1,000 ampères and 45 volts. The compound appears fused and crystalline. It is decomposed by water into calcium hydrate and the hydride. The phosphide, having the composition Ca_3P_2, has a dark red color and is made by heating tricalcium phosphate with carbon in the electric furnace.

Moissan remarks that the binary compounds of calcium are decomposed by water giving hydrated oxide and a gaseous hydride. Calcium hydride gives hydrogen, calcium carbide gives acetylene, the nitride gives ammonia, the phosphide, phosphine, and the arsenides and antimonides their corresponding hydrides.

Fluorine.—This element has at last been liquefied by Moissan and Dewar. Its boiling point is —187°C., and if air is admitted to the liquid, it instantly liquefies. Two layers are formed, the lower consisting of fluorine of a pale yellow color. Wood, caoutchouc, and ebonite float on the liquid, while amber remains suspended. Its sp. gr. is therefore about 1·14. Even at temperatures as low as —210°C., it instantly unites with hydrogen, producing light and heat. Oil of turpentine causes an explosion with a separation of carbon.

In the preparation of fluorine Moissan has shown that copper vessels can be used for the electrolysis of hydrofluoric acid as they are less attacked than other metals. A very thin layer of copper fluorine is formed, which is insoluble in hydrofluoric acid and unattacked by free fluorine. According to Moissan, when fluorine is passed into water cooled to 0°C., the oxygen which is set free contains 10 to 14% of ozone.

* *Journal of the American Chemical Society*, 1899, p. 359. † *Zeitschrift für Anorganische Chemie*, 22, 158.

Germanium.—Prior and Spencer* describe an argyrodite from Bolivia, which contains 75% Ag, 5% Ge, 3% Sn, 16·5% S.

Helium.—Travers finds that the amount of helium obtained from cleveite by the action of sulphuric acid in an exhausted tube is about twice that obtained by heating the mineral alone. If helium exists in combination in minerals, it would come off as hydride, which being unstable would immediately break up into helium and hydrogen. Hydrogen is found when fergusonite and cleveite are decomposed by acids, and from this the conclusion is made that helium exists in these minerals in a combined state.

Hydrogen.—Attempts have been made to solidify hydrogen since the fall of 1898, when liquid hydrogen was obtained in quantities of from 100 to 200 c. c. Dewar has finally succeeded in producing solid hydrogen and has determined more accurately the properties and constants of both liquid and solid hydrogen. The solidification was accomplished by jacketing a bulb of liquid hydrogen with a vacuum tube. When the pressure in the latter was reduced to 30 to 40 mm., the hydrogen suddenly solidified to a white frothy mass resembling frozen foam, and below it was a solid layer resembling ice. The solid melted at —257°C. or 16° on the absolute scale. The boiling point of hydrogen at 760 mm. is —252°C. or 21° absolute scale. Its sp. gr. at the boiling point is 0·07, and its maximum density not over 0·086. The lowest temperature yet attained was —259 to —258°C. or 14 to 15° absolute scale, which was reached by the evaporation of solid hydrogen in a vacuum. The critical temperature of hydrogen is 30 to 32° absolute scale, and its melting point is about one-half its critical temperature. As solid hydrogen does not possess a metallic character, it has been stated that it should be classed with the non-metallic elements, yet Bayley has shown from a simple calculation based on its melting point that hydrogen falls consistently in the first place of the alkali group of metals. The dangerous character of this work is seen when one realizes that at the low temperature employed, rubber becomes hard and brittle, cement cracks, and air leakage gives an extremely explosive mixture.

Iridium.—When iridium is purified by fusion with lead and subsequent treatment with nitric acid, the residue contains rhodium, osmium, lead, and gold. Leidie's method† for the purification of iridium consists in mixing finely powdered iridium with twice its weight of fused sodium chloride and heating to redness in chlorine. The product of the reaction is treated with cold water and acidulated with HCl. The solution is filtered, heated to 50 to 60°C., sodium nitrite added, a little at a time, until the solution becomes neutral to litmus. The solution is brought to boiling, allowed to cool and filtered. Impurities such as lead, iron and gold are removed while the filtrate contains the ruthenium, rhodium, and iridium as double nitrates and sodium osmate. Soda is then added in excess and chlorine passed through the solution when the osmium and ruthenium distil off as oxides. The residue is then evaporated with hydrochloric acid when the double chlorides are formed. The two double chlorides are heated to 440°C. in dry chlorine. The cold material is treated with cold chlorine water and filtered. The solution contains Na_2IrCl_6, which is treated

* *Chemisches Centralblatt*, 268, 1899. † *Comptes Rendus*, 122, 214.

with ammonium chloride and the ammonium salt formed. The ammonium salt heated in hydrogen gives pure iridium.

Lanthanum and the Didymiums.—Scheele* finds that it is easy to prepare pure lanthana from a mixture of lanthana and praseo-didymia, etc., by the crystallization of the double ammonium nitrate. However, praseo-didymia is not completely separated from lanthana.

Pure praseo-didymia is prepared by diluting the nitrate with water, acidifying with nitric acid, and precipitating the praseo-didymia as an oxalate. The supernatant liquor now contains all of the lanthana.

Melikoff and Pissarjewsky have obtained a peroxide of lanthana by the action of hydrogen peroxide on the nitrate. It is a white gelatinous precipitate resembling alumina.

Muthmann and Stutzel, in a spectroscopic study of the didymiums consider that praseo is composite. The composite nature of neo-didymium is still in doubt as it has not yet been obtained pure. A spectroscopic method was used for the estimation of neo and praseo. The neo-didymium was found to be twice the praseo-didymium in orthite, from Miask, in cerite from Riddarhyttan, and in allanite from Llano County, Texas.

Niobium and Tantalum.—Melikoff and Pissarjewsky† have prepared perniobic and per-tantalic acids by the action of hydrogen peroxide on the hydrated acids. Per-tantalic acid is white and the potassium salt is white, while perniobic acid is yellow and the potassium salt white. A new mineral, which has been named loranskite, has been found at Imbilax, Finland. It contains 49% Ta_2O_5, 13% of the cerium and yttrium earths and 20% ZrO_2. It is brittle, has a choncoidal fracture and a strong submetallic luster. Its hardness is 5, and its sp. gr. 4·6. The streak is greenish gray, and it is translucent on the edges.

Osmium.—Vezes shows that the oxidation and subsequent volatilization of osmium in air or oxygen are dependent on the physical condition of the metal, and on the method of its preparation.‡

Brizard has confirmed the work of Joly on the so-called osmiamic acid and gives it the formula $OsO(NO)OH$, which corresponds to a similar derivative of ruthenium.

Rosenheim and Sasserath§ have made an elaborate study on the derivatives of osmium; they form mixed double salts of osmic and sulphurous acids with alkali metals, by conducting sulphurous acid through a solution of perosmic acid in sodium or potassium hydrates. A careful study is also made of the double halides and the osmisulphites.

Palladium.—A discovery of this metal is reported near Mount Torrens, South Australia, which gave by government assay 17 oz. 6 dwt. 11 g. palladium with 16 dwt. 21 g. gold per ton. Its value is £112 per pound. Prof. Hodkinson, Capt. Waring, and Capt. Desborough find that palladium will not alloy with cadmium, and that there is a little more tendency to alloy with zinc. Their experiments were made in hydrogen and in vacuum.

Radium and Polonium.—These two new radio-active substances, discovered

* *Berichte*, 32 (3), 409.
† *Zeitschrift für Anorganische Chemie*, 20, 340.
‡ *Zeitschrift für Anorganische Chemie*, 1899, 20, 230.
§ *Ibid.*, 1899, 21, 122.

by M. and Mme. Curie have been noted in THE MINERAL INDUSTRY, Vol. VII., 594, 595. They were discovered in pitchblende. Polonium resembles bismuth, radium strongly resembles barium. The distinction of radium is that it is extremely radio-active while barium is not. By solution of the chloride in water and precipitation with alcohol, the substance may be fractionated until the chloride is 200 times more active than uranium. Mme. Curie* has fractionally crystallized 2 kilos of purified radiferous barium chloride which was extracted from half a ton of residues from uranium mineral. The treatment concentrated the radio-activity in the least soluble parts. Several fractions were made from the more active portions, and the activity was found to concentrate in the precipitates. The atomic weight increases with the activity. She concludes that the spectrum examinations and the experiments show that radium does exist and possesses a slightly higher atomic weight than barium.

Fritz Giesel† has obtained from other ores than pitchblende, a substance consisting chiefly of barium sulphate. The substance is similar to radium. The active constituent has not been isolated. The chloride, bromide, and iodide, phosphoresce without any previous illumination, especially when dry. In moist air they lose this property, but gradually regain it on heating. The stronger a specimen phosphoresces the lesser amount of Becquerel rays it emits.

Demarcay‡ has carefully studied the phosphorescent spectrum of radium and finds a new series of lines, eleven of which are different from barium. The radiant power is 7×10^4 as great as uranium.

Derbiene§ finds in pitchblende a substance not precipitated by hydrogen sulphide but precipitated by ammonia or ammonium sulphide. Its radio-activity is 100,000 times that of uranium. It differs from radium in not being luminescent, and resembles titanium in general properties.

Ruthenium.—Antony and Lucchesi‖ have prepared pure K_2RuCl_6 by fusing six parts of KOH with one part of ruthenium and adding potassium chlorate until the ruthenium dissolves. The operation is effected in a silver dish, producing the salt as minute red-brown crystals, soluble in water but insoluble in a solution of potassium chloride.

Selenium.—Szephansky has recently used selenium cells in the construction of his telelectroscope, owing to the property of changing its conductivity by light. Selenium cells for this purpose are made by Kohl, in Chemnitz, which have a resistance of 10,000 to 100,000 ohms reducible to a certain fraction by sunlight. In this case the red selenium when applied to a copper plate and placed under a negative, gives a picture. The copper is etched and can be used for reproduction.¶

Experiments made by Perreau on selenium show that the resistance of a selenium cell fell from 40,000 to 34,000 ohms when exposed to the action of X-rays at a distance of 5 cm. The reduction in resistance corresponds to that caused by a gas jet at a distance of 1·5 m. Electric waves have no effect on selenium, and in this respect X-rays are similar to light.

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* *Comptes Rendus*, Dec. 15, 1899.
† *Annalen der Physik und Chemie*, 1899, 2, 91.
‡ *Comptes Rendus*, CXXVII., 129.
§ *Comptes Rendus*, CXXVII., 129, 598.
‖ *Gazeta*, 29, II., 82.
¶ *Die Chemische Industrie*, May 1, 1899.

Matteucci and Guistiniani* state that a bright incrustation found during the eruption of Vesuvius, July 3, 1895, contained selenium, fluorine, phosphorus, and boron. Hugot finds that by treating a mixture of selenium and alkali metal with liquid ammonia an insoluble selenide, Na_2Se or K_2Se, is formed when the metal is in excess, while if selenium predominates, a polyselenide is formed.

Silicon.—This element and the silicides have been studied to a considerable extent recently. The latter present a promising field for use in alloys. G. de Chalmot by his electric furnace method, prepares silicon iron alloys which contain from 12 to 46% Si, the silicon and iron being chemically combined; there is no free silicon and no free iron present. If more than 50% Si is forced into the iron, the excess crystallizes out in small black crystals. This article deals only with silicides containing 25 to 50% Si.

These silicides seem to consist of a mixture of two distinct compounds between iron and silicon—compound A, which contains 25% Si, and the compound B, which contains 50% Si. These compounds are represented by the formulas Fe_3Si_2 and $FeSi_2$. When iron silicides with about 25 to 28% Si are allowed to cool slowly from the liquid condition, compound A will crystallize out in well-developed crystals which have tetrahedral faces.

Iron silicides are invariably crystalline, and are white or gray in color. Silicide with from 25 to 30% Si takes a fine polish and resembles silver. The higher silicides are darker in color.

Lower grade silicides cast very well with sharp outlines and corners. The silicides of 34 to 40% Si usually form blowholes, besides cracking when the castings cool. All these silicides are but slightly magnetic and those over 30% Si are quite non-magnetic, which shows that there is no free silicon in the alloy. The specific gravity falls with the percentage of silicon.

These alloys conduct electricity well and are very hard. They are brittle, the more so when the percentage of silicon increases. They are unalterable in air or in water. Acids, excepting hydrofluoric, attack them very slightly, especially if they are not pulverized.

The raw materials used by the Willson Aluminum Co. are a good iron ore, river sand and coke. The materials are finely ground, with the exception of the sand which should be coarse; on account of its volatilizing easily and being very light, it is readily carried away by the draft of the furnace.

At the Holcomb Rock works, iron silicide is made in furnaces of about 150 h. p. which are run continuously for one week. On account of their great purity and small bulk these alloys are superior to those made in the blast furnace, but can only be used where the character of the work admits of paying a larger price per unit of silicon.†

Warren,‡ in preparing metallic silicides, mixes the oxide with graphitoidal silicon. The silicon is made by fusing two parts of cryolite and one of sodium silicate at a white heat and introducing an ingot of metallic aluminum which frees the silicon. Reduction occurs tranquilly, while with potassium silicofluoride the reaction is often attended with explosive violence. The silicon is purified from the 10% Al accompanying it by hot hydrochloric acid.

* *Zeitschrift für Krystallographie und Mineralogie*, 31, 399.
† *Engineering and Mining Journal*, Feb. 18, 1899. ‡ *Ibid.*, Feb. 11, 1899.

Molybdic or tungstic acids are mixed with silicon and heated in a specially prepared crucible in an oxyhydrogen blast furnace for 20 minutes. A well-melted ingot containing upward of 20% Si is obtained. If the heat has not been high enough a metallic half-fused mass is produced which contains varying amounts of silicon. Both silicides are highly crystalline and are very hard, some specimens scratching glass. The silicide of tungsten is the hardest. All silicides are difficultly attacked by acids except hydrofluoric, while at red heat all are attacked readily by chlorine.

Hyde, in preparing graphitoidal silicon, reduces finely pulverized white sand with magnesium powder and subsequently fuses the product with cryolite and aluminum. A silicon aluminum alloy is formed, which when treated with hydrochloric acid gives graphitoidal silicon. The yield is 10% by weight of the silica employed.

Steinhart* states the following chief uses of metallic silicon: (1) The addition to iron castings to promote the formation of graphitic carbon. Silicon is desirable in all cases where delicate castings are desired. (2) In many low carbon steel castings silicon is advantageous, as it reduces oxygen compounds without increasing the percentage of carbon; ferrosilicon and ferromanganese, owing to their carbon contents increase the carbon in the steel.

Copper silicide with 10 to 35% Si is now made in ingots weighing from 15 to 18 lb. Its uses are: (1) For the final refining of copper instead of the copper phosphorus compound, accomplished by introducing 0·1% Si into the molten copper in the furnace; (2) for strengthening the ordinary telephone wires of electrolytic copper; (3) for tinting zinc bronzes instead of copper phosphide. The bronzes thus prepared are very elastic and heat resisting and much less silicide is required than phosphide. About one-half the material is saved when silicide is substituted for phosphide. (4) For making alloys by addition to aluminum bronze, whose tensile strength, and hardness range from that of ordinary brass to the hardest phosphor bronze. Silicon copper is now manufactured in the United States by the Cowles Electric Smelting and Aluminum Co., at Lockport, N. Y.

Tellurium.—The preparation of the metal has been studied by Lenher,† who finds that the reduction is most readily accomplished by means of the addition of glucose to an alkaline tellurite solution. Large quantities of tellurium residues are obtained as a by-product in the electrolytic refining of copper. And as the tellurium appears in the condition of alkaline tellurite, it is only necessary to warm the solution and add glucose syrup to precipitate the metal in a finely divided condition. The black mass when dried is fused into a solid lump. Metallic tellurium now sells in Germany at $50 per lb., and this process would be of commercial value should any demand for the metal arise. Unfortunately, the addition of small amounts of tellurium to other metals destroys their valuable properties. Its bad effect on copper is well known. A minute quantity of tellurium introduced into the human system produces a breath of a strong garlic odor. The sp. gr. of the metal is 6·2 and its electrical resistance is

* *Engineering and Mining Journal*, June 17, 1899. † *Journal of American Chemical Society*, April 1, 1899.

enormous. This high resistance is doubtless due to the fact that the metal is extremely crystalline and invisible cracks are likely to be present.

Norris and Fay, of the Massachusetts Institute of Technology, have purified tellurium compounds by means of the basic nitrate and consider that this compound tellurium is best adapted for the preparation of pure tellurium. Tellurium can also be prepared electrolytically, a method of possible commercial importance in the future.

Thorium.—With great care, Hintz has varied the proportions of the rare earths and has studied the effect on the luminosity of the Welsbach mantle. He compares thoria and ceria alone and in mixed proportions. The unit of comparison is liters of gas per Hefner light unit. Pure thoria is 50 and pure ceria is 61. The minimum consumption and greatest efficiency is produced by the mixture of 99% thoria and 1% ceria; this requires but 1·4 liters per hour per Hefner unit. One percent of neo-didymia, lanthana, yttria, or zircona has no effect on the normal mixture of 99% thoria and 1% ceria, while 2% of zircona slightly diminishes the efficiency. Bunte, working along the same line, finds that while pure thoria gives 3 Hefner candle power and ceria, 6 to 7, the Welsbach mixture gives 70 candle power. Saulman,* of Germany, uses thoria in a mantle with a triatomic body, as boron, arsenic, antimony, and chromium; the combustible substance being burnt out.

Many other substances have been tried in the mantle, but so far nothing has been found that gives as great an efficiency as the Welsbach mixture. At the present price of thorium nitrate at $3·33 a pound, the cost is no longer so great a drawback as in previous years.

Swinton† has examined mantles of thoria and ceria and their mixtures under the action of the bombardment of the cathode rays in a Crookes tube; and states as follows:

"One theory to account for the brilliant incandescence of gas mantles requires the assumption of catalytic or some other chemical action between the oxides and the constituents of the Bunsen flame. The author finds, however, that the intense heat produced at the focus of the cathode rays will produce a very high incandescence with the earths at a sufficiently high degree of exhaustion, whether the small amount of residual gas consists of air, oxygen, or hydrogen."

Among other experiments, "a mantle was made up of four patches or sections, as follows: (1) Pure thoria; (2) pure ceria; (3) 50% thoria and 50% ceria; (4) 99% thoria and 1% ceria. With an intensity of rays which gave a brilliant light with sections (1) and (4), sections (2) and (3) gave practically no light, becoming barely red hot; while mixture (4) gave slightly more light than pure ceria and heated up and cooled more rapidly than the latter. It is noteworthy that with a given intensity of bombardment, the thoria should have given light, while the ceria gave practically none; in the gas flame, thoria and ceria give about the same poor amount of light."

Voelker‡ prepares filaments for incandescent lights by using the carbides of uranium or thorium. Two parts by weight of chemically pure nitrate of the

* United States Patent No. 640,326. † *Journal of the Society of Chemical Industry*, Aug. 31, 1899.
‡ United States Patent No. 637,979.

earth metal with one part of cane sugar are dissolved in water, and the solution heated until spontaneous combustion takes place; nitrous fumes are evolved and the mass chars. The mass is cooled and fused in the electric furnace.*

Wyrouboff and Verneuil have contributed largely to the chemistry of both cerium and thorium. Their industrial extraction of thoria is given in the *Comptes Rendus,* CXXVII., 2, 412. They have studied the constitution of the rare earths and find that ignition gives a polymerized substance with thoria as well as with other earths. When thorium oxalate, nitrate, chloride, or hydrate are carefully ignited, the product is a mixture of two polymerides of the oxide, ThO_2. The nitrates have been studied more carefully and appear to separate from nitric acid in succession by the careful addition of increasingly concentrated nitric acid. (See "Cerium.")

A number of new thorium-containing minerals have been reported this year and it is interesting to note the existence of 0·2 to 0·3% of thoria in the Tennessee phosphates, found by C. N. Harding.

Job, Wyrouboff and Verneuil have contributed articles to the *Comptes Rendus* for the year. The precipitate formed by adding potassium carbonate to cerium nitrate was dissolved in excess by Job, and then an excess of hydrogen peroxide was added, which produced three series of oxidation states of cerium: cerous, Ce_2O_3; ceric, CeO_2, and per-ceric, CeO_3. He showed also that the capacity for oxidation by air depends on the nature of the salt dissolved in the alkaline carbonate. Ammoniacal ceric nitrate remains in the ceric state while cerous nitrate goes to ceric and then continues to fix the oxygen of the air until a limit is reached which depends on the dilution of the solution. In his work, he isolates a double salt of the per-carbonate with potassium carbonate from the solution to which hydrogen peroxide had been added. The double per-carbonate is blood-red in color. Wyrouboff and Verneuil found that, in addition to the oxides, CeO, Ce_3O_4, and the peroxide, the oxides $Ce_3O_4 \cdot 3CeO$ and $Ce_3O_4 \cdot CeO$ exist. The former gives uncondensed oxides while the latter is stable only when the CeO is replaced by one of the cerium or yttrium earths. When ceroso-ceric oxide is ignited, it is insoluble in nitric acid, while when mixed with other oxides, it is soluble. Ceroso-ceric oxide combined with lanthana, didymia or yttria forms complex oxides of cerium which polymerize with great ease, giving two isomeric condensed oxides. A knowledge of these facts is important in the separation of cerium from similar elements. Since the rare earths possess such a great tendency to polymerization and to form compounds with themselves, it is difficult to separate them completely. These chemists determine cerium by precipitating the nitric acid solution of the earths by ammonia and peroxide of hydrogen. This solution is evaporated on a water bath and heated until ammonium nitrate begins to come off. The residue is dissolved in nitric acid, evaporated to a syrup, dissolved in boiling water and a small quantity of ammonium sulphate added. A precipitate is formed which contains 90% Ce in a pure state. It is ignited to a white heat. To the filtrate is added 0·05 g. of ammonium persulphate and 1 c.c. of 50% sodium acetate solution. On boiling, the liquid becomes clear and the remainder of the cerium is precipitated. This precipitate should be of a pale

* *Engineering and Mining Journal*, Dec. 16, 1899.

yellow color. It is strongly ignited. Job* determines cerium by treating the salt with lead dioxide and nitric acid in the cold and titrating with standard hydrogen peroxide. The estimation works in the presence of thorium, lanthanum, and didymium. Mengel† is of the opinion that the best method for the separation of cerium is by means of sodium peroxide. The cerium solution is treated with Na_2O_2 in ice-cold water, and the red brown precipitate dried at 120 to 130°, when it turns bright brown to yellow. It is then dissolved in nitric acid and the cerium precipitated as the double ammonium salt in the usual way.

Tschernik has described a new cerium mineral from the Caucasus which carries 34·2% of cerium and is essentially a titanate and zirconate of cerium, although nearly all of the rare earths occur in it. Sixteen substances appear in the analysis. The mineral contains a gas which consists mainly of nitrogen and argon with no helium.

An analysis of the ash of a coal from Tkwibuli gave chiefly calcium sulphate, alumina and silica with about 10% of ceria and lanthana, and over 1% of helium.

Titanium.—In a paper on the universal distribution of titanium,‡ Baskerville shows its widely distributed occurrence. "Roussel found it in basalt; Aleksiejew in certain clays; Holland found it in certain igneous rocks. Dunnington observed its occurrence in the soil of Albermarle County, Va.; later the same writer, with McCaleb, found it in 16 specimens of soils collected from different sections of the United States and subsequently asserted its universal occurrence in the soils of the world. W. A. Noyes found it in a number of Arkansas minerals. Hillebrand has shown its presence in a large number of rocks and minerals collected by the United States Geological Survey. Watt found it in the ashes of wood and coal. Haywood found it in the ashes of both wild and domestic strawberries; Langenbeck speaks frequently of its occurrence in clays. The author found it widely distributed in North Carolina and Ganigon found traces in mineral waters.

"No statement in the literature has been found of the presence of titanium in animal ashes. The ash from incinerated fresh beef, beef bone, human flesh, and bone free from dirt was examined by Baskerville with the following results: Beef bone, 0·0195%; beef flesh, 0·013%; human bone (rib and clavicle), a trace; human flesh (pectoral muscles), 0·0325% titanic oxide.

"J. L. Howe states that titanium is found in abundance in dead bone and muscular tissue. C. E. Wait writes that he has made an examination of a large number of bodies and believes that element occurs in nearly all of them. He has made a large number of estimations of titanium in vegetable bodies, and has examined animal flesh and bone, and human excretory products."

F. J. Pope§ has made an extensive study of the magnetic iron ores from Eastern Ontario with particular attention to the titaniferous magnetites. He finds that the titaniferous magnetites always carry vanadium, nickel and cobalt and it is impossible by magnetic concentration to make more than a partial elimination of titanic oxide from titaniferous magnetites.

* *Comptes Rendus*, Jan. 9, 1899. † *Zeitschrift für Anorganische Chemie*, 19, 67.
‡ *The Journal of the American Chemical Society*, December, 1899.
§ *Transactions of the American Institute of Mining Engineers*, October, 1899.

The Government analyst* has ascertained that practically the whole of the titanium in the recently analyzed Taranki black sands is in the magnetite. This goes to prove that both the electro-magnetic process and the sifting process are useless for removing titanium from this sand. In fact, the sifting or riddling process rather increases the proportion of titanium in the ore saved for smelting, as the magnetite is the finest-grained part of the sand. This applies also to the electro-magnetic process.

Uranium.—Peculiar interest has been attracted to uranium as its minerals contain many of the new elements. The uses of uranium, though limited, are of considerable importance. An alloy is used in the manufacture of a yellow fluorescent glass which arrests chemically active rays. Some of the salts are used in porcelain painting for producing black tints. Uranium carbide has a higher fusing point than platinum and may possess advantages over platinum for use in pyrometers and similar instruments. The carbide makes an excellent alloy with steel and, if it could be obtained cheaply, it might replace nickel and tungsten in the manufacture of high-class steels.†

The occurrence of uranium ores is given in THE MINERAL INDUSTRY, Vol. VII., p. 597. Aloy‡ has prepared uranous hydrate by heating the sulphate with water. The reaction proceeds better in an alkaline solution. The hydrate thus prepared, is soluble in acids, while by heating U_3O_8 in hydrogen, an insoluble mass is formed from which it is difficult to prepare uranous compounds. Uranous oxalate and arsenate were prepared. Friedel and Cumenge have described a new mineral discovered by Poulot in Montrose County, Colo. It is essentially a hydrous potassium uranium vanadate, carrying about 64% U. It is said to contain radio-active elements.

Vanadium.—Patents are now being issued in Germany whereby vanadium is introduced into gas mantles with zircona and thoria, either of the two latter being from 98 to 99%. The metal§ has not received the attention it deserves at the hands of electrometallurgists. It oxidizes slowly in the air, melts at 2,000°C., is neither volatile or fusible when heated to redness in hydrogen, and is not attacked by hydrochloric or sulphuric acids. The addition of vanadium increases the tensile strength and ductility of iron, copper, and aluminum. The present use of the metal is almost exclusively confined to converting aniline into fixed dyes for indelible ink, and to the coloring of glass.

Baskerville reports vanadium, chromium, and titanium in varying amounts in the ash of peat from Hyde Swamps, N. C. F. J. Pope,‖ in an investigation of the magnetic iron ores from Eastern Ontario, observes that vanadium appears to be a constituent of those magnetite ores which are titaniferous. From a number of analyses of the titaniferous magnetites, vanadic and titanic oxides follow a ratio approximating 1 to 28, and from the regularity manifested it is possible that these two elements are related to each other in a manner analogous to that existing in the complex inorganic acids of Gibbs, Marignac, and others.

* *The Engineering and Mining Journal*, Aug. 19, 1899, from the *New Zealand Mines Record.*
† *The Australian Mining Standard*, October, 1898.
‡ *Bulletin Société Chimique de Paris*, July 5, 1899.
§ Cowper-Coles, *Institution of Mining and Metallurgy*, March 15, 1899.
‖ *Transactions of the American Institute of Mining Engineers*, October, 1899.

A deposit of uranyl vanadate has been discovered by A. B. Frenzel on the San Miguel River, San Miguel County, Colo.* The deposit is a mottled green color, silicious, very even in texture, and easily worked. The ore body can supply any demand for vanadic acid. Vanadic acid sells for $9 a pound and as it occurs in small quantities in many of the enormous slag dumps from blast furnaces, a cheap method for its extraction would be welcome. Beside the uses suggested by Cowper-Cowles, vanadic acid is used in dyeing silk and in printing calico.

Victorium.—For several reasons, Crookes† considers the name victorium more appropriate for the new element which he announced as monium in his presidential address before the British Association, September, 1898. The element is obtained by a complicated scheme of fractionation from crude yttria, which has been derived from samarskite, gadolinite, cerite, and other minerals of similar character. The first operation frees it roughly from the earths of the cerite group by utilizing the fact that the double sulphate of potassium and the cerite group are soluble in a saturated solution of potassium sulphate, while the cerium group is difficultly soluble. The crude material is converted into nitrate which is heated until it fuses to a clear liquid; the red fumes set free showing the partial decomposition of the nitrate. The mass is then treated with water and the liquid boiled, precipitating basic nitrate. The basic salt is dissolved in nitric acid and this treatment repeated as long as material is at hand. This treatment separates the victoria and yttria from the bulk of the associated earths. The portions now containing most victorium are fractionated by the crystallization of the oxalates from a solution strongly acidulated with nitric acid. The fractions rich in victoria are now collected and converted into nitrates, a small quantity being again thrown out by the partial decomposition by the heat. The filtrate is again evaporated to throw out more. In this fractionation, victorium collects in the middle fractions, being less decomposed than the cerium group and more than the yttrium group. The fractions rich in victorium are converted into sulphates and repeatedly fractionated from hot potassium sulphate solution; the victorium being finally separated in this way and identified by its phosphorescent spectra. The earth, in its present state, is pale brown and easily soluble in acids. It is less basic than yttria and more basic than most of the terbia earths. From a hot nitric acid solution, victorium oxalate precipitates before yttrium oxalate and after terbium oxalate. The double sulphate is less soluble than the yttrium salt and more soluble than the terbium and cerium group. The nitrate is more easily decomposed than yttrium nitrate. The oxide is assumed to be V_2O_3 and the atomic weight not far below 117.

Yttrium.—Urbain has continued his study of the theory of fractionation, and gives the method employed, stating the characteristic reactions of the yttrium earths and the place of the metals in a rational classification of the elements.‡ A new silicate of yttrium found in Dalecardia, Sweden, has been described by Benedicts. It has been named thalenite. The mineral yields a gas which spectroscopically contains nitrogen and helium.

* *Engineering and Mining Journal*, June 17, 1899. † *Chemical News*, Aug. 4, 1899.
‡ *Revue Générale des Sciences*, Sept. 15, 1899.

SALT.

THE increase in salt production in the United States in 1899 was moderate, the total being 2,780,677 short tons, against 2,625,895 in the previous year. Prices continued firm and slightly higher than in 1898. The main producing States are, as usual, New York, Michigan, Kansas, and those in the Ohio River Valley. Of these four great producing districts Kansas alone showed a marked increase, and Louisiana, which in previous years had shown small but extremely variable production, marketed more salt from the mines of Petite Anse than previously.

PRODUCTION OF SALT IN THE UNITED STATES.
(In barrels of 280 lb.)

Year.	California	Illinois.	Kansas.	Louisiana.	Michigan (c)	Nevada.	New York. (c)	Ohio and W. Virginia.	Utah.	Other States. (c)	Total Barrels.	Total Short Tons.
1895.....	365,280	68,000	1,168,951	159,775	3,929,342	8,428	5,919,155	b 1,704,000	411,784	541,139	14,275,854	1,998,619
1896.....	462,450	55,660	1,624,121	173,117	4,321,144	7,818	6,714,394	b 1,829,771	178,393	343,440	15,707,908	2,199,107
1897.....	484,650	58,764	1,224,980	209,392	5,470,621	5,540	6,172,631	b 1,010,300	657,340	528,705	15,822,923	2,215,209
1898.....	667,298	57,796	1,810,809	120,300	6,728,005	5,325	7,625,809	b 1,044,086	121,429	515,592	18,756,394	2,625,895
1899.....	590,386	54,500	2,172,000	357,143	6,699,060	5,900	7,736,172	1,215,000	448,132	594,555	19,861,948	2,780,677

(a) Amount inspected. (b) Includes the production of Pennsylvania. (c) Includes salt used for the manufacture of alkali.

California.—The production of salt evaporated and from the coast islands showed a falling off, though prices were higher than in the previous year. The 590,386 bbl. produced came mainly from tide water lands in the vicinity of San Francisco, principally in Alameda County.

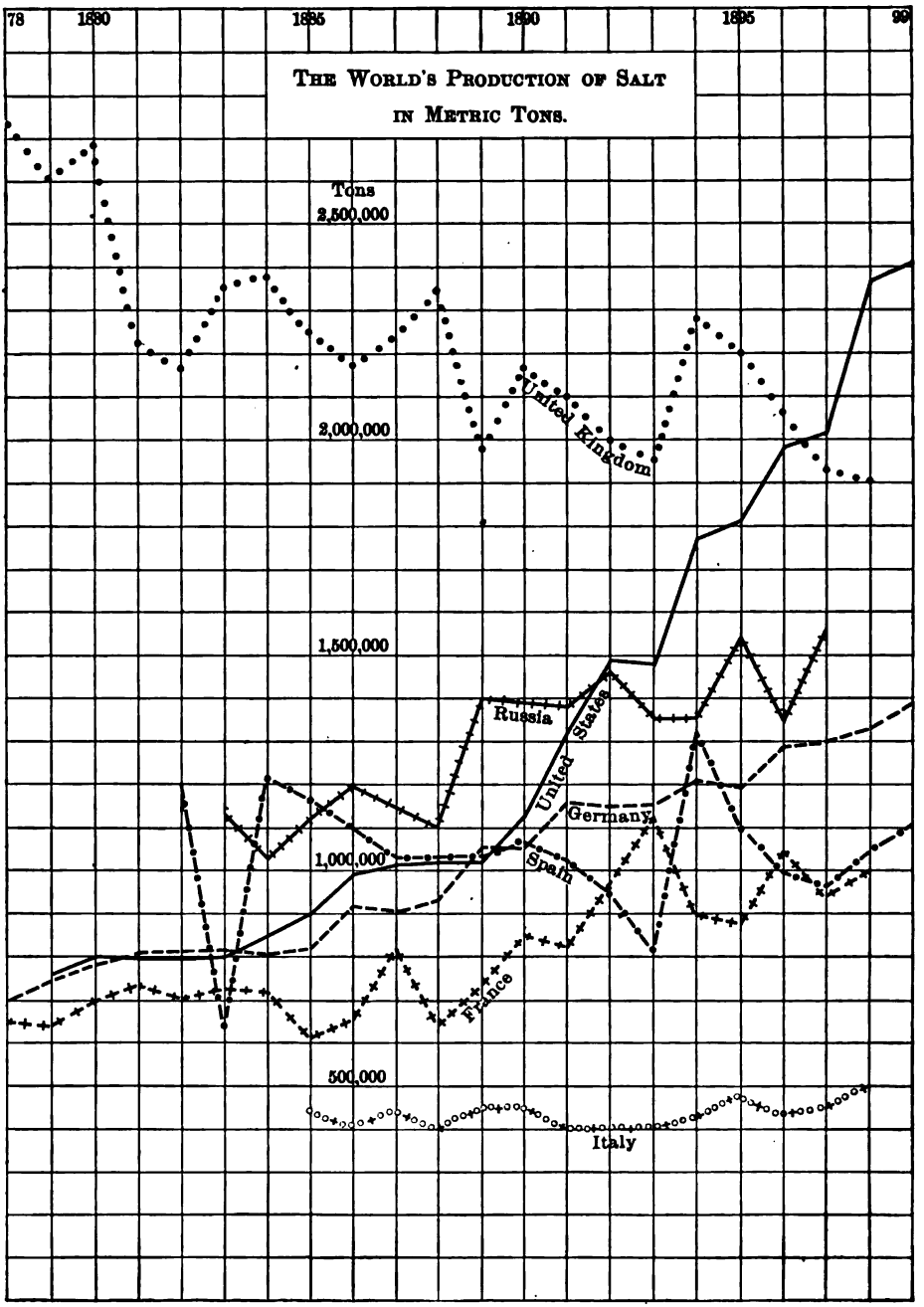
Kansas.—In 1898 Kansas produced 1,810,809 bbl., while in the past year 2,172,000 bbl. were produced and marketed at higher prices than in any previous year. The rock salt of this State, first located by Prof. Hays, in the Permian formation, has been proven at the present time by frequent borings to extend over a considerable area, and to have great thickness in places.

The usual way of extracting salt in the Kansas deposits is by using water as a solvent and pumping the almost saturated brine to the surface with subsequent evaporation of the water by direct application of heat.

Louisiana.—As mentioned previously the production of Louisiana increased. Several new companies have been formed to work the deposits in New Iberia which have been proven to be more extensive than formerly thought. The salt is of excellent quality and there is but slight doubt that the industry will be permanently profitable.

Michigan.—The production of salt in Michigan, including that used in the

manufacture of alkali, fell somewhat below that of 1898, although several new plants had been erected or were in process of erection. It is stated that the cost of salt in brine in Michigan does not exceed 2·5c. per ton, while in New York it reaches as high as 25c.



New York.—One of the most important features of the salt industry during the year was the combination formed early in 1899 under the title of the National Salt Co. The companies included in the consolidation at its commencement were the Le Roy Salt Co., of Le Roy, N. Y.; Kerr Salt Co., Rock Glen, N. Y.; Ithaca Salt Co., Ithaca, N. Y.; Warsaw Salt Co., Warsaw, N. Y.; Silver Lake Salt Co., Perry, N. Y.; Pavilion Salt Co., Pavilion, N. Y.; Pearl Salt Co., Pearl Creek, N. Y.; Bradley Salt Co., Warsaw, N. Y.; Glen Salt Co., Watkins, N. Y.; Hawley Salt Co., Warsaw, N. Y.; W. C. Gouinlock, Warsaw, N. Y.; Cayuga Lake Salt Co., Ludlowville, N. Y., and the Empire State Dairy Salt Co., Warsaw, N. Y. Later, the company is said to have acquired large interests in Michigan, Ohio and West Virginia. In the table showing the production of the State of New York 2,820,392 bbl. are included, valued at $89,559, which never reached the form of crystalline evaporated salt, as the brine itself was pumped and used in the manufacture of various soda products. The production, therefore, of New York exceeded that of Michigan, due deduction having been made for a similar use of salt in brine, by less than 200,000 bbl.

Utah.—The production in Utah increased considerably above that of 1898, partly due to the formation of the Inland Crystal Salt Co., the decrease in accumulated stock, as well as new producers entering the field of production, owing to the firm prices which were maintained. The combination has absorbed the plants of the Inter-Mountain Salt Co., that of H. S. Stowe, those of Williams & Wallace and the properties of the Solar Crystal Salt Co. It absorbed likewise nearly all the vacant land on the Lake shore as far as Promontory which was suitable for salt evaporation. Prices for crude salt ranged from $1·50 to $2·50 per ton in bulk and $3 to $4·50 when sacked, which were material advances on the prices current in 1898. Fine salt sold at $10 f. o. b. cars, practically the same price as in the previous year.

Austria.—The production of salt in Austria, where it is practically a Government monopoly, has been on the increase, though the methods of mining need improvement. The use of dissolving chambers so long practiced, is about to be given up for direct mining operations, the fines produced in the mining operation being dissolved and recovered in the ordinary manner. New methods adopted, it is said, have increased the production more than 30%.

SALT PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD. (*a*)
(In metric tons and dollars.)

Year.	Algeria.		Austria. (*d*)		Canada.		France.		Germany.	
1894.....	17,890	$74,763	311,597	$11,655,048	51,890	$170,667	990,607	$2,287,624	1,257,597	$4,359,578
1895.....	25,758	102,990	278,675	11,472,855	47,515	160,455	871,312	2,406,155	1,212,336	4,340,321
1896.....	19,658	85,121	308,932	9,809,068	39,880	169,693	1,042,614	2,473,568	1,308,319	4,416,952
1897.....	23,222	78,058	321,064	10,357,496	46,584	225,730	948,000	2,239,823	1,306,684	3,834,426
1898.....	21,300	86,000	342,059	10,796,700	51,796	234,520	999,253	2,115,120	1,370,341	3,954,743

Year.	Greece.		Hungary. (*d*)		India.		Italy.	
1894.....	21,310	$332,436	169,282	$7,153,440	1,318,196	(*b*)	30,793	$114,885
1895.....	22,238	335,800	169,395	6,456,480	1,120,832	(*b*)	29,315	110,190
1896.....	23,800	344,280	180,183	6,191,861	1,043,172	$1,097,909	29,274	113,328
1897.....	20,421	308,357	171,711	5,875,788	987,898	975,260	31,526	117,504
1898.....	37,125	579,150	178,551	5,679,584	1,043,828	1,485,702	29,745	120,715

Year.	Russia.		Spain.		United Kingdom.		United States.	
1894.....	1,354,218	$1,723,200	186,121	$228,374	2,271,687	$3,618,145	1,763,133	$7,576,315
1895.....	1,540,194	2,388,743	326,320	952,096	2,208,025	3,548,755	1,818,135	6,472,892
1896.....	1,346,247	3,026,000	521,751	1,158,880	2,054,715	3,333,065	1,995,017	5,328,250
1897.....	1,551,894	2,711,077	508,606	1,159,294	1,933,949	3,104,490	2,009,625	3,898,556
1898.....	(c)	(c)	479,358	1,025,698	1,908,723	3,100,575	2,382,197	4,753,664

(a) From the official reports of the respective countries. For Austria, Hungary, Russia, Spain, and the United States, the production of all kinds of salt is given; Germany, rock salt and common salt; Greece, sea salt; France, rock and sea salt; Algeria, sea and rock salt; Italy, rock and salt from brine; United Kingdom, rock and brine salt; India, salt which is liable to British salt tax only, and does not include salt made in certain native States. (b) Not reported in the official statistics. (c) Statistics not yet published. (d) The high valuation of salt in Austria and Hungary is due to the government monopoly of production and high taxation.

Persia.—The waters of the salt lake Urmi have been examined by R. T. Gunther, M. A. and J. J. Manley, of Magdalen College, Oxford.* Analyses showed a hypothetical approximate composition of the total salts evaporated as sodium chloride, 86·33%; magnesium chloride, 6·66%; magnesium sulphate, 4·21%; calcium sulphate, 0·98%; potassium sulphate, 1·74%. No iodine or bromine was detected, though traces of barium were found in spectroscopic examinations. There was found to be about 14·81% of solid matters. The area of this lake is about 800 sq. miles, and the average depth about 20 ft.

Russia.—The Donetz basin continues to produce largely, the Bakmut district yielding the greater quantity. In the Kars region, on the Russo-Turkish-Persian frontier, the beds of rock salt, notwithstanding their sizes, are but sluggishly worked. The completion of the Kars Railway will undoubtedly stimulate the production at this point. All the salt produced in Siberia, which is a small quantity, is from brine or salt lakes. The price of salt is consequently as high as $57·47 a ton, while in European Russia the price varies according to locality between $3·19 to $12·77 a ton.

Sicily.—The export of salt from Trapani in 1898 was 85,462 tons, against 124,733 tons in the preceding year, 1897.† The falling off was due to the Spanish-American war, which caused a scarcity of tonnage, all steamers being employed in carrying coals and provisions to the West Indies, and also on account of the price of salt in the Spanish salines having been greatly reduced by the high value of gold in Spain.

Spain.—The salt mines of Castellar at Villa Rubia in Santiago, Toledo, are said to contain more than 15,000,000 cu. m. of rock salt of extraordinary purity. The deposit is 17 m. thick, more than 2 km. long and of an average width of 500 m. Sodium sulphate is found underlying this deposit.

A New Method of Manufacture of Table Salt.—The following method recently introduced at the salt wells of Schoenebeck, Germany, is stated to produce a nearly chemically pure, perfectly white salt, in form of very fine granules of crystalline structure. The clarified brine is first concentrated in covered, round pans to effect a complete separation of the least soluble constituents. To remove alkaline sulphates, sufficient chemically pure calcium chloride is added to convert them into the insoluble calcium sulphate, which is separated by filtration along with the other insoluble matter.

The salt solution is then further evaporated in pans provided with mechanical stirrers, and the separated salt dried in centrifugal machines, whereby the last traces of the more soluble contaminating salts are thrown out. The salt, which contains now only 1 to 1·3% of water is further dried and finely sieved.

* Paper read before the Royal Society, June 15, 1899. † Diplomatic and Consular Report, No. 2331, Aug., 1899.

SILICA.

Diatomaceous Earth.—The production of diatomaceous earth in the United States for 1899 fell to 1,136 short tons ($8,216) from 1,392 short tons ($11,002) in 1898. Notwithstanding the small production there are many large deposits known in the United States. In Maryland, near Coke Creek, a large deposit is worked by the New York Silicite Co. Immense beds are known to exist in Florida and Virginia. In New York State, at White Lake, Herkimer County, a deposit on the bottom of the lake itself has been worked. In Nevada, while several deposits have been known to exist, those worked by the Electro-Silicon Co. have been the only productive ones.

Pumice.—The production was still confined to Utah. There was a slight increase in the production of the Chicago Pumice Co., which operates these deposits; 340 short tons, valued nominally at $1,700, was produced in 1899.

Quartz and Flint.—While flints exist in large quantities in chalk beds in Texas and Florida, those imported from France can be landed here so cheaply that it is impossible to mine these deposits economically owing to their remoteness. Flint is used for grinding clay and latterly for grinding, not only such materials as phosphate rock, gypsum and cement, but gold and silver ores. In this country quartz, and in places broken tiles, have been used instead of flints in acid towers.

Quartz is mined in various parts of the country. The production in 1899 as returned by ten producers, was 24,305 short tons, valued at $56,481, against 20,593 tons, valued at $47,333 in the previous year. While no doubt these figures are incomplete it is believed that they represent the greater part of the domestic production. It will be noted that these statistics for 1898 do not agree with those published in THE MINERAL INDUSTRY, Vol. VII., as certain amounts of other forms of silica have been deducted, which were reported incorrectly by producers under the term of quartz crystal.

Novaculite, an uncrystallized quartzite, containing 98% SiO_2 has been successfully used in places for the manufacture of roads and streets. According to Prof. J. B. Johnson, of Washington University, the roads have proven to be hard and indestructible, either from wear or disintegration and decay. To construct a road properly an ordinary pavement foundation of limestone rock should be made. This should be covered with 5 or 6 in. of coarse novaculite with its natural bond. After rolling this hard it should be covered with 2 in. of a finer or screened novaculite and bond and rolled thoroughly into a solid mass.

Tripoli.—The production of tripoli in 1899 increased from 1,865 short tons,

valued at $3,367 in 1898 to 2,102 short tons, valued at $4,729 in 1899. It will be noticed that the average value increased somewhat. The production came almost entirely from Newton County, Mo.

A small amount of tripoli is produced in Nova Scotia. The Bass River Silica Co. worked actively during the year, and the Premier Tripoli Co., of St. Anne, Cape Breton, erected machinery and kilns. The production amounted to 600 tons from these two sources.

Whetstones.—The statistics of the production of whetstones, scythestones and oilstones in the United States are summarized in the following table:

PRODUCTION OF WHETSTONES AND SCYTHESTONES IN THE UNITED STATES.

Kind of Stone.	1896.		1897.		1898.		1899.	
	(a) Quantity.	Value.	(a) Quantity.	Value.	(a) Quantity.	Value.	(a) Quantity.	Value.
Hindostan (finished product)	275,000	$10,000	250,000	$5,625	275,000	$8,225	325,000	$9,750
Indiana sandstone (finished product)			200,000	4,000	100,000	2,000	91,000	1,820
Arkansas stone ("rough rock")	3340,000	50,000	100,000	4,000	40,000	1,600	60,000	2,400
Washita stone ("rough rock")			350,000	4,375	350,000	4,575	350,000	4,575
Miscellaneous oilstones (c)	.....	.....	1,677	167	.....	2,500	.....	4,000
New Hampshire (Indian Pond, etc.)	15,000	35,000	12,000	33,000	13,000	35,750	13,000	35,750
Vermont (Lamoille, etc.)	907	3,176	4,200	14,700	5,000	17,500	5,000	17,500
Ohio and Michigan grit	4,129	7,025	18,114	31,362	15,696	36,100	.....	39,425
Total	.....	$105,201	.....	$97,229	.....	$107,990	.....	$115,220

(a) Quantities given for Hindostan, Indiana, Arkansas and Washita stones and miscellaneous oilstones are pounds; for New Hampshire and Vermont scythestones, and Ohio and Michigan grit the quantities are number of gross. (b) Finished products in 1896. (c) Including Queer Creek, Chocolate, etc., estimated.

It will be observed from the above table that there is very little change in the industry. The amount of Arkansas rough rock quarried increased 20,000 tons above that of 1898, and the production suffered from overproduction in previous years. Prices remained substantially the same as in previous years, though there was a slight tendency toward increased values.

Silicon.—The Fabrik Electrometallurgischen Producke zu Frankfurt-Bockenheim has evolved a secret process of manufacturing silicon rods. The method is said to differ from that of Chalmot, described in THE MINERAL INDUSTRY, Vol. VII. The same works also offers copper in bars weighing 8 to 9 kg. containing 10, 15, 25 and 30 to 35% silicon. This alloy is used for the reduction of suboxide of copper in metallic copper, replacing copper phosphide. Its use increases the tensile strength of copper wire, but decreases its conductivity.

Preparation of Graphitoidal Silicon.—Frederick S. Hyde* has described a new method. A mixture of four parts of finely powdered white sand and one part of powdered magnesium is gently ignited in a test tube. The lower part of the tube becomes incandescent and the incandescence spreads gradually to the rest. After cooling, the contents of the tube are ground and mixed with powdered cryolite, and placed in a crucible with a piece of aluminum in the bottom. This is heated in a furnace until the aluminum melts, when the mass is well stirred with an iron rod. When tongues of bluish flame are evolved the heating is stopped. The metallic mass is then treated with warm hydrochloric acid, which dissolves the aluminum and leaves the silicon in the form of spangles. These are washed first with water, then with alcohol, and finally dried on the filter.

* *Journal of the American Chemical Society.* XX., p. 668.

Sand for Rolling Mill Reheating Furnace Bottoms.

By Charles Ferry.

Under this title I refer to the varieties of sand which are peculiarly adapted to the construction of the bottoms of reheater furnaces in rolling mills. The characteristic feature of a good bottom sand is, of course, its ability to "set," forming a hard, durable furnace bottom. It is not safe to use sands which contain clay or are of a "loamy" nature, since by reason of the plastic, sticky nature of the clay constituent of such sand the tendency is for the iron to "take up" the bottom sand, and when a bar is rolled the sand is imbedded in it.

After a good sand is spread carefully over the bottom of the furnace it should become hard on top within 15 minutes after the furnace is again closed. While it is essential that the bottom sand should set thus quickly, it must not be of such a fusible nature as to become seriously affected by the heat of the furnace, since if it is partially fused, or even approaches that state, it assumes the same sticky property which clay imparts to it, and that invariably leads to trouble. There are but few deposits of sand which can be used for the purpose to good advantage. It is hard to realize, until a practical test of bottom sand is made, in how many slight, yet important, particulars this material may fail to do its work satisfactorily. A furnace bottom is not expected to last more than one heat without renewing. After several heats it becomes thoroughly fused and runs off as a slag. In this way the furnace bottom gradually wastes away and needs to be kept built up. The property of "setting" in the furnace, that sand must have, is affected by the very slightest tendency of the mass to fuse.

For a long time it seemed to be impossible for the chemist to determine the true comparative value of different grades of bottom sand. I regard this, however, as due to the general belief that the "setting" property is dependent solely upon the chemical composition of the material, but in practice we find that the size of the grain of the sand determines this to a large extent, the coarse sands requiring more heat than the finer ones, when the chemical composition of both is identical. This subject has been the source of a great deal of annoyance to many iron and steel chemists. I have in mind one who had investigated the chemical composition of many of the best bottom sands of the country with a view of finding from them a definite composition which he could rely upon as being characteristic of an efficient bottom sand. The result of his work led him to believe that the percentage of total silica was the all-important feature and determined the adaptability of sand to furnace bottoms.

His theory served him well till he began to determine its applicability to fire sands. According to chemical analysis many of them carried the proper quantity of silica, the only other constituents being "combined water" and alumina, at least practically so. These sands were composed of pure quartz, mixed with a little clay, but were not clean and sharp like the best grades of bottom sands.

If the theory that "any sand carrying the fixed and standard percentage of total SiO_2 will set successfully," had been true, the various specimens of fire sand should have set, but in practice they showed no tendency to do so. But besides their inability to set, there was another feature in connection with them which would have condemned them. The trace of clay present was taken up by the hot iron which, when rolled into bars, showed lumps of sand on the surface and

inside. The sands which were tested had the following composition: *A, good,* SiO_2, 91·55%; $Al_2O_3+Fe_2O_3$, 7·40; CaO, 0·51; MgO, 0·11; water of constitution, 0·61. *B, poor,* SiO_2, 91·47; $Al_2O_3+Fe_2O_3$, 4·88; CaO, 0·21; water of constitution, 3·61. The first was the sand in regular use in the rolling mill where these tests were made, and at that time was the best for the purpose available in that locality. The second was that which approached most closely the first with respect to silica content. The second was tried at different times under the most favorable conditions, and was left in the furnace for 20 minutes, but without avail, for it would not manifest even the slightest tendency to "crust over." While under the same identical conditions the first would "set" hard in 15 minutes.

The most important difference between the two sands, as shown by the analyses, was in their percentages of alumina and ferric oxide and water of constitution. In *A* there was 7·4% alumina and iron together, with hardly more than a trace of water, while in *B* there was only a little more than half the quantity of alumina and iron, and more water. Of course the analyses are defective in not reporting Al_2O_3 and Fe_2O_3 separately, since the effect of these two substances upon refractoriness of the clay is different. Iron combines with silica to form fusible silicates, while alumina unites with silica to form kaolinite, which is an exceedingly refractory substance.

Let us take, for example, two bottom sands which are alike in every other respect except size of grain. If tested in a rolling-mill furnace they would not "set" the same. The coarser-grained sand would require more heat than the finer. Again, if they were of the same size grain and identical in every other physical condition, and the percentage of silica and total fluxing bases were equal, but individually the percentage of some of these bases differed, as, for instance, magnesia and fixed alkalies, the two samples would not set exactly alike.

If a chemical analysis is to be a guide in the choice of bottom sand, the results should be interpreted and arranged as follows: (1) Silica (combined), alumina, quartz, *i. e.*, refractory constituents; (2) water (combined), *i. e.*, plastic property; (3) lime, magnesia, ferric oxide, fixed alkalies, *i.e.*, fluxes. Besides these chemical considerations, there is to be observed the size of the grain.

In the first place, we have to avoid sands which have any plasticity. They should be short, sharp, and clean, which is always characteristic of the sands containing no clay. The presence of combined water, say more than 0·1%, usually indicates the presence of kaolinite; therefore to choose a good bottom sand it is essential that the water of constitution should be low. The fluxes, lime, magnesia, iron, and the fixed alkalies are the substances which play the most important part in the "setting" of the bottom sand. In the aggregate these bases probably amount to 4 or 5% in many superior bottom sands, and, as the fluxing powers of these substances with silica are different, to bring about the same results the same bottom sand would require possibly twice as much of one as of the other. To illustrate, let us suppose that we have two specimens of sand, both of which give good practical results and are the same in chemical composition, except with respect to the principal fluxing base. If the latter be potash or soda, much less will be required to accomplish the same effect than if it were iron; for the alkali is a more powerful flux than the iron.

SLATE.

THE production of slate in the United States in 1899 showed a large increase over the previous year, and although the export trade fell off to some extent, good prices were obtained for all varieties, and the year may be considered very favorable to the industry. The stocks of roofing slate at the quarries at the end of 1899 were materially less than at the corresponding period for several years past. The details of production are given in the subjoined table.

PRODUCTION OF SLATE IN THE UNITED STATES. (*a*)

States.	1898.				1899.			
	Roofing Slate. Squares.	Value.	Mfrs. Value.	Total Value.	Roofing Slate. Squares.	Value.	Mfrs. Value.	Total Value.
California	400	$2,800	*Nil.*	$2,800	810	$5,900	*Nil.*	$5,900
Georgia	3,845	14,419	*Nil.*	14,419	2,500	12,500	*Nil.*	12,500
Maine	26,389	112,945	$66,484	179,429	23,412	99,033	$29,364	128,397
Md., Pa. (Peach Bottom)	23,335	101,778	1,771	103,549	27,959	127,129	*Nil.*	127,129
Pa. (Chapman's Quarries)	25,753	69,448	*Nil.*	69,448	25,300	73,876	*Nil.*	73,876
Pennsylva. (other districts)	689,610	1,810,096	388,229	2,198,325	648,432	1,848,031	442,668	2,290,699
Vermont and New York	*c* 282,752	537,229	62,812	600,041	316,395	711,889	56,642	768,531
Virginia	88,548	125,281	8,560	133,841	58,060	175,630	8,160	183,790
Other States	*e* 1,000	*e* 4,000	*e* 1,000	*e* 5,000	*e* 500	*e* 2,000	*e* 3,600	*e* 5,600
Totals	1,041,633	$2,777,996	$528,856	$3,306,852	$1,098,374	$3,055,988	$540,434	$3,596,422

(*a*) Not including slate pigment. (*b*) Statistics of the California State Mineralogist. (*c*) The figures differ from those given in Vol. VII., which included a considerable amount of stocks. These have been deducted in the above table, so that the figures for both 1898 and 1899 represent the actual output. (*e*) Estimated.

NOTE.—The production of slate pigment, including Baraga graphite and various kinds of mineral black, in 1899 was 4,911 short tons, valued, after grinding, at $50,338, against 7,886 ($70,671) in 1898.

Export Trade.—The export of slate from the United States in 1899 was: Roofing slate, 252,436 squares, valued at $1,114,018; slate manufactures, $127,733; total value, $1,241,751. Compared with 1898 there was a decrease in exports of 54,271 squares, valued at $162,811, and slate manufactures, $8,572, a total decrease of $171,343. These valuations include the cost of the slate, rail and ocean freights, insurance, etc. The lessening of the exports was due to the increased production of the Welsh quarries and to the higher ocean freight rates. The greater part of the roofing slate exported from the United States was shipped to the United Kingdom, while school slates and mill stock were shipped to all countries, and manufactured slate was exported to Canada and to European countries. General freight rates to London varied from 13 to 20s., with occasional lower quotations.

Slate Manufactures.—The production of mill stock in 1899 was less than in 1898, due to the increased cost of rough slate. The consumption of slate for use in the manufacture of switch boards by the electrical companies has been on the increase. At the opening of the year mantles were quoted at from 50 to 80c. per superficial foot. Later in the year, however, prices ranged higher. The prices of mill stock were maintained with fair success by the manufacturers in the Northumberland and Lehigh regions. Although efforts were made

at various times during the year to form an agreement among the quarrymen, outside of the Bangor Association nothing seems to have been accomplished. The output of school slates in Pennsylvania is now under the control of the National School Slate Co., which was organized early in the year and assumed control of the three large factories; the Hyatt, at Bangor; the American Slate Works and the works of Edward J. Krause at Slatington. The shipments of school slates during 1899 through Slatington and Walnutport, Pa., were 18,423 cases, against 19,935 in 1898. The value of a case of school slates is about $3. There was a lessened demand for blackboards in 1899. Early in the year contracts were made on a basis of 10, 11 and 12c. per sq. ft., depending on the size of the boards. Later these prices advanced to 12, 13 and 14c., and were maintained by the manufacturers. Marbleized slate is manufactured mainly in the Vermont-New York district. Prices vary from 50 to 80c. per superficial foot, according to quantity and workmanship. This product meets with the severe competition of the cheaper marbles of Vermont. Billiard slabs, of which a quantity was exported during the year, ranged in price from 18 to 23c. per superficial foot. The use of slab slate for lining acid tanks is becoming more general. It is also used for washtubs and in building construction. At times in 1899 slab slate sold as low as 7c. per superficial foot, although the average price was much higher. The demand for powdered slate increased, especially for the manufacture of pigments and cement. The prices vary from $9·25 to $11·25 per short ton, according to color and quantity. Cement made from slate flour was worth from 4 to 7c. per lb., depending on quantity and package. The attempts made in Pennsylvania to use powdered slate in the manufacture of brick have not been commercially successful as yet. Slate pencils are now made from slate dust by the hydraulic pressure process of the D. M. Stewart Manufacturing Co., of Chattanooga, Tenn. The dust is mixed with some adherent and pressed, forming pencils that are free from grit. This company is credited with an annual production of 25,000,000 pencils, a considerable part being shipped abroad. The production of slate pigment was 4,911 short tons, valued after grinding at $50,388 in 1899. Under this heading we include all mineral blacks made from slate, graphitic shale, etc., as well as the so-called Baraga graphite. The production of slate pigment proper is made chiefly in the Vermont-New York region, where various colors, including green, red, olive, slate and dark brown are prepared. The ordinary ground pigment sells at $7·50 to $8·75 per short ton, depending on quality.

Roofing Slate.—The prices of roofing slate per square in 1899 f. o. b. at quarry were as follows: Maine, Monson's No. 1, $4·80 to $7·20; Maryland No. 1, $4·60 to $5·35; Pennsylvania, Bangor, $3·25 to $4·50; Bangor ribbon, $2·90 to $3·50; Albion or Jackson Bangor, $3·10 to $4; Lehigh, $3 to $3·50; Vermont-New York slates were: No. 1 sea green, $2·20 to $3·50; unfading green, $3·25 to $4·25; red roofing slate, $8·25 to $10.

Maine.—This State ranks third in the production of slate, all of which comes from Piscataquis County. The slate is very superior for roofing purposes and there is also a great demand for mill stock. In 1898 the amount used for roofing was 60% of the total output. The Piscataquis County slate has a

lustrous black color which does not fade upon exposure and a grain of unusual fineness. The component minerals are quartz, biotite, magnetite, pyrite with about 3% of organic matter and no carbonates. The slate is stronger than the average building stone, having a crushing test which exceeds 29,270 lb. per sq. in. on the bed—perpendicular to the cleavage—and 16,750 lb. parallel to the cleavage. The principal quarries are at Monson and Brownville. Many of the quarries are very deep, notably those at Monson, where the workings are 400 ft. below the surface.

Maryland.—In the Peach Bottom district, which is a continuation of the Pennsylvania beds, the production of roofing slate increased somewhat and the prices have advanced.

Pennsylvania.—The production of roofing slate increased slightly, due to the resumption of work at several of the larger quarries that were shut down in 1898. Some new quarries also have been opened. Perhaps the most important feature of the industry in this section is the tendency toward the enlargement of the holdings controlled by the railroads. The American Slate Co. was formed with a capitalization of $1,000,000 to govern the business of the larger quarries in Northampton County. The president of the Lehigh & New England Railroad, Mr. W. J. Turner, is at the head of the concern. The properties that have been acquired are, the Ætna, Bangor Southern, the Old Delabole, the Chrome (formerly the North Bangor), the Danielsville and Gem quarries at Danielsville, the Star & Standard at East Bangor and the American Bangor. There has been a partial resuscitation of the Bangor Roofing Slate Manufacturers' Association, which was dissolved in 1898.

Vermont-New York.—Practically all the slate produced in Vermont at the present time comes from Rutland County, on the extreme western border. The stone is of the Lower Cambrian age and the territory in which commercially valuable slate is found has a north and south extension of 25 miles, with an average width of 1·5 miles. The slate is classified according to the prevailing color, as follows: Sea green, unfading green, purple, purple variegated and mottled. The sea green variety is the cheapest, as it is in greater predominance and is easy to work. The color changes upon exposure to the weather, but apart from this fault it is a strong and durable stone. The unfading green is somewhat deeper in color than the sea green and commands the highest price. It is in great demand for roofing, especially in England. A syndicate that was formed in 1897 controls the output of this variety, most of which is exported. The use of channeling machines in slate quarries was first instituted in 1897, and has proved very successful, as it has lessened the cost of production and has diminished the percentage of waste. The principal use for Vermont slate is for roofing purposes, and for several years past, the amount used for this purpose has been over 90%. In the red slate district of New York the National Slate Co., a new concern, carried on a close competition for the trade and caused prices to drop for a time. Later on, arrangements were made by all the leading producers with the H. H. Mathews Slate Co. for the sale of this slate at a stipulated price. This concern took an option on the properties for an indefinite period, with the privilege of purchasing.

SODA AND SODIUM.

THE production of soda, caustic, carbonate and bicarbonate, reduced to a common basis of 58% ash, in the United States in 1899 was 387,020 metric tons, against 340,622 metric tons in 1898. Following the proportionate production in the previous year the amount produced as soda ash was 280,000 metric tons, as bicarbonate of soda 39,500 metric tons, and as caustic soda 53,000 metric tons. The companies mentioned in THE MINERAL INDUSTRY, Vol. VII., are still in operation and some of them enlarged their capacity materially, to meet the increasing demand.

During the year, ammonia soda has recovered from the depression of the two preceding years in this trade, though disturbance is feared in the future, as several new producers have appeared in the field. The demand for Leblanc soda is still decreasing. Shipments to the United States from England have not been extensive, though prices ruled slightly higher than ammonia soda. The market for bicarbonate of soda has been exceptionally good, but prices, owing to increased competition, have been on the decline.

This falling off in realized values is due to the establishment of new works at Laramie, Wyo., increased facilities for production at the Pennsylvania Salt Co.'s works, the utilization of the previously wasted carbon dioxide from various breweries in Milwaukee, and the close competition between producers. These elements probably will cause a still greater decline.

The demand for crystal carbonate, which is now manufactured by only one firm in the United States, is quite likely to lead to the erection of competitive establishments. Sal soda has been in fair demand, though prices have fallen off. New plants in Chicago, New York, Milwaukee and Pittsburg have been erected to manufacture sal soda from soda ash. These are able to supply local demand at a profit, owing to the difference in freight between soda ash and sal soda, as in the first case no water of crystallization is carried. There has been a greater consumption of bicarbonate of soda by manufacturers of carbonated waters. With the prevailing prices and contracts made for soda ash in bulk, caustic produced by the electrolytic process will have to be sold in very close competition, and the manufacturers by this method have still to expect the increasing demand for soda ash and displacement of caustic, as large soap

manufacturers have been erecting plants for the production of soda from ammonia ash.

It is not improbable that the natural soda fields and springs of the Western United States will be in time an important factor in regulating and probably depressing the market. It is stated that soda ash in Wyoming can be produced at a figure not to exceed $7 per ton, while under favorable conditions in the vicinity of salt wells the ammonia process is said to cost $9 per ton of ash.

An interesting feature is the entrance of glass manufacturers into the soda ash industry. The Columbia Chemical Co. has been formed by people identified with the Pittsburg Plate Glass Co., and works are to be erected at Barberton, near Akron, O., for the manufacture of soda ash.

IMPORTS OF SODA PRODUCTS INTO THE UNITED STATES.

	1895.		1896.		1897.		1898.		1899.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
		$		$		$		$		$
Soda, nitrate..	265,388,480	3,778,360	258,728,960	3,566,744	212,721,600	2,810,187	330,386,560	2,296,240	328,142,080	3,486,313
Soda, bicarb...	8,862,028	123,425	4,343,786	60,878	965,669	13,962	280,988	5,794	162,396	5,219
Soda, caustic..	72,019,114	1,211,090	47,827,247	859,809	57,742,392	988,318	24,961,873	354,270	13,863,539	186,006
Soda ash (a)...	307,026,094	2,221,612	191,780,537	1,444,417	154,656,060	1,122,867	73,084,707	447,119	56,053,837	425,205
Oth'r soda salts	1,081,887	31,581	1,279,285	40,089	7,927,145	100,999	23,354,295	256,958	26,340,840	355,502
Lime, chloride.	104,317,968	1,628,877	99,491,335	1,441,600	109,176,451	1,460,799	108,462,828	1,229,978	123,583,061	1,208,344

(a) Including sal soda.

Market Conditions.—The demand for soda ash has improved materially for the domestic article to the detriment of foreign makers. The average monthly price of domestic 58% ash per 100 lb. in lots of 50 tons or more f. o. b. works, is shown in the following table:

Year.	Jan.	Feb.	Mch.	April.	May.	June.	July.	Aug.	Sept	Oct.	Nov.	Dec.	Average.
1897.....	72·50	72·50	63·12	63·13	60·00	60·00	60·00	60·00	60·00	65·00	66·25	66·25	64·06
1898.....	53·75	51·25	53·75	50·50	57·50	57·50	51·50	52·50	50·00	59·00	57·50	57·50	54·35
1899.....	57·50	59·20	62·50	66·50	67·50	63·75	68·75	70·00	71.75	80·00	82·50	83·50	69·45

Caustic soda in the first quarter of 1899 showed an increasing demand, and the price for domestic high test advanced from $1·40 to $1·70 per 100 lb. for immediate shipment, while future orders were booked on the basis of $1·35 to $1·40. The foreign article at the same time brought $1·60 to $2 per 100 lb. in New York for prompt delivery, and $1·60 to $1·70 for shipment. The second quarter opened with inquiries for 1900 delivery, which were taken by domestic makers at $1·40 to $1·425 per 100 lb. f. o. b. works, and by importers on the basis of $1·60 in New York. At the same time domestic sold for immediate shipment as low as $1·375, but makers were not anxious to take long time contracts on this basis. In the third quarter domestic soda ash became scarce for prompt shipment, and makers quoted up to $1·65 f. o. b. works, while they booked 1900 contracts at $1·45 to $1·55. The foreign article remained at $1·60 to $1·70. In the last quarter the continued scarcity in goods for prompt delivery caused prices to soar, domestic selling up to $1·90 f. o. b. works, and the foreign at $1·95. Future shipments of domestic were booked at $1·80 to $1·85, and some 1900 and 1901 contracts were reported taken at

$1·60 to $1·80 f. o. b. works. The foreign was also firmly held, and closed the year at nominally $1·85 to $1·95 in New York.

Domestic sal soda sold in January at 50c. per 100 lb. f. o. b. works, and the English at 62 to 62·5c. in New York. Up to July there was but little change in prices. In July prices rose to 70c. for domestic sal soda, and 62·5 to 70c. for foreign. In August the increased demand for domestic raised the price to 90c., while importers quoted 67·5 to 70c. This competition lowered prices in September from 60 to 65c. for domestic, and 65 to 67·5c. for foreign. In October importers advanced their price from 67·5 to 72·5c., notwithstanding the fact that the domestic makers in competition had slaughtered their prices to 50 to 55c. f. o. b. works. In November, however, the domestic article strengthened from 60 to 70c., while the foreign sold at 67·5c. The year closed with a stronger tone in prices.

Domestic bicarbonate of soda (ordinary grades) sold at $1·125 to $1·375 per 100 lb. f. o. b. works, and the extra grades at $3·25 to $3·50, less the usual discounts. Foreign bicarbonate of soda brought $2·125 to $2·375 per 100 lb. in New York.

English prime bleaching powder sold in January at $1·50 to $1·65 per 100 lb., and other English and Continental makes at $1·40 to $1·65, according to quality. Spot supplies of bleaching powder began to grow scarce in February, and prices advanced. In March prime Liverpool makes for immediate shipment brought up to $1·85, and other grades as high as $1·65. In July prices dropped to $1·40 for prime Liverpool, and $1·25 for other makes. Therefore prime goods became scarce, and in November when the agents of the United Alkali Co. reported that their stocks were depleted, prime Liverpool bleaching powder brought $3 to $3·50 on spot, and $2·25 and up for arrivals, while other English makes and Continental goods ruled at $1·80 to $2·25, according to quality and time of shipment. In December prices were easier at $2·375 to $2·50 for prime Liverpool, and $2·15 to $2·25 for other sorts.

The United Alkali Co., Ltd., paid the usual dividends on the preferred shares, but nothing on ordinary ones. Earnings decreased on account of high prices of raw materials purchased to complete existing contracts. Considerable capital was diverted to rehabilitation of old works, and the liabilities were increased by £67,689 to a total of £305,257.

Natural Sodium Carbonate.—The production reduced to a basis of 58% ash of natural soda in the United States in 1899, came from California, Nevada and Wyoming, and amounted to 10,500 short tons ($147,000) as against 7,150 tons ($94,650) in 1898. The producers were the Inyo Development Co., of California and Nevada; Griswold & Epperson, of Wadsworth, Nev.; Occidental Alkali Co., of Hawthorne, Nev.; the American Soda, Glass & Chemical Co., of Laramie, Wyo., and the Green River Soda Works, at Green River, Wyo. The Chicago firm of Stephen Padden & Co. did not produce in 1899.

Sodium Nitrate.—Chile Saltpeter.—For potassium nitrate or niter, see under "Potash," page 479.

In 1899 the production of sodium nitrate in Chile showed an improvement over 1898, and prices also were better. The total exports amounted to approx-

imately 1,360,000 long tons, against 1,254,000 in 1898, and 1,060,000 in 1897. The *oficinas* claim that production reached the maximum in 1899. Repeated efforts have been made to resuscitate the old combinations, but apparently without success; nevertheless an understanding between the producers is said to exist, and prices are thereby kept from general demoralization.

Manufacture of Sodium Nitrite.—As descriptions of the details of manufacture of sodium nitrite from Chile saltpeter are infrequently found in technical literature, the following description by N. A. Darbon will prove of interest,* as this substance is now very widely used in the dyeing industry. Commercial saltpeter is melted in large cast-iron vessels. The operation causes the evaporation of the hygroscopic water and the decomposition of a part of the iodides and iodates which accompany the saltpeter. A temperature of 310°C. is necessary, but before adding the lead to effect the decomposition the heat is raised to about 420°. About 280 parts of lead to 100 of saltpeter are required. The lead should be free from small quantities of other metals, particularly of antimony. When all the melted lead has been added, the melted mass is ladled out and run in fine threads into cold water with constant stirring. A certain amount of caustic soda which dissolves some of the lead oxide, is decomposed by adding nitric acid up to the neutral point. In this manner saltpeter is again formed, while lead oxide is precipitated in a state of insoluble hydroxide. We have thus an aqueous solution of sodium nitrite, undecomposed saltpeter, caustic soda holding lead oxide in solution, and soluble impurities. Lead oxide and peroxide and a certain amount of metallic lead are left in an insoluble residue. The solution is then diluted to 68°B. and neutralized with nitric acid, dilute sulphuric acid, or again with solution of lead nitrate. Acid is added so long as a precipitate is formed. Although many treatises on chemistry state that sodium nitrite has alkaline reaction, as a matter of fact it is absolutely neutral when pure. The neutralized solution is separated by siphoning or decanting from the insoluble residue and concentrated in cast-iron basins to 42 to 45°B.

The insoluble residue is thrown on a filter, washed with warm water, and the wash-waters added to the principal solution. The concentrated solutions are added together in cast-iron vats and left to crystallize; if the crystals thus obtained are not pure they must be re-dissolved and re-crystallized. The pure crystals are separated in a centrifugal machine, washed, dried and packed. The residuary lead oxide may be either melted and cast as it is, reduced to the metallic state, or transformed into minium; it can also be used for the preparation of white lead, of nitrate, or other plumbic compounds.

Metallic Sodium is manufactured in the United States by the Moore Process Co., West Paris, Me., and by the Niagara Electro Chemical Co., Niagara Falls, but statistics of the production are not available. Abroad the metal is produced by the British Aluminium Co., Ltd., at Oldbury, England, and at several places in Germany, all using the Castner process, which electrolyzes fused caustic soda. The Acker process, which is controlled by the Acker Process Co., of Niagara Falls, is not in use at the present time. This process forms electrolytically an alloy of lead and sodium or potassium and obtains the latter by distillation.

* *Chemiker Zeitung*, 1899, p. 173.

The Manufacture of Soda.

Throughout the entire period of the ascendency of the Leblanc process, America remained wholly dependent on England for its supply of soda. So far as my information goes, the first attempt to produce it in the United States was made about 1880 by Mr. Hermann Frasch, near Bay City, Mich., with a modification of the ammonia process.

This effort speedily failed and it was not till 1884, after nearly three years of preliminary work, that the first soda was successfully produced in this country, at Syracuse, by the Solvay Process Co., working under the patents of Ernest Solvay, the discoverer of the process which bears his name. Since then this company, which has the remarkable advantage of the right of reciprocal visitation and adoption of all the improvements discovered in the other factories working under the auspices of Solvay & Cie in different countries of Europe, has greatly extended its Syracuse plant and has built another at Detroit.

Since 1884 various other concerns have tried the manufacture of soda by the Solvay process, the fundamental patents of which have fallen into the public domain, but most of them failed to surmount the initial difficulties. Even the attempt at Cleveland was abandoned when prices fell to the low level of recent years, although it was backed by magnates of the Standard Oil Co., who were able to provide a market free from competition for the entire output of their plant, so that the failure can only be ascribed to the inherent difficulties of this branch of chemical manufacture, as they lacked neither market, financial resources, nor good chemists. The factories at Brandenburg, Ky., and Trenton, Mich., have been dismantled, and that at Saltville, Va., was partially idle during 1899.

The establishment at Wyandotte, Mich., which appears to be the only other one reasonably successful, was openly said by its principal owner to have made a loss of $300,000 before it was able to produce soda at a profit.

Beside the ammonia process, electrolytic processes are now on trial in several places for the production of caustic soda and chlorine.

Numerous attempts have been made to utilize the various deposits of natural soda in our Western States. The article found there is mainly sodium sulphate, and the serious attempt made by the Union Pacific Railroad Co. to convert this into a marketable product ended in disaster and caused the abandonment of their works.

The Leblanc process proper has never obtained a foothold in America.

The method of manufacture known as the ammonia process—the only one really at work on a commercial scale in America, which was fully described in The Mineral Industry, Vol. II., in an article entitled "Alkali Manufacture in Great Britain"—is based on the theoretical reactions chemically expressed as follows:

(1) $2NaCl+NH_4HCO_3=2HNaCO_3+2NH_4Cl$.

(2) $2HNaCO_3$ (heated)$=Na_2CO_3+CO_2+H_2O$ for the manufacture of soda; and

(3) $2NH_4Cl+CaH_2O_2=2NH_4HO+CaCl_2$ for the recovery of the ammonia.

Salt, lime, ammonia and carbonic acid are the main materials, beside fuel, for the manufacture of soda.

The salt is derived mainly from natural brines, though in some places rock salt or even solar salt is dissolved to furnish this ingredient. In one case at least the attempt was made to concentrate weak natural brines for this purpose, but without success. As it is difficult to utilize much more than half of the salt in the brine, it is evidently necessary to obtain the salt cheaply. But this is not the only consideration. The brine must also be reasonably pure, as very small quantities of calcium and magnesium are detrimental to the apparatus. Where these salts amount to 1% of the solid matter in the brine, it is a serious matter; not only are they decomposed without useful effect, in the same manner as the salt and at the same expense, but also they foul the apparatus to such an extent that much cleaning is entailed; furthermore, in being transformed, these impurities almost always carry down with them a considerable proportion of the available soda, which it is impossible to recover. Another harmful impurity is sulphuric acid, which tends to make a very objectionable scale in the distilling apparatus; salts of iron also are to be avoided on account of their effect on the color of the product.

Several manufacturers have been sadly disappointed in their expectation that they could obtain pure brines by the action of water on great beds of pure rock salt.

Either the water from which the brine was made contained originally enough impurity to impair seriously the value of the brine, or else it took up enough from the rock walls of the salt, or perhaps both. Unfortunately it is true that the presence of salt adds materially to the solubility of several of the most harmful impurities, such as calcium sulphate, and it not infrequently becomes necessary to abandon a salt well on account of the increase in such impurities, which is probably due to the gradual exposure of veins by the continued erosion of the salt.

It is hardly necessary to add that brines of less than 95% salinometer test should not be considered for soda manufacture.

There are two principal types of kilns employed in the soda industry; the vertical type and the horizontal, or Hoffman kiln.

The latter is said to be more economical in one factory, where a number of vertical kilns have been shut down in its favor, but it is said to give a poorer gas. For the rest, there are nearly as many forms of kilns as there are soda factories. It would be onerous to go into details on a point where there is so much divergence of opinion as to what is best, especially as the best form probably depends to a considerable extent on the nature of the limestone to be burnt. Some stones require a materially higher temperature than others, and some cause trouble by falling to pieces on heating. The whole subject of lime burning (it is not a new industry) has been carefully treated in THE MINERAL INDUSTRY, Vol. VII. A sufficient number of kilns should be provided to burn at least three of limestone for every two of soda to be manufactured daily, with one additional apparatus necessary for repairs.

The quality of the limestone is likewise a matter for very careful considera-

tion. On general principles the purer the stone the better, but some impurities are far less injurious than others. Magnesia is nearly innocuous, but silica (especially in the soluble form), iron and alumina, and sundry other common impurities are to be avoided as far as possible in the choice of stone. At times, the chemical analysis fails to distinguish differences which nevertheless have a most important bearing on the value of the lime.

Theoretically, the ammonia continues to be used over and over again in this process, yet in practice there is a constant loss of this expensive material. Some fifteen years ago this loss amounted to about 5% of ash manufactured. To-day the loss should not rise above 2·5%, and while some factories certainly go a little lower, this figure would probably be difficult to realize for a new factory. It seems impossible to account for this loss. The quantities which thus pass off daily in a great soda factory are very large, and yet there is often no pronounced odor of ammonia to be detected anywhere, and the quantities which can be determined in the various escaping liquors and gases would not account for one-fifth of what is shown to have disappeared by the inventory. It is very characteristic of this industry that it is not the foreseen losses which make the main difficulty, but those which seem to be anomalous, and which appear unaccountable.

It is worthy of notice that in several large factories an important part of the ammonia consumed is obtained as a by-product of the manufacture of the coke used. This is accomplished by the use of the Semet-Solvay coke ovens, which were especially designed for this purpose, and are now being introduced in great numbers both in America and in Europe. This question, like that of lime kilns, is too nearly independent to be treated at length here, though the practice seems to be spreading, and as all the main products of retort coke ovens, *i.e.*, coke, gas and ammonia, find immediate application in soda works, it is only natural that soda makers closely driven by competition should look in this direction for a chance to make another economy.

As the brine reaches the works, it passes first through either a coke tower or a bubbling washer. The latter consists of six or more typical compartments of the ammonia soda process (Fig. 1), *i.e.*, a false bottom, having a short length of pipe which rises through a central hole and has over it a kind of umbrella with indented edges to assure that the gas rising through the central opening shall pass off on all sides of the umbrella. Some engineers have perforated the top of the umbrella for a more thorough distribution of the gas, but the value of this arrangement is questioned.

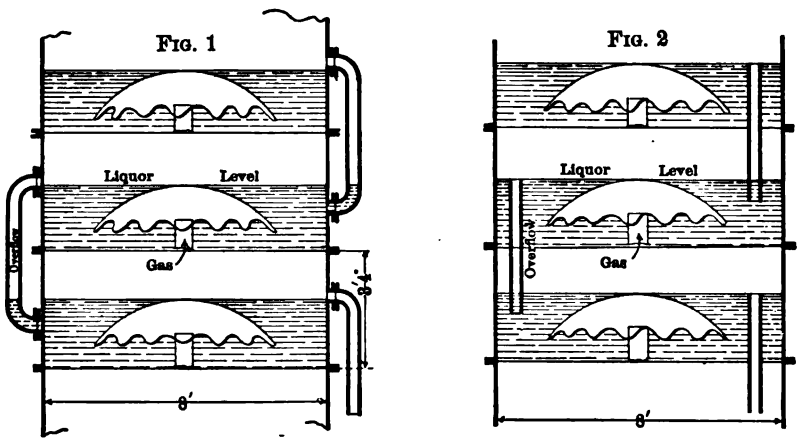
In this washer the brine meets the waste gases from different sources and washes the remaining ammonia out of them.

Some engineers have preferred to build these typical sections with internal instead of external overflows. In this manner, they claim to take up less room and to avoid an additional chance of leakage. The apparatus is constructed as in Fig. 2. After washing the escaping gases, during which operation a little ammonia and considerable moisture are taken up, the brine passes on to the ammonia saturator, a vessel consisting of two or three sections similar in

general character to those just shown, but with a somewhat greater depth of liquor and with a cooling apparatus, to take up the great amount of heat given off by the absorption of the ammonia.

From the saturator the brine passes on to a reservoir or reservoirs, where it is allowed to stand so that it will cool and clear itself should it be rendered turbid by the ammonia, as is frequently the case. The reservoir space should not be less than two cubic yards per ton of daily output. Dimensions for the washer vary somewhat. In some soda works, which have the advantage of natural, almost chemically pure carbonic acid gas from springs, the volumes of gas to be washed are much less than in those which depend upon the very impure gas from lime kilns. However, a 50-ton plant erected not long ago had its washer sections about 8 ft. in diameter, with from 15 to 18 in. depth of liquor in each section. The diameter of the saturator was about the same, with about twice the depth of liquor. These works used kiln gas.

The saturated brine goes from the reservoirs to a cooling apparatus, where it is chilled an amount dependent on the temperature of the water available for



Bubbling Washers.

cooling, and goes thence to the absorbing apparatus, where it meets the carbonic acid. If the brine fails to become clear in the reservoirs, it is filter-pressed on its way to the absorbing plant, to prevent the throwing down of insoluble matters with the bicarbonate which is there precipitated.

The towers for carbonic acid absorption are of the same general type as the washer, with variations to adapt them to their special duty. Carbonic acid is much more difficult to absorb than ammonia, and consequently these towers have been pushed to a great height. Lunge, in his treatise on "Sulphuric Acid and Alkali Manufacture," gives this height at from 16 to 20 m., equivalent to from 53 to 66 ft., and in some instances the height has been carried at least as high as 70 ft. There has been a reaction, however, from this extreme height and the most recent towers have been less elevated. At the same time the height of the sections has been reduced throughout a part of the height of the tower at least. These sections were formerly from 36 to 40 in. high, but have

now been reduced to as low as 24 inches. The principal characteristic of these towers is the suppression of the short tube over the central hole in the false bottom, and of the outside or other overflow, so that the liquor passes down through the same central orifice by which the gas rises, as shown in Fig. 3. The invention of this form of absorber was the crucial point of Solvay's process. Prior to its use all attempts at the ammonia process had ended in failure, due to the difficulty in the absorption of the carbonic acid, and Solvay himself had gotten into serious financial embarrassment before discovering it.

Since that time but one other apparatus has appeared which has competed against it at all. This was in successul operation two or three years ago, and presented at least one important advantage over the Solvay absorber, namely, better facilities for cleaning and repairs.

This apparatus consisted of a set of four inverted equilateral cones, each

FIG. 4

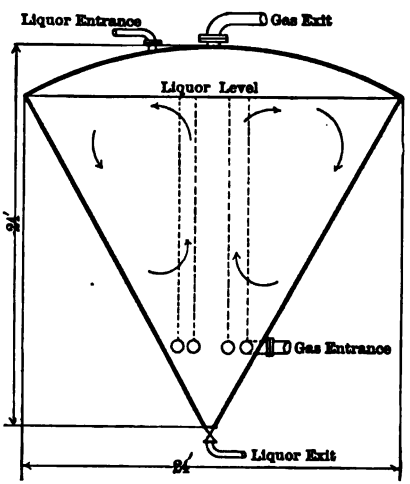
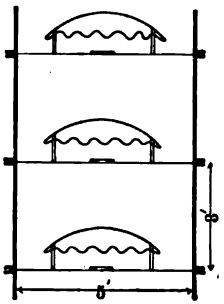


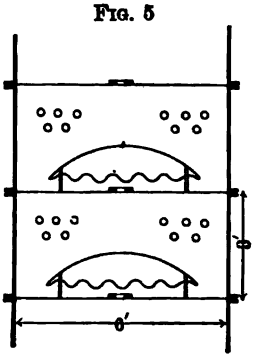
FIG. 3



ABSORPTION TOWER.

ABSORPTION APPARATUS.

FIG. 5



ABSORBER WITH COOLING TUBES.

about 20 ft. high, with a valve at the bottom, and a little above this, a series of concentric perforated coils of pipe, through which the carbonic acid was uniformly distributed over the central part of the tank, as shown in Fig. 4. This made the central part always lighter from the presence of bubbles in the liquor, which caused a rapid circulation up the center and down the sides and brought the liquor least acted on into contact with the gas. Of these absorbers one is always being discharged, one being filled, and two are in the intermediate stage. Generally, they are arranged in series so that the gas entering the last one, from which the liquor together with the precipitated bicarbonate are being drawn, passes up to the top of this and then down outside and into the bottom of the third, thence up through the third, and likewise through the second and to the first, which is being filled with fresh liquor, and finally to the washer. Whenever one is drawn empty, the fresh liquor is pumped to this and the drawing begins from its immediate successor in the

series, while the one which was previously filled enters the intermediate stage. This apparatus is reported to give at least as good decomposition of the salt as the Solvay absorber, and it is said never to foul, whereas a Solvay absorber requires frequent cleaning. This plant is said to have been sufficient for a production of from 60 to 80 tons daily, according to the season and the extent to which it was pushed. For this amount at least four Solvay absorbers 5 ft. in diameter and 60 or 70 ft. high were necessary, with at least one extra to allow for repairs and cleaning. The capacity of these absorbers is said to be increased by enlarging the section and adding cooling tubes, as shown in Fig. 5. The absorbers with these cooling tubes have never reached the degree of decomposition obtainable with the old form, which cools externally. The tubes themselves are an abiding nuisance, as the unequal expansion tends to cause endless leaks whenever the absorbers are steamed out to clean them, and require renewal from time to time, as they are corroded by the action of the surrounding liquor.

The crude bicarbonate resulting from the reaction in the absorbers is generally filtered out of the mass of mother-liquor accompanying it in vacuum filters, where a vacuum of from one-half to two-thirds of an atmosphere draws the liquor down into a tank under the filter and leaves the crude bicarbonate on the surface. This is then washed with warm water to remove the ammonia and chlorides remaining in the mother-liquor (the solubility of ammonium chloride increases far more rapidly than that of sodium bicarbonate with increase of temperature up to the point where the bicarbonate begins to decompose, *viz.*, about 150° F.), and is then dried with the vacuum till the moisture is diminished to 25 to 30%

Centrifugal machines have been tried recently for this filtration, washing and drying, and have reduced the moisture to 20% or possibly less; an important economy in view of the wasteful method of drying which is necessary to employ in the following stage.

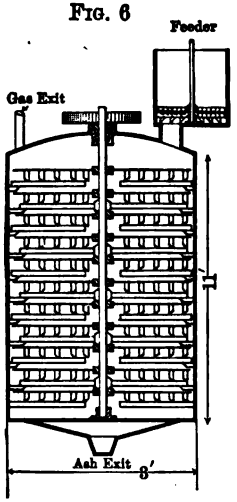
In this next operation the bicarbonate from the filters is dried and at the same time reduced to monocarbonate by the action of heat. To recover the carbonic acid given off, and particularly whatever ammonia remains in the crude bicarbonate, it is necessary to use indirect heat. Many different apparatus have been tried to accomplish this, some intermittent and others continuous, some with coal and others with gas fuel, but as yet none has proved satisfactory. The fuel consumption for this purpose is far above theoretical requirements and the repair account is serious. A recent apparatus, which has not as yet been as fully developed as the others, and which may possibly prove to have new advantages, is shown in Fig. 6. It consists of a series of round, hollow floors, over which the bicarbonate is plowed by inclined plows fastened to stirrer arms in such a manner that the first set drives it from the center, where it is fed in, to the outside, where it falls to the floor below. On this lower floor it is then driven to a central hole where it falls again to the next floor below, and so back and forth from floor to floor until it reaches the bottom in a finished condition, and is there drawn out and passed to the grinding apparatus. The products of the combustion which decompose and dry the bicar-

bonate and convert it to ash in its progress through the apparatus, pass upward, zigzagging back and forth through the hollow floors. It is continuous in action and there seems to be no reason why it should not be tight, and thus avoid leakage of carbonic acid and ammonia. If such is the case, it more nearly fulfills these requirements than any other apparatus known to the writer. This apparatus is said to be comparatively economical of fuel.

From the roasting operation the ash passes to the mills where it is ground and thence to the bolts, where it is sifted to the requisite degree of fineness and is then ready to be packed and shipped.

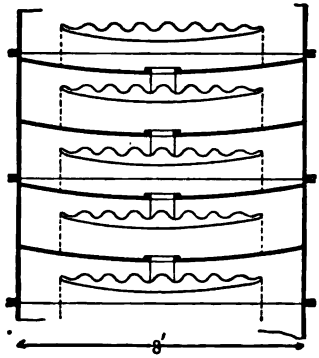
There yet remains to discuss the recovery of the ammonia. The mother-liquor, after separation from the crude bicarbonate in the centrifugals or other filters, is distilled. It is first treated in an apparatus where the carbonic acid is driven off, and for this purpose eight or ten rings similar in type to those of the washer shown in the first sketch may be employed; this can be accom-

FIG. 6



DRYING APPARATUS.

FIG. 7



APPARATUS FOR EXPELLING CARBONIC ACID.

plished with less counter-pressure by substituting double the number of perforated plates, so arranged as to cause a continuous rain of the ammonia liquor across which the volume of steam used for distillation is compelled to pass on its way through the apparatus, as shown in Fig. 7. It is then mixed with milk of lime and distilled, until entirely free from ammonia, in another apparatus, which consists entirely of rings of the washer type. These rings should be half as large again as those of the washer, and eight or ten are necessary for this operation. The gases coming from the bicarbonate roasters should also be washed in two or three similar rings to cool them and to remove the ammonia before it is mixed with the gas from the kilns.

A soda plant should be provided with a boiler plant capable of burning not under 1·25 tons of coal per ton of ash to be produced, and in districts where the boiler feed is bad, which is almost universally the case with soda ash man-

ufacture, an ample margin should also be allowed for cleaning and changing boilers.

It has been estimated that a plant completely equipped would probably cost from $30 to $40 per ton of annual output, and with the present inflated prices for machinery, etc., the upper limit is not too high.

None but factories which have reached a state of perfection by improvements made from time to time, will be likely to have all of the advantages of equipment possessed by the most complete establishments. Many of these advantages, however, are indispensable when working on the extremely narrow margin of profit which has lately characterized the soda industry. For instance, several factories have put up very expensive plants for the recuperation of the lime in the waste. This plant alone in one factory is understood to have cost about a dollar per ton of annual production, and it is doubtless considered economical, for this same concern has recently erected a second plant for a factory previously unprovided with it. Gas producers and electric plants for driving parts of the machinery have been applied on a very extensive scale in many factories, and an expensive repair shop is a necessary adjunct for any works situated away from the immediate neighborhood of machine shops.

The amount of labor necessary for the production of soda ash varies extremely, on one hand according to the precautions taken for the highest possible economy of the main materials used—brine, limestone, fuel and ammonia, and on the other, with the special appliances for economy of labor. Naturally, the highest economy of material is incompatible with the lowest figures for labor, and it is necessary in each individual case to make a special study of local conditions to determine the most economical quantity to be used. In some European factories it is understood that the labor requisite is over two days per ton of ash produced, while in others it does not exceed a day and a half, exclusive of the amounts of labor required for packing and shipment, repairs, etc. The amount also varies somewhat with the general intelligence and experience of the workmen. The amount of labor necessary for repairs is very heavy, and this again of course varies greatly with the complications of the machinery. On the whole there seems little doubt that the more highly equipped factories are the most economical, especially in this country, where labor is expensive, although, especially in starting, a very highly complicated factory is doubly liable to accident, and accidents set up some of the most serious difficulties in this line of manufacture.

Another fact which must be considered is the danger of subsidence of the surface when brine continues to be drawn in large quantities from underground. The English Government has dealt with this question in the great English alkali manufacturing territory by imposing a tax on all soda manufactured in the district, from the proceeds of which, real estate owners are compensated for the damages due to subsidence; and doubtless some such measures will be necessary in other places. The amounts thus raised in England are graduated from time to time according to the requirements, and a special board is appointed to award compensation from it to sufferers.

Very close attention to market conditions is essential in locating the soda

ash industry. There must be sufficient market available to take care of the entire output. It is necessary to be able to maintain very near the maximum output, for the production of a factory can only be varied economically within very narrow limits, as any considerable decrease in the amount produced adds materially to the cost per ton of production. If it were not for the dispersal of the working force, it would generally be much more economical to shut down the works entirely than to run on half time, and the erection of any competing factory in a territory already sufficiently supplied by existing works, instead of reducing selling prices by competition, would be more likely to raise them by increasing the cost of production for each factory of which the output was reduced by the new product coming upon the market.

In spite of the increase in consumption with the growth of the country, importations have been very heavily reduced in the last few years, and it is estimated that the new plant now being erected at Barberton, O., will be more than sufficient to produce an amount equal to the imports in the last fiscal year, and beside this, important extensions are reported to be under construction in the existing factories. It looks at present as if any new attempt at the manufacture of soda in this country could only be successful by overthrowing some of the factories already established.

THE ELECTROLYTIC PRODUCTION OF CHLORINE AND CAUSTIC SODA.

BY ALFRED T. WEIGHTMAN.

THE past year has been particularly active in this industry, prompted no doubt by the success which electrolytic companies have met with both at home and abroad. American companies, owing to the abundance of waterfalls, enjoy an advantage which many companies of England and the Continent do not possess, though still able to pay substantial dividends. Some half a dozen companies have been incorporated in America during the past year to undertake electrolytic alkali work, and numerous patents have been issued. Great secrecy is observed with regard to their intentions, but as far as is known only two of the new companies seem to be making any headway—the Acker Process Co. and the American Alkali Co.

The Acker Process Co. was formed early in the past year, with a capital of $5,000,000, to take over the patents of Mr. C. E. Acker for manufacturing bleach and caustic soda at Niagara Falls. This is a fusion process in which the metallic sodium is deposited from fused sodium chloride into metallic lead. The lighter alloy of lead and sodium is continuously drawn off from the surface of the fused metal and the sodium extracted in the form of caustic soda by means of steam or water. Large works are now being erected at Niagara Falls, and it is expected that they will be placing their products on the market early this year.

The American Alkali Co. was incorporated in Philadelphia, with a capital of $30,000,000, primarily to take over the Canadian and American rights of the Rhodin process from the Commercial Development Corporation of England. Works are to be established at Sault Ste. Marie, Mich., utilizing 20,000 h. p.

from the enormous power canal there, which is now fast approaching completion, and also at Sault Ste. Marie, Ont., for the purpose of supplying the Canadian markets. The latter works, now in the course of erection, will utilize about 2,000 h. p. from the Lake Superior Power Co.'s canal. As transportation facilities are excellent and salt and limestone readily obtainable in the vicinity, both on the Canadian and American sides of the river, the situation is an ideal one for the establishment of an electrolytic alkali works. The Rhodin cell, which has already been described* is on the moving mercury cathode principle, in which the mercury receives its charge of sodium in the anode compartment and gives it up in the cathode compartment.

This constant change of surface is brought about by the motion of a number of inverted bell-jars, the lips of which just touch the surface of the mercury and which contain the brine and positive electrodes, the mercury itself acting as the negative electrode. On the charged mercury being transferred to the cathode compartment, the sodium is not taken out by means of the current, as in the case of the Castner-Kellner and other cells, but merely by its action on the water, the iron side of the containing vessel forming with it what chemists term a "galvanic couple." A great advantage of the Rhodin cell is that the rapid interchange of mercury from one compartment to another effectually keeps down the saturation of the mercury, reference to which is made elsewhere. As the solubility of sodium in mercury is very slight at temperatures below the solidifying point of the amalgam, high current densities can be used and, it is claimed, a small cell be made to do a considerable amount of work. The Rhodin process was bought out some two years ago by the Commercial Development Corporation, who erected a small works near Manchester, England, but from the first were harassed with legal proceedings against them by the Castner-Kellner Alkali Co., who claimed an infringement of their patents. Judgment was given in favor of the Castner-Kellner Co., but on appeal the judgment was reversed, and the matter is now awaiting its final decision. The American Alkali Co. also possesses the Hermite & Dubosc patent, which, although in itself scarcely a practical cell, is said to anticipate all cells using a moving mercury cathode.

Among the older works the Mathieson Alkali Co. are now working the Castner-Kellner process at Niagara Falls with such success that the plant is about to be doubled in size. The capacity of the present plant is about 10 tons of caustic soda and 24 tons of bleach per day. There are 11 motor dynamos, each capable of giving 500 ampères at 225 volts. Ten of these are each connected on to 60 cells, the eleventh being kept in reserve in case of accident. This is driven by means of a gas engine, so as to be entirely independent of the outside electrical supply. If any interruption of this supply takes place, this dynamo is immediately switched on to all of the 600 cells, which are arranged in parallel across the main leads in 10 sets, each containing 60 cells in series. In this way a small current (50 ampères) can be sent through all the cells, not so much for the purpose of doing any appreciable amount of work, but to keep the cells continuously working. If the cells stop working for more than a few minutes

* THE MINERAL INDUSTRY, Vol. VII., p. 243.

local action takes place on the surface of the sodium-charged mercury in the anode compartment, producing caustic soda and eventually sodium hypochlorite. When the cells are restarted these undesirable products give trouble, as under electrolysis they evolve oxygen and not chlorine at the anode, which lowers the commercial efficiency of the cell. This evil is still further exaggerated when carbon anodes are used, since these rapidly deteriorate when oxygen is evolved at their surface, although they will stand chlorine well. To guard, therefore, against this two-fold evil it is found commercially economical always to keep a current passing through the cells. Each cell measures 6×3 ft. ×6 in., being divided into three compartments each, 6×1 ft.×6 in. The middle compartment contains the cathode, the two outer ones being anode compartments. The cells are rocked by eccentrics on a shaft extending the length of the cell room, beneath each row of cells. The eccentric has a throw of about 0·75 in. Each shaft is connected by worm gearing to another shaft in the dynamo room running at right angles, driven by an electro motor. As the anode liquor contains a considerable quantity of chlorine and a small quantity of sodium hypochlorite, it is highly corrosive and has to be elevated by means of an earthenware pump, a gutta-percha pipe being used to convey the liquid.

The oldest process in America, and probably the oldest in the world working successfully on a commercial scale, is the Le Sueur process. This was taken up about 1892 by the Electro Chemical Co. Works were erected at Rumford Falls, Me. At the present time the cell is quite different in general appearance from the original cell, although retaining the original principles. It is a horizontal diaphragm cell, as Le Sueur first saw the advantage of maintaining a higher level of solution on the anode side of the diaphragm than on the cathode. By steadily keeping a higher level on the anode side, a steady flow of solution is maintained through the diaphragm and any tendency which the caustic liquor on the other side may have to diffuse back into the anode compartment is compensated by this steady flow. The cells are now made in a rectangular form 9×5×1·5 ft. deep, being constructed of sheet iron with brick and slate cement lining. The diaphragm is not quite horizontal, but is made in the form of a W to facilitate the escape of the hydrogen gas. The whole anode compartment is covered with ordinary roofing slates fastened on with slate cement and drilled for the purpose of receiving the anodes, which are of special construction. The diaphragms, which are formed of asbestos, last from three to six weeks and represent only a small item in the cost of working. Each cell of the above size takes about 800 ampères at 6 volts and gives an efficiency of 70 to 80%. It is capable of turning out liquor as high as 10 to 15% caustic, and can be made to yield higher than this at the sacrifice, however, of efficiency.

The anode calls for special attention, as it is constructed of fine platinum-iridium wire containing 10% of iridium and 90% platinum. The wire is cut into 4-in. lengths and flattened out for about three-quarters of their length. Twenty of these are bunched together, and the unflattened ends are fused into the end of a glass tube about 1 ft. long, so that the ends just project through into the inside of the tube. They are then bent over and spread out radially, contact

being made with the ends fused into the glass tube by means of a drop of mercury into which dips an iron rod as long as the tube. Each anode which costs about 75c., is capable of carrying 10 ampères without any undue drop of voltage. Not only is this probably the cheapest platinum-iridium anode ever introduced, but it has the special advantage that the fine wire allows the chlorine to escape readily from the surface and so reduces polarization. About 80 of these anodes are placed in each cell of the above size and are suitably connected to the main leads by means of small cups containing a drop of mercury in which copper connecting pieces dip. The Electro Chemical Co. have now dismantled their plant at Rumford Falls and have rebuilt at the works of the Burgess Sulphite Pulp Co., Berlin Falls, N. H., where the chlorine is being used for the bleaching of sulphite wood pulp, the caustic liquor being used in another stage of the process. In this way great economy is effected, as both caustic soda and chlorine are used in solution, saving expense of concentration and making bleaching powder. Furthermore pulp mills are often situated near water power so that the expenditure for power is reduced to a minimum. In this case the energy efficiency of the cell is not a matter of great consideration.

Comparison of Various Electrolytic Methods.—At the present time there are three distinct methods of producing bleach and caustic soda electrolytically: (1) Mercury cathode (examples: Castner-Kellner and Rhodin processes); (2) porous diaphragm (examples: Le Sueur and Hargreaves-Bird processes); (3) fused electrolyte (examples: Hulin and Acker processes).

Of these the first is probably regarded the most favorably at the present time, but the second and third certainly run it very close. Each one has its advantages and disadvantages.

1. *Mercury Process.*—Mercury cells, as a rule, give fairly high current efficiencies and as the voltage is only about 4 volts per cell, the watt efficiency is also fairly high. In a well designed mercury cell there is great freedom from any secondary reactions which caused so much trouble formerly, but it must not be assumed that this is necessarily the case. In order that too great a saturation of sodium in the mercury should not occur, it is important that the mercury should change frequently from anode to cathode compartment, otherwise decomposition will take place in the former, notwithstanding the connection there of the mercury with the negative pole. In the Castner-Kellner cell the mercury is kept steadily at about 0·2% of sodium. On the other hand if the amount of sodium in the mercury is allowed to get too low, the current having little or no sodium to oxidize in the cathode compartment starts oxidizing the mercury.

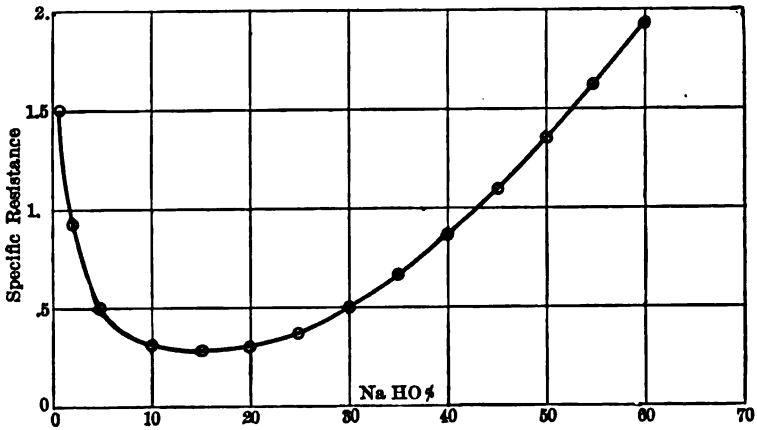
In a cell in which the sodium is not taken out by the current but by the galvanic couple formed by a secondary electrode, such as iron, the decomposition of the sodium in the anode compartment is still more liable to take place. Of course in this way all chance of oxidizing the surface of the mercury is avoided, but on the other hand a much larger surface of mercury is necessary in the anode compartment, entailing greater expense and plant for a given capacity. The relative superficial areas of the anode and cathode compartments in this case have to be in the proportion of about 1 to 6 with normal current densities.

Should the cathode area not be sufficiently large the mercury becomes too highly saturated, when local action takes place in the anode compartment, forming caustic soda and eventually sodium hypochlorite, with all its resultant evils.

The removal of the sodium from the mercury in the cathode compartment must be at the same rate as its deposition in the anode compartment. Traces of foreign matter, such as carbon, on the surface of the mercury will bring about the same effect as saturation, owing to local action with the highly negative amalgam. These difficulties can be overcome in a properly designed cell and a further advantage is then gained by the absence of these secondary reactions and their consequent secondary products, namely, that carbon anodes can be used, since the evolution of oxygen (the chief agent in the destruction of carbon anodes) is reduced to a minimum.

Fairly strong solutions of caustic soda can be obtained, but it becomes commercially uneconomical when the resistance of the liquor becomes excessive.

Caustic soda solutions, in common with other electrolytes, do not necessarily decrease in electrical resistance as the strength of the solution increases. When we commence with water with its high resistance and add say 5% of sodium hydrate to it, the resistance decreases enormously. When made up to 10% it is still further reduced but to a less extent than the previous 5% reduced it. The resistance still diminishes, but in a less and less proportion for every 1% of soda added until it reaches about 15%, when the resistance keeps practically steady for some time. It then actually *rises* in a greater and greater proportion as more soda is added, as the accompanying curve and table will show:



CURVE SHOWING THE RELATION BETWEEN THE SPECIFIC RESISTANCE AND THE STRENGTH OF VARIOUS SOLUTIONS.

NaHO%.	Specific Resistance.	NaHO%.	Specific Resistance.
1·0	1·502	30·0	0·498
2·5	0·925	35·0	0·669
5·0	0·511	40·0	0·867
10·0	0·322	45·0	1·082
15·0	0·290	50·0	1·350
20·0	0·301	55·0	1·625
25·9	0·371	60·0	1·925

Percentages.

So that from a purely electrical point of view the solution should not be allowed to go above 15%. To what extent it is more economical to work at a higher strength than this, so as to reduce the fuel consumed in concentrating the caustic liquor than to contend with the greater increase in resistance, depends on the relative prices of fuel and power in the particular locality. This applies equally to the case of cells in which the sodium is not taken out of the mercury by sending a current through it, but by "local action," which decreases in activity as the electrical resistance of the solution increases. The strength of the soda solution may be further limited by the fact that a certain amount of it may be carried mechanically into the anode compartment; and the stronger the solution is the more soda is carried over. This is brought about by the mercury becoming wet and then in passing from the cathode to the anode side not being wiped completely dry by the dividing partition. It follows also that this equally applies when the mercury passes in the other direction, small quantities of brine being carried into the caustic liquor, and it is sometimes necessary to pass the liquor through a salting-out pan before discharging it into the caustic pots, or more generally to eliminate the salt by fishing from the caustic pots themselves.

The objections to mercury cells are their high initial cost and the large amount of mercury required. About one ton of mercury is required for each ton of caustic produced weekly. Thus a plant producing 10 tons a day requires 70 tons of mercury. Though the power required for the mechanical movement is small, still it adds to the complication of the plant. Further, there is the disadvantage of working with rock salt instead of brine, and the expense of crystallizing it.

2. *Diaphragm Process.*—In general the diaphragm process does not give a higher efficiency than 70 to 80%. Since the brine, containing a considerable amount of chlorine in solution, is constantly subjected to the reducing action of the hydrogen from the cathode and however perfectly they may be kept apart mechanically, it is essential that at some period the charged brine will have to come in contact with the cathode. This is a necessary condition of the diaphragm cell, and it not only represents a loss of chlorine but an equivalent loss of soda, either by the sodium combining directly with the chlorine at the moment of its deposition or by the action of the hydrogen, formed by the secondary reaction of the sodium on the water, thus forming hydrochloric acid which eventually reaches the cathode compartment and there neutralizes an equivalent of caustic soda. The voltage runs fairly high (5 to 6 volts), but this is largely dependent on the resistance of the diaphragm, which resistance is itself largely dependent on its ability to check diffusion. In the Hargreaves-Bird process a lower voltage and higher efficiencies have been claimed than the above, but in this case carbonate was being made and not sodium hydrate. In making carbonate there is this advantage: owing to the much higher resistance of the sodium carbonate solution than the sodium hydrate solution, the current is carried from anode to cathode to a lesser extent by the carbonate than by the hydroxyl ions, and consequently to a greater extent by the sodium chloride ions.

It is very important in all diaphragm cells to get a uniform rate and equal distribution of flow over the whole diaphragm surface. This is prevented by (1) unequally distributed porosity of the diaphragm; (2) unequal pressures of the solution at various points on the diaphragm. Of these two causes the first can be remedied by paying proper attention to the mechanical construction of the diaphragm, but the second presents greater difficulty. If a cell is divided into two portions by means of a porous vertical diaphragm and solutions of different specific gravities poured into each side, the pressures over the surface will not be evenly distributed.

Let us consider Figs. 1, 2, and 3. In all cases the solution *A* is of greater specific gravity than *B*, but in Fig 1 the two solutions are at a level; in Fig. 2 *B* is higher than *A*, and in Fig. 3 *B* is lower than *A*. Then the curve of pressures of the solution, and consequently the rate of flow at any point, will be represented in each case by the lines, *x*, *y*, *z*. The result is that if the rate of flow is

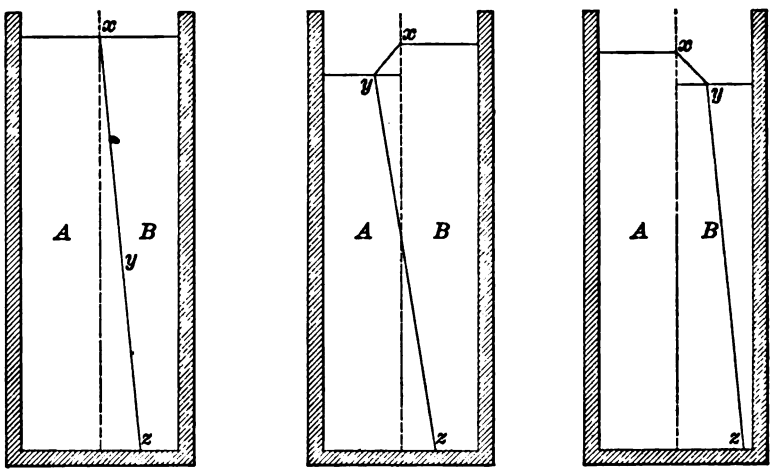


FIG. 1. FIG. 2. FIG. 3.

ILLUSTRATIONS SHOWING THE CURVE OF PRESSURE OF THE SOLUTION AND THE CONSEQUENT RATIO OF FLOW AT ANY POINT.

correctly adjusted at any one point, it is either too fast or too slow at any other point. This difficulty can be met in one of three ways: (1) By having the distribution of the electric current uniform over the diaphragm surface, but constructing the diaphragm so as to be more porous at the points of least pressure and less porous at the points of greatest pressure; (2) by having the diaphragm of uniform porosity, but so arranging the electrodes that where the flow of solution is greatest the electric current shall be proportionately great and vice versa; (3) by having a horizontal diaphragm, in which case the pressures would be equally distributed over the surface.

A great disadvantage of the diaphragm process is the comparatively weak liquor which it produces, seldom over 20% caustic, frequently less; the extra cost of concentrating and salting out any sodium chloride which may be present in this is objectionable. Where the caustic liquor can be used direct at the

works this objection is not valid. The cells are invariably cheap to construct and their depreciation is slight. They require very little skilled labor and excepting the periodical renewal of the diaphragms, few repairs. Brine can be used direct without crystallizing out the salt and in the case of works which are situated near salt wells this is important.

3. *Fusion Process.*—In this process the salt is fused in some form of crucible by external heat or by the electric current. When a current is sent through the fused salt, chlorine is evolved at the positive electrode and sodium at the negative. As the sodium is of less specific gravity than the fused electrolyte, unless means were adopted to keep it at the bottom of the crucible it would float and so recombine with the evolved chlorine. It is therefore usual to have a layer of molten lead on the bottom of the crucible to alloy with the sodium from which it can afterward be extricated by treating with water or steam, yielding caustic soda; else by volatilizing the sodium which leaves the lead behind. As the sodium alloy is considerably lighter than the lead it does not form a uniform alloy with the latter. Hulin overcomes this difficulty by having a certain amount of lead chloride present with the fused salt and thus deposits lead directly with the sodium. Acker draws off the richer alloy continuously from the surface of the molten metal and after extracting the sodium by means of steam, discharges the lead again into the bottom of the crucible to receive a fresh charge of sodium. As fused salt is a very good electrical conductor, very high current densities can be used and consequently a large output can be obtained from a comparatively small plant. The further advantage lead has over mercury of forming a sodium alloy is that the cell being at a fairly high temperature, alloys can be obtained, as high as 20% or more, without any appreciable decomposition taking place. Another point is that very concentrated solutions of caustic can be obtained and the cost of evaporating partly avoided. The depreciation and renewal of fusion pots is, however, very heavy and particularly so when the electrolyte is kept in a state of fusion by the application of external heat and the red hot chlorine evolved from the cell is not an easy thing to deal with. Whichever way it is done the cost of fusion must be considered. Notwithstanding numerous statements to the contrary, the watt efficiency is low. Kershaw (see MINERAL INDUSTRY, Vol. VII.) says 41·5% as against 37 to 56% for other processes, but here the calculation is made for metallic sodium, which although perhaps the final product of the cell is not the final product of manufacture, with which the financial success of the process is concerned, and the heat of the reaction of sodium and water is lost, since it takes place outside the cell. Consequently, in comparing it with other forms of cells for manufacturing alkali, this lost energy should be taken into account.

Progress in Europe and in the United Kingdom.—The year 1899 has been marked by considerable development in the electrolytic alkali and chlorine industry in Europe; and in the United Kingdom the period of keen competition between the old Leblanc process and the new electrolytic processes, has now been entered upon. On the Continent the Leblanc process has never been very widely adopted, and the future competition in caustic soda production will be chiefly between the electrolytic and the Solvay ammonia-soda processes. Since

the latter process is still unable to recover economically the chlorine of the waste calcium chloride liquors, the electrolytic processes have a decided advantage, and it is in Germany, France, Switzerland and Russia, therefore, that the most striking progress is now occurring.

That the oldest of the three processes for making alkali and chlorine compounds will not succumb without a long struggle, is certain; and it is the opinion of those best qualified to judge that the three methods of manufacture —the Leblanc, the Solvay and the electrolytic, will continue to exist side by side for many years to come. Each of the older methods offers secondary advantages, which to some extent balance the greater simplicity and directness of the electrolytic processes; and in the Leblanc process especially, "by-products" of considerable value are obtained.

Turning now to a consideration of details, we find that the processes worked by the Elektron Co. of Frankfort, and the Elektrochemische Werke of Bitterfeld show the most rapid development. This is partly due to the German system of industrial finance, which permits the large electrical engineering firms to co-operate actively in the establishment of the electrolytic alkali works. These processes are reported to be diaphragm processes, and were originally used solely for the production of caustic potash and chlorine compounds; but they are now we believe being used for production of caustic soda also. The diaphragms are of cement, and with great care in their use they may, according to Haber, last two years. Great secrecy has been observed with regard to the cell details and beyond the facts named nothing has been allowed to escape. In 1898 the two companies arranged to work in combination and to share profits, and during 1899 this consolidation of interests has been strengthened by an additional agreement, under which the Allgemeine Elektricitats Gesellschaft of Berlin sinks an additional $864,000 in these undertakings. The Elektron Co. of Frankfort is now to take the technical control of all the works owned by the combination. The following financial details of the development of the two parent companies will be of interest, in the light of their success in establishing subsidiary undertakings in almost every country in Europe.

The Chemische Fabrik Elektron Aktiengesellschaft was formed in 1889, to manufacture alkalies and chlorine products by an electrolytic process at Griesheim, near Frankfort. These works commenced to operate in 1890, and in 1893 a second works was erected at Bitterfeld in Saxony. In 1897 with a share capital of $960,000, the net profits were $110,400, and the reserve fund stood at $187,200. The present capital of this company is $2,160,000, and the title has been changed to Chemische Fabrik Griesheim Elektron. The Elektrochemische Werke, Bitterfeld, was formed in 1893 with a capital of $720,000 for the production of electrolytic alkali and bleach, and works were started at Bitterfeld in 1894. In 1898 a second large works was erected at Rheinfelden, and 5,500 h. p. is now being utilized at these two factories.

During 1898 and 1899 both these companies have been very active in promoting the establishment of similar factories in other countries of Europe, and the following list of places where factories using either the Elektron or

Bitterfeld process will be operating in the year 1900, is believed to be correct: *Germany*, Griesheim, Bitterfeld (2), Rheinfelden, Ludwigshafen, Westeregeln; *France*, Moutiers, La Motte; *Russia*, Zombkowitz, Slaviansk; *Spain*, Flix.

The process which ranks next to the Elektron and Bitterfeld processes in the vigor of its development is the Castner-Kellner. This process is now being operated at Weston Point, England, and at Osternienberg in Germany; while additional works will be completed during 1900 at Jenappe in Belgium, and at a place near Moscow. The factories at the last three places are promoted by Solvay et Cie of Brussels who own the Castner-Kellner patent-rights for Europe. At Osternienberg 1,000 h. p. (steam) is being utilized, while at Weston Point the plant has been increased to its originally designed limit of 4,000 h. p. during the past year. At the annual meeting of the English company, held in May, when the accounts up to March 31, 1899, were presented, it was stated that the total outlay upon the works at Weston Point had been $1,401,600, and that the net profits had been $141,859. This had been earned by the operation of 3,000 h. p. for varying periods—equal in the aggregate to 1,000 h. p. for 27 months. An 8% dividend was declared for the year, and a similar interim dividend has been declared for the half year ending September 30, 1899. The company is therefore a financial success for the present, and although the arrangement with the Solvay Co., for the sale of the bleaching powder alluded to in THE MINERAL INDUSTRY, Vol. VI., ends with the current year, the directors are confident that they can sell this product in the open market, at a price that will leave them a profit. The capital of this company has been raised to $2,160,000 during 1899 by issue of debentures. It is reported that the company has latterly been losing mercury in some unexplained way, and that platinum has been substituted for carbon in the anode compartments of the cells, owing to the disintegration of the latter. The loss of mercury may of course prove serious, if it proceeds from an unavoidable cause; but the fact that this company has worked for eighteen months with financial success, would seem to indicate that the loss of mercury is due to some temporary difficulty. The substitution of platinum for carbon will simply mean an increased capital outlay per unit of plant, and rather higher depreciation charges.

The Hargreaves-Bird process, of which full details have appeared in earlier volumes of THE MINERAL INDUSTRY, is now operating at two places, namely, at Farnworth in England, and at Chauny, near Paris. During 1899 the long delayed flotation of the larger company which is to work this process in England was effected, and in November the prospectus appeared of the Electrolytic Alkali Co., with a capital of $2,400,000, of which at present only $1,440,000 is being raised. It is proposed to erect works with a decomposing capacity of 9,000 tons salt per annum, at Middlewich in Cheshire, and it is estimated that the capital outlay upon this works will not exceed $672,000. Some portions of this plant are expected to be ready for operation during the current year. The further developments of this interesting diaphragm process will be watched with considerable interest both in Europe and America. The small experimental works at Farnworth, near Widnes, continued to operate during

1899, but it will be shut down before manufacture at the new works is commenced.

The remaining electrolytic alkali processes now being operated in Europe are each represented by only one works, and details concerning their progress during 1899 are scanty.

The Richardson & Holland process at St. Helens, England, has again met with difficulties; during the past year there has been a temporary stoppage, and also a change of management. These difficulties have now been overcome, and in November, 1899, the plant was again put in operation under conditions more favorable to success. By the improved process it is stated that bleaching powder will cost only $13·20 per ton.

With regard to the progress of the Hulin process (of which very full details were given in THE MINERAL INDUSTRY, Vol. VII.), at Les Clavaux, France, there is nothing new to report. In July, 1899, Mon. Hulin informed us that the process was operating well and was fulfilling expectations. The Acker process, which is about to be worked at Niagara Falls (see above), is a similar process, fused salt and molten lead being used as electrolyte and cathode material respectively.

The erection of works for the "Consortium für Elektrochemische Industrie," at Golling in Austria, which was commenced in 1897, has been delayed owing to the lack of financial support, but fresh financial arrangements were made in 1898, and it was expected that bleaching powder, caustic alkalies and chlorate would be produced there under the Kellner patents in 1899. The capital of the reorganized company was to be $1,480,000, and 4,500 h. p. (water) was to be utilized. We have been unable to learn whether the promised developments have been carried out.

In addition to the above, electrolytic alkali processes of which no details can be obtained, are already in operation, or will be operating in 1900, at the undermentioned places: Mieussy, St. Michel, Chevres, Monthey, Thusis; and similar works are projected at two unnamed places in Italy.

The following table gives in convenient form the information relative to the electrolytic alkali works in Europe:

Country.	In Operation.		In Course of Erection.	
	Places.		Places.	
United Kingdom.	St. Helens, Farnworth, Weston Point	8	Middlewich.	1
France	Chauny, Les Clavaux, La Motte, Mieussy, Moutiers	5	St. Michel.	1
Germany	Griesheim, Bitterfeld (2), Rheinfelden, Ludwigshafen, Osternienberg	7		.....
Austria	Golling	1		.....
Russia	Zombkowitz	1	Slaviansk, Moscow	2
Switzerland	Chevres, Monthey	2	Thusis.	1
Belgium	.....	.....	Jenappe.	1
Spain	.....	.....	Flix.	1
Italy	.....	.....	Unnamed.	2
Total	.....	19	Total	9

The year 1899 did not bring any new process prominently forward in Europe,

and the developments recorded above have been chiefly in connection with the older processes of electrolytic alkali manufacture.

The litigation between the Castner-Kellner Co. of England, and the Commercial Development Corporation, with respect to the validity of the patents for the Rhodin cell reached a further stage; and in April the English Court of Appeal delivered a judgment in favor of the latter company. The Castner-Kellner company now talk of carrying the case to the "House of Lords," the final Court of Appeal under English law.

Borchers, in *Zeitschrift für Elektrochemie* (July 20, 1899), gives the following quantities as the output of electrolytic bleach and caustic soda throughout the world in 1898:

	Caustic Soda. (70%) In Metric Tons.	Bleaching Powder (40%). In Metric Tons.		Caustic Soda. (70%) In Metric Tons.	Bleaching Powder (40%). In Metric Tons.
Austri	7,200	15,750	Russia	2,400	5,250
Belgium	1,600	3,500	Spain	3,860	7,350
France	45,280	99,050	Switzerland	3,300	7,200
Germany	2,600	51,200	United States	5,120	11,200
Great Brita.n	11,200	24,500		82,560	225,000

In the same paper appears an interesting list of all the electro-chemical and metallurgical plants at present known to exist, of which the following apply to alkali, chlorine and chlorates, and include certain additions that have been made by the writer:

Process Used.	Company.	Proposed.			In Operation.		
		Water Power. H. P.	Steam Power. H. P.	Gas Power. H. P.	Water Power. H. P.	Steam Power. H. P.	Gas Power. H. P.
UNITED STATES.							
Acker	Acker Process Co., Niagara Falls, N. Y.	.....	.....	.....	.....	.....	.....
Le Sueur	The Electro-Chemical Co., Rumford Fall, Me.*	.....	.....	.....	1,200	.....	.....
Castner-Kellner	Mathieson Alkali Co., Niagara Falls, N. Y.	.....	.....	.....	2,000	.....	.....
Rhodin	American Alkali Co., Sault Ste. Marie, Mich.	20,000	.....	.....	.....	.....	.....
Franchot & Gibbs	National Electrolytic Co., Niagara Falls, N. Y.	.....	.....	.....	.....	.....	.....
Hurter	North American Chemical Co., Bay City, Mich.	.....	.....	.....	.....	.....	.....
BELGIUM.							
	Solvay & Co., Werke zu Jemeppe.	.....	1.000	.....	.....	1,000	.....
GERMANY.							
Elektricitäts A. G. vorm. Schuckert & Co., Nuremberg.	Badische Analin- und Soda Fabrik, Ludwigshafen	.....	.....	.....	.....	1,300	.....
Elektricitäts A. G. vorm Schuckert & Co., Nuremberg	Konsolidierte Alkali Werke, Westeregeln	.....	.....	.....	.....	1,300	.....
	A. G. Chemische Fabrik Buckau, Fabrik Ammendorf	.....	.....	.....	.....	.....	.....
Castner-Kellner	Deutsche Solvay-Werke. A. G. Bernberg	.....	1,000	.....	.....	1,000	.....
Electrochemische. Werke, Bitterfeld	Elektrochemische Werke, Bitterfeld	.....	3,000	.....	.....	3,000	.....
	Elektrochemische Werke, Rheinfelden	3,400	.....	.....	3,400	.....	.....
A. G. Elektron, Frankfurt a. M.	A. G. Elektron, Frankfurt a. M.	.....	.....	.....	.....	.....	.....
Elektricitäts A.G. vorm Schuckert & Co., Nuremberg	Werke zu Griesheim, Bitterfeld Ludwigshafen	.....	.....	.....	.....	4,000	.....

* This plant is now dismantled.

Process Used.	Company.	Proposed.			In Operation.		
		Water Power. H. P.	Steam Power. H. P.	Gas Power. H. P.	Water Power. H. P.	Steam Power. H. P.	Gas Power. H. P.
Laugenverdampf-System, L. Kaufmann-Aachen	Verein für Chem. Industrie, Frankfurt a. M. Werke: Schenkenzell (Schwarzwald)	.....	.....	.....	1,000	.....	.....
Siemens & Halske, Berlin, and F. Schichau, Elbing	Vereinigte Chemische Fabriken; A. G. Leopoldshall	.....	250	.....	.....	250	.....
ENGLAND.							
Castner-Kellner	Castner-Kellner Alkali Co., Runcorn, Eng	.....	4,000	.....	.....	2,000	.....
Rhodin	Commercial Development Corporation, Clifton Junc., Manchester	.....	.....	.....	.....	.....	.....
Richardson & Holland	Electro-Chemical Co., Ltd., St. Helens, Lancashire	.....	5,000	.....	.....	.....	.....
Hargreaves & Bird	General Electrolytic Parent Co., Widnes, Lancashire	.....	.....	20	.....	.....	.....
Hargreaves & Bird	Electrolytic Alkali Co., Middlewich, Cheshire	.....	2,000	.....	.....	.....	.....
FRANCE.							
	A. G. Elektron, Frankfurt a. M., Werke: La Motte-Brun	3,000	.....	.....	3,000	.....	.....
	Fives-Lille, Werke zu Moutiers	6,000	.....	.....	.....	.....	.....
Hargreaves & Bird	Société Electrochemique, St. Gobain	.....	.....	.....	.....	.....	.....
Elektricitäts A. G. vorm. Schuckert & Co., Nuremberg	Société Industrielle de Produits Chimiques, Paris	.....	.....	.....	.....	1,800	.....
	Société Soudieres electrolytique. Werk, Gavet	6,000	.....	.....	3,000	.....	.....
	Société Volta; Werk, Moutiers	12,000	.....	.....	.....	.....	.....
Maschinen Fabrik, Örlikon	Corbin, Berges & Co., Chedde, Hoch-Savoyen	12,000	.....	.....	8,000	.....	.....
Société d'Electrochimie. Paris (Gall & Montlaur)	Société d'Electrochimie; Works at St. Michel	4,500	.....	.....	4,500	.....	.....
AUSTRIA.							
Siemens & Halske, Wein (Kellner)	Consortium für elektrochemische Industrie, Golling	4,500	.....	.....	200	.....	.....
Elektricitäts A.G. vorm W. Lahmeyer & Co., Frankfurt a. M.	Oesterr. Soda Fabrik, Honschau	.....	.....	.....	.....	.....	.....
Elektricitäts A.G. vorm. Schuckert & Co., Nuremberg	Bosnische Elektricitäts A. G. Jajce, Bosnien	9,500	.....	.....	1,500	.....	.....
RUSSIA.							
Electrochemische Werke, Bitterfeld	A. G. Elektricitäts, Warschau. Works at Zombkowitz	.....	1,500	.....	.....	1,500	.....
SWEDEN.							
Gall & Montlaur	Superphosphat A. G. Mansboe	4,000	.....	.....	.....	.....	.....
SWITZERLAND.							
Maschinen Fabrik, Örlikon	Gesellschaft für Elektrochemische Industrie, Turgi, Aargan	700	.....	.....	500	.....	.....
Société d'Electro-Chimie (Gall & Montlaur	Société d'Electrochimie. Works, Vallorbes	3,000	.....	.....	3,000	.....	.....
Verdampfanlagen	La Volta Suisse Vernier	.....	.....	.....	1,000	.....	.....
L. Kaufmann, Aachen	Usine des Produits Chemiques, Monthey	.....	.....	.....	1,200	.....	.....
SPAIN.							
Elektricitäts A. G. Schuckert & Co., Nuremberg	Sociedad Electro-quimica de Flix, Barcelona	.....	.....	.....	2,100	.....	.....

STONE.

UNDER this caption are included the chief varieties of stone produced in the United States: granite, limestone (including marble and onyx), sandstone, and road metal. This last subdivision is based on a special use for stone and is not a variety itself. The subject of good roads, however, is one of such growing importance that it merits a place in the discussion of stones in general. The variety slate is discussed under a separate caption elsewhere in this volume.

It is practically impossible to collect statistics of the various branches of the stone industry which shall represent the exact conditions of the industry, a large production being made by individuals who keep no record of their work.

The value of the total production of stone in the United States (not including slate and sandstone used for manufacture of grindstones, which is reported elsewhere) was approximately $38,797,487 for 1899, and $35,404,900 for 1898, divided as follows; the values for 1898 are given in brackets. Granite, $10,200,000 ($9,500,000); limestone for building, $14,600,000 ($14,000,000); limestone for flux, $3,197,487 ($2,304,900); marble, $4,200,000 ($4,000,000); sandstone, $5,600,000 ($4,800,000); bluestone, $1,000,000 ($800,000).

THE STONE INDUSTRY.

BY CHARLES L. WHITTLE.

IN the preparation of this review, I wish to acknowledge my indebtedness to Wm. Beals, Jr., for the manuscript relating to Maine, Massachusetts, New Hampshire, Rhode Island and Vermont. I have freely made use of the publications of the United States Geological Survey, State Geological Reports and several periodicals of the stone trade. These periodicals are: *The Stone Trade News and Building Journal, National Granite Cutters' Journal, Granite, The Monumental News,* and *Stone.* I wish to express my obligations to Profs. George E. Ladd, William B. Clark, Fred. D. Smith, J. A. Holmes, A. J. Collier, J. E. Todd and Wilbur C. Knight for many useful notes, and also to Messrs. Henry A. Lee, W. S. Blatchley, H. Foster Bain, W. M. Courtis, R. A. Shifleff, H. B. Kümmel, S. W. McCallie and F. J. H. Merrill.

The year 1899 has witnessed an activity in the business of stone quarrying and stone trade which has permeated all branches of the industry, a renewal

which has been general over all parts of the United States. This favorable condition is not confined to a demand for any particular type of stone, and every State where stone is quarried or treated, is receiving its proportion of the increase. With the almost universal revival of business throughout the United States, this was to be expected, as nearly all lines of industrial activity are interdependent, and the common remark that prevailing conditions in the business world are reflected in this or that industry applies equally well to the stone trade. The great demand for stone has not only resulted in increased production of old quarries, but a large number of abandoned quarries have been reopened, and many new stones have been exploited.

On account of the increased demand for stone, many quarries were opened much earlier in the spring than usual and were kept in operation well into the following winter season. Reports are prevalent from many parts of the country of putting on two quarry shifts and working night as well as day, and this is true of finishing work also. The middle West is beginning to feel the influence of the spreading demand for macadam and telford roads, and the installation of crushing plants by towns and cities and by individuals for custom trade in that part of the country, reflects this new condition. The unusually large number of important Federal buildings and other extensive governmental undertakings, as dry docks and jetties, are calling for immense quantities of the best class of building stones. Railroads are taking advantage of increased earnings by stone ballasting their road beds and doing other work calling for the product of the quarry. Great activity in the construction of new buildings furnishes a steady demand for dimension stock, and also stimulates the limestone and cement quarries. There have been over 270 new quarries opened, old ones reopened, and new companies incorporated in the United States during 1899 to carry on quarrying or a general stone business, not taking into account the large number of individuals who have opened quarries of which there is no public record. This increase is greatest in New York, Illinois and Pennsylvania. Reports are widespread that this activity will increase during 1900.

Stone quarrying and stone finishing works have been beset with perhaps the usual number of labor troubles, but it is to be noted that disorders of this kind are confined in the main to the New England States, or have their origin there. This condition of affairs is owing to the fact that New England, with its long-entrenched and numerous quarries and consequent large number of employees, possesses labor organizations which act as a unit when differences between capital and labor arise. Between the large number of producers in New England, competition is more severe than in any other part of this country—a condition of things very likely to produce a conflict of capital and labor.

These conditions have been reflected in the efforts on the part of many quarriers in the two large centers of the granite industry—Quincy, Mass., and Barre, Vt.—to conform to the prevailing policy in the country at this time among many other industries, of trade combination or "trusts" in order to cheapen production and other expenses incident to the quarrying, manufacture and distribution of stone. While all endeavors of this nature have met with failure among the granite producers thus far, it is more than probable that at no distant day the very large

number of separate concerns now in operation will be very materially lessened by consolidation, although it is doubtful if stone trusts, controlling the price throughout the country, will be organized for some time to come. If such amalgamations shall be organized, and there are indications that a certain amount of consolidation will be deemed necessary by the granite manufacturers, the marble industry will naturally be the first one to be operated under one management on account of the limited number of quarries of this stone. The granite quarries are too widely scattered to be successfully operated under the proposed plan, especially when consideration is taken of the enormous capital invested in this industry, and the wide distribution of quarries and of rocks of this class. Outside of New England, New York and Pennsylvania, the new stone centers, each having a narrower market, so far as population is concerned, are as yet little effected by the keenness of competition common to districts whose natural resources have reached very nearly their maximum development, with the result that capital has little temptation to combine, and for years to come will not join in an attempt to control production and dictate prices. The Granite Cutters' National Union, which is seeking to establish a maximum eight-hour law and a minimum $3 wage scale throughout the country, beginning March 1, 1900, is using the mighty power of united labor to bring about conditions (if it shall be successful) which will be one of the strongest factors tending toward a consolidation of the quarry interests and other departments of the business. When the eight-hour law and uniform wage scale shall have been secured, solving of present difficulties will give place to the suffering brought about by idle plants and discharged employees incident to the economies which will be practiced by the so-called trusts. The business of granite cutting during 1900 promises to be beset with serious troubles which will affect all industries dependent on the use of stone. There are numerous indications that granite manufacturers will not unitedly agree to the eight-hour day and the minimum wage scale proposed by the union, and strikes of formidable dimensions and far-reaching in their effects are anticipated. With the fine trade prospects for 1900 over the entire country, it is to be regretted that the demands of the granite cutters should be pressed at this time to the detriment of the stone industry. The best view of the situation is that the labor-saving devices, which have been perfected during the last few years, may have so cheapened quarry and finishing methods that manufacturers may still be able to carry on business at a profit, when stone is in such great demand. If this shall prove to be the case, the prospective difficulties anticipated for 1900 will be postponed until the present activity shall have disappeared.

The eastern part of the United States is suffering somewhat, owing to the gradual development of the stone resources in the middle West and in the South.

Not a little of the increased activity in the stone industry is due to the building of telford and macadam roads in many States of the Union. States and individuals desiring the most available and suitable stones for road purposes have furnished them the incentive to investigate the ledges in their neighborhood—a process which is developing not only our road metal resources but which is bringing to light many stones for higher uses.

GRANITE.

Under this division are naturally grouped those igneous rocks whose physical properties approximate those of granite and whose use is similar. Very nearly one-half of the total output produced in the United States comes from Massachusetts. The value of the granite production in the United States in 1899 was approximately $10,200,000, against $9,500,000 in 1898.

About 60% of the output is used for construction purposes of all kinds, including bridges, retaining walls and similar structures. For these purposes it is used both in the rough and in the cut condition. About 25% is consumed in the construction and maintenance of highways. The balance is taken up for monumental and cemetery purposes. The progress of the granite industry during 1899 in the principle producing States is summarized as follows:

Alabama.—Granite occurring in the eastern part of the State, is used for rough work locally as well as in some buildings. Among the resources of the State are occurrences of gneiss and diorite, which will prove of value in the future as road metal and to some extent as building stone.

Arizona.—During 1899 granite quarries were opened in the San Rita Mountains, and the stone was shipped to Tucson, and there finished.

Arkansas.—Limited quantities of so-called granite are quarried. In the "blue granite," "gray granite," and "brown granite" the State produces stones which are there quarried exclusively. Blue granite is known scientifically as Pulaskite, which is an eleolite hornblende-syenite. Investigation of this stone shows a crushing strength abnormally high for a building stone, ranging from 26,745 to 34,964 lb. to the sq. in. It is generally harder to work than stones of its class, due to greater compactness and the absence of the brittle mineral quartz. Pulaskite is an important economic stone in Arkansas and is used for general building purposes, paving and as road metal both within and without the State. This stone is quarried mainly on Fourche Mountain, about 3 miles south of Little Rock. "Gray granite," or eleolite syenite, is another unique building stone quarried here, coming from Magnet Cove. It differs from the blue granite principally in the absence of hornblende. A brown phase of this rock is known as "brown granite."

It will be of interest to watch the weathering and other qualities of these stones under the various conditions they will encounter if they shall become widely used. These stones have many good features and are likely to attain considerable importance in a portion of the country where granitic rocks are not abundant.

California.—During 1899 the granite industry in particular has been buoyant, and it is confidently anticipated that renewed industrial activity, caused by the boom in mining, the discovery of petroleum, and the business expansion due to our new trade policy, will be experienced, not only by California, but by Oregon and Washington as well. The output during the past year was 128,924 cu. ft., valued at $141,070, as compared with 98,369 cu. ft. ($147,732) for 1898.

The Raymond granite from Madera County is being quarried for the San Francisco Post-office. Immense quantities of stone are taken from the Sweetwater quarries for the Coronado jetty. From all parts of the State business is reported good, and many new quarries are being opened up.

Granite is often worked in this State under difficulties not thought worth overcoming in the East. The Rocklin quarry, for example, is worked like many marble quarries; sinking is resorted to and on the vertical sides of a large opening, 100 ft. deep, the men work on platforms suspended by ropes, like house painters. On the other hand the Raymond quarry has opportunities not often seen for successful quarrying. In 1893, the bed being worked was nearly horizontal, 5 to 15 ft. in thickness, and extended unbrokenly for 325 ft. There are about 40 granite quarries in the State worked intermittently. There is a very large number of quarries getting out paving for the large cities. The stone used comes mostly from Sonoma County, and is a trachyte. This use of trachyte is unique in the United States. Paving stones are giving place to asphalt in the large cities. A use of trachyte and trachyte-tuff for ashlar is to be noted in Napa County, which keeps five quarries in operation interruptedly.

Colorado.—Gray granite is quarried near Gunnison; Platte Cañon furnishes a pink stone, and the quarries at Cotopaxi can supply a light gray granite in unlimited quantities.

Connecticut.—Norcross Stony Creek granite, which is a flesh-colored stone, coarsely ribbed with pegmatite bands as well as biotite streaks and patches, has recently been used to a considerable extent in Boston.

Delaware.—This State is only a producer of granite or closely related rocks classed as such. In 1890 the value of this stone amounted to $211,194, which rose suddenly to $677,754 in 1898 due to breakwater construction. Two new companies were incorporated in 1899 to quarry and deal in stone. Delaware is perhaps the first State to appear on the list as an exporter of crushed stone. The Delaware Granite Mining Co. is the lowest bidder in the work of furnishing the material for Puerto Rico highways, United States Government undertaking. Bridge stone will also be supplied by this company.

Georgia.—The granites of Georgia are now being investigated by the State Geological Survey. Nearly all counties in the crystalline area have more or less extensive areas of granite, and the industry has assumed large proportions in the Stone Mountain area at Elberton, Greenville, and Lithonia. Stone Mountain granite has a high crushing strength, 25,680 to 28,130 lb. per sq. in., while Elberton granite runs up to 30,320 lb. per sq. in. Georgia granites find a market all over the South and middle West.

Georgia is well favored in its supply of high-grade road metal, and the resources of the State in this regard have been investigated by the State Geological Survey, which will issue a report covering the subject at an early date. These road stones are granite, gneisses, trap rock (diabase) diorites, and other hornblendic rocks. Diabase occurs as dikes from a few feet to many rods in width. Some may be traced almost continuously for 40 miles or more. The trap is fine-grained and well-suited to the needs of modern highway construction.

Idaho.—The year 1899 witnessed some activity in the granite business: a new quarry has been opened on the Grand Round River and the Platt Cañon quarries will furnish the stone for the new Federal building in Boise. There is no State in the Union supplied with a larger quantity of granite than Idaho. The quartz diorite of the Seven Devils copper region, were it near a market, would afford a

beautiful building and monumental stone. The opening of the Pacific and Idaho Northern Railroad, now in process of construction, will furnish transportation facilities for this material.

About Weiser, the local trachyte-tuff is employed as a building stone.

Maine.—The principal granite area of Maine extends in a broad belt clear across the State, following in general the coast line. The great proportion of the quarries are within a short distance of the seaboard. The main producing counties are Knox, Lincoln, Hancock, Waldo, Kennebec, Washington, York and Franklin.

The quarries in Knox and Lincoln counties are among the largest and best equipped in the country. Most of the plants are working on a medium-grained biotite granite, ranging in color from light to dark gray. It has a good rift and grain and quarrying is assisted by a generally well-developed system of horizontal joints. The stone is cheaply extracted and large blocks may be obtained when occasion requires. The quarries in this section are best suited to furnish building stock, or material for foundations, piers or pavings. The granite occurring in Washington County, in the eastern part of the State, is a biotite granite with pink feldspars, which is well adapted for polished and interior work. The stone has a very even texture and takes an excellent polish, and is superior to the celebrated Scotch granites of this class. It is not surpassed by any red or pink granite now quarried in this country. The plants here are being worked to their full capacity, and business has not been so good at any time the past ten years.

The granite from Hallowell is a fine-grained, even-textured gray granite, and is deservedly popular as a superior building stone, as well as a monumental one. A similar granite is quarried at North Jay. This stone, as well as Hallowell, is well adapted for carved work and other elaborations.

Labor troubles the past year have caused no serious curtailment of the granite product except at Mt. Waldo, where a large force of men was engaged upon the work of furnishing 600,000 cu. ft. of stone for the Federal building at Chicago There was a strike of the workmen belonging to the National Granite Cutters' Union, who were discharged and Italian workmen were engaged in their place. The workmanship of the latter was unsatisfactory, and a compromise with the Americans was made, which, however, was only temporary, and the Italians were reinstated.

Maryland.—The McClenahan & Brothers Granite Co. has opened a quarry in granite on the Northern Central Railway, in order to furnish stone ballast for the Pennsylvania road. This plant is turning out 500 tons per day.

Massachusetts.—This State is pre-eminently a granite producer, due to its immense and varied granite resources, and can furnish stone from the highest grade, suitable for monumental stock, down to material used for paving or road metal. In addition, the principal centers of production have been long established and are fully equipped to meet all demands made upon them. Many of the quarries are situated to take advantage of cheap transportation: notably the plants at Cape Ann in the N.E. part of the State. A recent discovery of granite at Endicott is now being exploited that has attracted some attention, owing primarily to its unique color merits. It is a medium-grained, even-textured granite, of

light brown color, with reddish feldspars giving a faint flesh-colored tint to the stone. The granite closely resembles the Dedham porphyry used in the building of Trinity Church, Boston. Relatively small quantities of biotite and hornblende are present; there is, however, a little epidote and scanty amounts of magnetite. The stone is favorably situated for railroad transportation.

Michigan.—The granite and serpentine quarries were closed during 1899.

Minnesota.—This State is supplying native granite in part for its new capitol at St. Paul, the stone for the upper three stories coming from Georgia, where quarries are operated by Minnesota capitalists. Ortonville reports four firms actively engaged in granite quarrying. The quarry at Fountain City is busy supplying stone for the United States Government for river improvements, and Winona stone is being extensively used in Tomah, Wis., for bridge work, curbing, and sidewalk purposes.

Montana.—The most prominent quarry is at Baxendale, where a porphyritic, hornblende-biotite granite, or granitite is worked to some extent. This stone is unusually fresh, as shown by the translucency of its feldspars, and will prove a very satisfactory building material. The total amount of granite quarried in 1899 was valued at $10,022.

New Hampshire.—The granite producing section of this State is confined for the most part to the southern and middle portions. The New Hampshire granite is pre-eminently a building stone. It is commonly fine-grained, and of a light gray color. Its essential constituents are clear granular quartz, pure white orthoclase feldspar, and both muscovite and biotite micas in small flakes. An almost total absence of pyrite preserves it from weather discolorations. The State has a number of well equipped plants, and there is a growing demand for monumental stock. The best use of the stone, however, is for buildings. Within a few years a large business has been developed in Carroll County. The stone occurring here is a coarse to medium-grained, biotite hornblende-granitite. The prevailing color is a soft pink, although some phases show a grayish-green effect, and the stone has gained great favor with architects. Structurally it is a strong stone, having a compression test of 22,370 lb. per sq. in., and should be by its composition very enduring, and should prove a successful competitor of the well-known granite occurring at Milford, Mass. Seven companies have been added this year to the list of granite producers of the State.

New York.—In the Adirondack area of the pre-Cambrian rocks of northern New York, there are inexhaustible quantities of norites (a variety of gabbro) which deserve exploitation as building stones. This gabbro type of stone presents many phases both as regards color and grain. Two varieties are prominent; one is very dark, gray-green, and medium coarse-grained, with many slightly porphyritic areas of labradorite feldspar, showing, particularly when polished, a brilliant play of colors, due to the refractive effect of this mineral on transmitted light. This stone is especially adapted for monumental and interior work, as well as for general building purposes. It possesses a crushing strength exceeding 29,000 lb. per sq. in. The other prominent variety is a fine-grained, gray-green stone known as anorthosite, made up almost entirely of feldspar with some quartz and scattered minute red garnets. This phase is well adapted for

monumental and ashlar work. Both stones take a good polish and are admirably suited for purposes where great compressive strains are to be withstood, as well as for road metal.

North Carolina.—This State is well supplied with granite areas, and the condition of the industry is the best ever known. All the quarries are actively worked and the prospects for 1900 are excellent. Prominent granite quarries are found in Rowan, Wilkes, Surrey and Iredell counties, and their outputs are becoming widely known throughout the South.

Oregon.—The most important quarry in operation is located at Morrison. During 1899 this quarry furnished 1,000 tons of building material and 2,000 tons of rip-rap for railroad use. Part of the output went to San Francisco.

Rhode Island.—This State leads in the production of granite for ornamental and monumental purposes. The principal quarries are situated at Westerly and Niantic, and furnish practically all of the monumental stock produced. Westerly granite has marked advantages for this class of work; it is a fine-grained, even-textured biotite granite, of a dark gray color, almost a blue tint in some quarries, and can be dressed to a fine edge without fear of "plucking." It is a strong stone, having a compressive test of 17,750 lb. per sq. in., and is especially adapted for elaborate work on large blocks.

The occurrence of "knots" and "streaks" in Westerly granite is not uncommon, and evidences of these disfigurations are not always present until considerable labor has been put on the stone. The manufacturers allow only high grade stock to leave the shops. Although this entails loss, it has served to maintain a high and well-deserved reputation. Special machines are used here for turning and sawing granite. In securing blocks of monumental stock considerable waste accumulates, which is sold as rip-rap or split into paving blocks. Since 1896 there has been an increase in granite for building purposes in Westerly and Niantic, and some stone has been quarried at Centredale and Greenville for foundations and street work. Pascoag furnishes considerable biotite gneiss for paving blocks.

South Carolina.—A new company has been chartered to work the gray and blue granites of the Moody Hills, Supo, Rocky Creek, Brick Church and Guignard areas. Other districts producing granite are Edgefield and Pittsfield, Edgefield County. A new granite is also being quarried at Winsboro.

South Dakota.—South Dakota is reported to be a producer of granite, but the stone quarried under this name is really a highly cemented sandstone, or quartzite identical with that quarried in Minnesota, where it is known as "jasper."

Texas.—Large quantities of crushed granite from Granite Mountain are finding a market in New Orleans for macadam.

Vermont.—This State stands second only to Massachusetts in the production of granite. The great increase in production is due to developments at Barre and surrounding towns, where there are immense quantities of granite suited for all requirements of both ornamental and building work. Nearly 70% of the output from the State is obtained from this section. The quarries at Barre and vicinity are finely adapted to produce enormous amounts of superior quality

of monumental and building stock at a small cost. The location of many quarries with regard to transportation to the railroad could not be bettered, and advantage is taken of the latest methods of quarrying and most improved machinery. The stone known as Barre granite is a medium to fine-grained gray granite, lean in muscovite (white mica). The stone carries but a small percentage of iron, thus insuring freedom from discoloration upon exposure to the weather. There are three grades of this granite commercially known as "dark," "medium" and "light" Barre, but the stone of each class presents great uniformity both in color and grain. Much of the granite occurs in more or less regular sheets, noticeably in those quarries producing the light Barre. Under such conditions channelling machines are used to a great extent. This process of quarrying, although expensive, has met with favor on account of its insuring so little waste. In other quarries, especially where the "dark Barre" is produced, much powder is used. The Knox system of blasting is well adapted for such conditions, and has met with pronounced success in many cases.

The use of granite for monuments is steadily increasing and is becoming a serious competitor of marble for this purpose. This is in a great measure due to the recent improvements in machinery and new appliances for cutting and polishing granite, which lower the cost appreciably. In addition to this is the lower price at which rough granite suitable for monumental work can be bought ranging from 70c. to $2.50 per cu. ft., as contrasted with $2.25 to $12 per cu. ft. for the better grades of marble. On the other hand, granite is not so well adapted for elaborate work, and we may look to the continued use of marble in monuments where fine carving is desired.

The compression tests of Barre granite, although not very extensive, perhaps because of the fact that the stone is so well known, are generally low for this class of material, the crushing strength for the dark being something over 17,000 lb. per sq. in.; that for the light being a trifle less. This, however, is ample strength for building purposes. One of the Hardwick companies has recently put in a complete set of machinery for its finishing works, using electricity as the motive power. A large granite area two miles south of Groton has been exploited, and a number of firms are now engaged in the business. The granite lies in sheets averaging 11 ft. thick, and the exposures extend for three-quarters of a mile in length. The facilities for cheap extraction are excellent. The stone is similar in many respects to Barre. Fifteen new concerns have been organized the past year in this State. Of these 11 are engaged in quarrying granite.

Virginia.—Granite comes principally from Petersburg, Richmond, Manchester and Fredericksburg. The construction of a sea wall at Annapolis Academy, and the building of a railroad station in Richmond increased granite trade in Virginia during 1899. A new granite quarry is being opened near Martin's Ferry.

Washington.—The only granite comes from Granite Point on the Snake River, above Riparia. This stone, which is now being used for the custom house building in Portland, is a somewhat porphyritic gneiss with appreciable quantities of biotite mica and hornblende. The rock bears evidence of having been derived from a granite by dynamic agencies; both its quartz and feldspar are in

a partially sugared or granulated condition, and the stone will accordingly have a low crushing strength and a high ratio of absorption.

Wisconsin.—The granites of this State are unique in their range of color, and also in their strength tests. The Montello granite has the enviable quality of being the strongest granite known, having a maximum compression test of 43,973 lb. per sq. in., and a modulus of rupture of transverse strain, amounting to 3,793·8 lb. per sq. in. There are many other granites in the State, much above the strength of the average stones of this kind.

The igneous rocks of Wisconsin combined, including granite, rhyolites, diabase, gabbros, and diorite, are all of pre-Cambrian age and occur in the Archæan and Keeweenawan Series. Among the stones wrought and classed as granites are many purplish-black to black rhyolites. These are fine-grained rocks often having small, porphyritic crystals of pink or red feldspar arranged in planes due to the flowage of the rock. This stone has a crushing strength the greatest of any building material known, namely 47,674 lb. per sq. in.

Rhyolite is composed essentially of quartz and feldspar, is very fine-grained and exceedingly hard to work in comparison to most stones. It is shown in this State that rhyolites can be successfully quarried and worked for various economical purposes, and, as this is the first time such stones have been exploited on a commercial scale in this country, the lesson taught is very valuable. The rhyolite breccias found in many places along the Atlantic Coast afford stone of rare and unusual beauty, which would furnish a material particularly adapted for interior, mosaic or tile work. Wisconsin granites and rhyolites are widely known, and find a market for monumental purposes in many parts of the United States.

Wyoming.—The Union Pacific Railroad uses a large quantity of disintegrated granite from the Rocky Mountain divide on this road at Sherman, Wyoming. This material is used for railroad ballast and for highways, and is shipped to many middle Western points outside of the State.

LIMESTONE.

The varieties of limestone, representing many conditions of physical structure and chemical composition, are employed in numerous ways. The massive varieties and dolomite furnish excellent material for structural purposes usually in a dressed condition. The crushed stone, while inferior to most rocks, is also used for making roads. The purer varieties of limestone, free from silica and magnesia, are very extensively employed as a flux in smelting ores of iron, copper and lead, and find an additional use for the manufacture of lime* and cement. Crystalline limestone or marble is also employed in the manufacture of lime. As a building material it is more valuable than the fine grained varieties. Black and variegated marbles are highly prized for their artistic effect in interior work, and often approach in value the onyx marbles. The best variety for ornamental purposes is the onyx marble which is deposited

* Reference should be made to the very complete article on "Lime Burning," by Hans A. Frasch, in THE MINERAL INDUSTRY, Vol. VII.

from an aqueous solution in different colored layers, forming a beautiful stone that can be utilized very advantageously to produce ornamental effects. The variety of limestone called lithographic stone has for many years been indispensable in lithography; of late this stone has been replaced to a considerable extent by aluminum in plates which offer the special advantage that the plate after use is of value.

The production of limestones for building in the United States in 1899 was valued approximately at $14,600,000, against $14,000,000 in 1898, while the approximate values for limestone used as flux were for 1899 and 1898 respectively $3,197,487 and $2,304,900.

Alabama.—At the present time, the greater part of the limestones and dolomites quarried is used for lime and for fluxing purposes and has kept pace with the growth of the iron industries, which use it as a flux. The Government quarry supplies large quantities of limestone to the Colbert Shoals Canal, near Riverton.

California.—There are many quarries where limestone is extracted for fluxing purposes and lime.

Illinois.—One-half of the limestone produced in this State is used for building stone, coming from Joliet and Lamont, one-third for paving and road making and the remainder is used for flux.

The stone cutters of Chicago struck against the use of machines in cutting stone, and their demand was allowed. Crushed stone from the Kankakee quarries has been advanced from 75 to 85c. per yd. New quarries have been opened near Makanda, Momence, Gilman, and Hamilton. The Hamilton quarry has been opened by the Toledo, Peoria and Western Railway, which proposes to sell the products to towns along the line for macadam. Ten new companies have been incorporated in the State to engage in quarrying and other branches of the trade.

Indiana.—This State occupies first position as a producer of limestone for building purposes. Most of this comes from the Bedford oölitic beds. The stone is used all over the United States, and is particularly adapted for carving and ashlar work. A new quarry has been opened near Harrodsville.

Iowa.—For the last decade quarrying has been done almost exclusively in the dolomites and limestones, and to a very limited extent in the sandstones. One-half of the total output of limestone is used in building, one-fourth is burned into lime, and the remainder is devoted to road making, as paving or surfacing. The quarry industry, while small, has not suffered to the extent common to most of the States during the last ten years. In 1895 and 1896 an attempt was made to quarry the Lower Carboniferous, or Hardin County marble, the production reaching $37,740 in the latter year, but the undertaking was abandoned.

The limestones quarried are from the middle Niagara dolomite, which are burned mostly for lime; and the upper Niagara stone, which is of a grayish to cream-yellow stone, is used for building purposes.

The stone wrought is sold almost exclusively within the State limits. Quarry practices are very primitive in the majority of the plants. A few of the larger quarries at Cedar Valley, Anamosa and Marshalltown use channellers, although the stone is customarily loosened by crowbars or by blasting. Railway bridge

construction is now calling for a great deal of the galena limestone from near Dubuque.

Kansas.—The Sub-Carboniferous limestones are extensively employed for building purposes, road making, rip-rap and lime. Much of the stone used in the larger towns comes from Missouri. The output of all kinds of building stone in 1899 is estimated by the State Geologist at $250,000. There was also produced 400,000 tons of railroad ballast valued at $300,000, and lime to the value of $65,000.

Kentucky.—Considerable limestone is quarried which is used mostly for road material, and somewhat for lime and general building. Limestone suitable for building purposes is widely distributed over the State, and only awaits development. The Upper Silurian limestone is an important stone which is used somewhat for general construction.

Maine.—Almost the entire output of limestone comes from Knox and Waldo counties, the former furnishing the greater part of the product. This State ranks first in the valuation of the lime produced.

There is an inexhaustible quantity of limestone at and about Rockland which furnishes a very pure white lime, which has been accepted for many years as the standard. The stone is hoisted to the surface of the ground, and thence cheaply transported by cables and other improved steam-driven conveyances to the kilns. The wood for burning is procured from Canada.

Labor troubles have been prevalent at the quarries since April, 1898, owing to a differentiation of wages between Rockland and the adjoining towns, Thomaston and Rockport. This has served to cripple the output to some extent. The lime industry of this section has recently been acquired by a syndicate which controls all the best quarries and plants for burning lime at Rockland and vicinity. A cessation of labor troubles may now be looked for, together with a lessening of the cost of the product.

Michigan.—The largest quarries in this State are operated for limestone for the alkali works at Trenton and Bellevue, and during the year shipments of limestone have been made to the United Alkali Co. of Widnes, England, to be used in glass making. A new stone quarry has been opened at Omer.

Missouri.—A large part of the limestone quarried for building purposes comes from Carthage. The limestone quarries in the vicinity of St. Louis furnish immense quantities of stone used for macadam, some dimension stock, local material for foundations, and limestone for burning.

The so-called saccharoidal sandstone, used for glass making, which is quarried at Pacific and Crystal cities, has furnished the raw material for this important industry in Missouri. The works at Crystal City have been closed down by the plate glass trust and 2,000 people have been thrown out of employment.

Montana.—Limestone has furnished the bulk of the stone quarried in the past. It is used mainly for fluxing purposes. In 1899 there was produced 3,839 short tons of lime valued at $21,897.

Nebraska.—A large quantity of limestone is used for building and road making. The Barton-Nash Stone Co. of Omaha has made a two-years' contract to supply crushed stone to the Chicago, Rock Island and Pacific Railway from the

company's quarry at Coralville. The stone will be used for railroad ballast, and over 65,000 yd. will be required. Several new quarries are undergoing development.

New York.—Limestone is used for constructional purposes of all kinds, for road making, and for lime burning, and is quarried from the Calciferous, Chazy, Black River, Trenton, Niagara, Lower and Upper Helderberg, Corniferous and Tully formations. The dark blue to gray Trenton stone is quarried near Glens Falls, Canajoharie, and all through the Mohawk and St. Lawrence valleys, as well as in many other localities. Niagara limestones are mostly used for foundation work, and somewhat for general building. Lower Helderberg limestone is used mostly for masonry, while the aggregate value of the Upper Helderberg stone quarried is greater than any other formation in the State. There are over 300 active limestone and marble quarries in the State.

Ohio.—Two new quarries have been opened at Ganettsville, and the Brokensword Co., of Bucyrus, has been incorporated to quarry limestone.

South Dakota.—Limestone is quarried to supply the demand for flux at the Black Hills mining district, and to a limited extent for building purposes.

Tennessee.—The greater part of the limestone quarried is burned for lime or used for blast-furnace flux.

Vermont.—The limestone product of this State comes principally from Addison and Franklin counties; other productive counties are Chittenden, Windham and Windsor. Practically all the limestone quarried is converted into lime of very excellent quality. A number of analyses give over 98·50% CaO.

Virginia.—Limestone has been extensively quarried for use as a blast-furnace flux.

West Virginia.—Considerable limestone is quarried, which is mostly burned into lime.

Wisconsin.—Limestones show strength tests of good record, *viz.*, 42,787 lb., and several other limestones running up to 33,400 or even 40,000 lb. per sq. in. While such strength is not essential for most building purposes, the stones are peculiarly adapted to use where excessive strain is to be withstood.

MARBLE.—The production of marble in 1899 was valued approximately at $4,200,000, against $4,000,000 in 1898.

Arizona.—During 1899 marble quarries have been opened in the Santa Rita Mountains, and the stone brought to Tucson and there finished. The marbles are reported to be of excellent quality, having a color range of white, black, blue with mottled effects, and occurring under conditions which will allow the extraction of blocks of any size.

Colorado.—Marble will soon be a regular production in this State. On Yule Creek, marble has been recently discovered, and a railroad 31 miles in length is being constructed to afford transportation. The Cañon City Marble Co. is also opening up quarries in this stone. The occurrence of true marble in this part of the United States is not known to be widespread.

Florida.—The output of this State consists of Coquina limestone, used for road making, rip-rap and lime making.

Georgia.—Most of the marble comes from Pickens County in the vicinity of

Tate. There are four main quarries in operation: the Georgia, Southern, Herndon, and Piedmont. The output of these quarries has increased during the past year with the exception of the Piedmont. Georgia marbles are widely known in this country and deservedly so, as they are both durable and handsome.

Georgia marble is known to the trade as "Kennesaw," "Etowah," "Creole," "Cherokee," "Piedmont," and "Southern." Besides the coarse, white marbles, there is the well-known Creole, which is white, with bands and random areas of dark gray to black, due to small quantities of foliated graphite. The Etowah is a pink phase of great beauty, colored with ferric oxide, and having many impurities in the shape of various silicates. Crushing strength averages from 14,000 to 15,000 lb. per sq. in. Most Georgia marbles are dolomitic without being true dolomites.

Maryland.—In the Potomac marble or Potomac breccia is found perhaps the most beautiful marble exploited for interior work known in the United States to-day. Potomac marble occurs in the Triassic formation, and the development is not great as compared with several marble quarries in the State, but a sufficient quantity of the stone no doubt occurs to supply a large demand. The following description of this stone appears in the Maryland Geological Survey, Vol. II., p. 191: "The conglomerate (or breccia) is made of pebbles of limestone of varying size, which sometimes reach a foot in diameter, although usually averaging about 2 or 3 in. The fragments, which are both rounded and angular, range in color from gray to blue and dark blue, and occasionally pebbles of quartz, chloritic schists and white crystalline marble occur. All are embedded in a red calcareous matrix mixed with a greater or less amount of sand."

Massachusetts.—A number of new firms have started in business during 1899. Several new deposits are being exploited, and long abandoned and partially developed quarries have been reopened. The marble is principally used for outside work on buildings.

There are a number of quarries in the western part of the State working on limestones, or, more properly speaking, dolomites. Inconsiderable amounts are used for the foundations of buildings, and the bulk of the product is burnt into lime for local consumption.

Montana.—Marble is quarried by Beaudette Bros., at Dempsey Creek, 12 miles from Anaconda. The quarry is equipped with channelling machines, and the stone is cut to the desired size by gang saws. This marble has a crushing strength of 12,500 lb. to the sq. in., and is essentially a pure lime carbonate with slight amounts of silica and magnesium. The stone is fine-grained, white to faint blue in color, and is somewhat banded with graphite.

New York.—Marble is quarried in two principal areas: from pre-Cambrian of St. Lawrence County, notably at Gouverneur, in the northern part of the State, and from Westchester County, near New York City. The marbles of the latter area are really coarse dolomites of Calciferous-Trenton age. The Westchester County marble is extensively exploited at Tuckahoe, and the product is widely known throughout the Atlantic Coast States. The gray to dark blue marbles of Gouverneur find a market largely for monuments and building purposes mainly in Western markets.

Associated with the norites, 6 miles north of Westport, which make the bold bluffs occurring along the west shore of Lake Champlain at this point, are beds of marble that are unique types of this kind of stone. The marble is medium coarse-grained, light gray in color, and is composed essentially of distinct grains of calcite, in which are irregularly distributed rounded quartz grains, a slight amount of graphite and several silicates, among which are coccolite and scapolite. The stone dresses almost white and resembles very closely when bush-hammered the well-known Hallowell granite from Maine.

Tennessee.—The marbles come from the Lower Silurian, and the phases occurring are many. General color and color arrangements form the basis of the several varieties. The stone has a high crushing strength, and its ratio of absorption is remarkably low, and while it is essentially a monumental stone, it is also well adapted for interior work. Prominent quarries are located in Blount, Knox, Union and Grainger counties. Several companies have been formed to operate new quarries and to reopen old ones.

Vermont.—From 55 to 75% of the marble output goes into monuments or tombstones, furnishing the larger part of the total production used in the United States for this purpose.

Marble in an unusual degree fulfills the conditions of a perfect building stone, namely: Strength, durability, beauty and first cost. It is strong constructional material, having a crushing strength exceeding 15,000 lb. per sq. in.; it resists fire to a much greater extent than other building stones, enduring a heat which would seriously injure if not destroy sandstone and granite, and its low ratio of absorption adds greatly to the life of the stone.

Reports of the past year show a great increase in all branches of the marble industry. The marble belt borders the western portion of the State and extends for a distance of nearly 150 miles in parts of Bennington, Rutland, Addison and Chittenden counties. Rutland County, however, produces the bulk of the material. The marble here occurs in irregular beds with a steep easterly dip, except in the West Rutland and Proctor quarries, where the beds are nearer vertical. There is a great range of color and often a considerable difference of texture. The pure white variety is very rare; ordinarily there is a gray or blue tint to the stone, due to the presence of very minute particles of graphite. The beds are irregularly banded or veined by alternate layers of blue and white, while in others oftentimes the veining is absent and a mottled effect is produced. In some quarries the marble is distinctly blue. The West Rutland marble is superior to any yet discovered for the finest kinds of work and supplies the great proportion of stock for cemetery uses in the whole United States. The stone is an unusually pure lime carbonate and contains traces only of magnesia, iron or other impurities. The plants for quarrying and manufacturing the product are the best equipped in this country.

The motive power for the works at Proctor is taken from the immense water power on Otter Creek, in close proximity to the quarries. Franklin County has large deposits of variegated marbles metamorphosed from once highly fossiliferous limestone beds. The fossils have been replaced by secondary quartz and calcite. The marble as a rule consists of a dark ground mass, generally of a

red shade, with irregular patches or blotches of a lighter red, gray or white color, producing a variegated effect. Other beds show a network of lines traversing the ground mass. The marble is particularly adapted for polished wainscoting and for tiling.

Very interesting discoveries have been made the past year of marble deposits occurring three miles north of Bristol, in Addison County. G. H. Perkins, the State Geologist, of Vermont, reports them to be Cambrian and of the same age as the Swanton and Winooski marble. The marble has a light cream-colored ground mass with variously blended pink to pale red color-groupings, frequently shading off into the yellowish background. The stone as a whole presents a beautifully mottled effect with mellowed tones of pink and yellow. Other phases show a white ground mass with more intense red mottlings. The first variety is much the more artistic and if supplied to the trade in ample quantities will displace a large part of the colored Italian marbles, notably the Sienna variety, which it most resembles. The artistic capabilities of this stone are not equalled by any American marble now on the market. The marble is very hard for this class of material, and takes a very brilliant and enduring polish. After close investigation the United States Commissioners for the Paris Exposition accepted this stone for the United States building at Paris as the most decorative marble in America.

ONYX AND VERDE ANTIQUE.—*California.*—Travertine onyx are quarried to a limited extent. The Island of Catalina affords serpentine and verde antique, used successfully for interior work in this State, and are finding an introduction as electrical fuse blocks, tablets, and switchboards.

Maryland.—The serpentine and verde antique are quarried only to a limited extent. They are derived from older basic igneous rocks, known as peridotites.

Utah.—Ample quantities of onyx of great variety and of exceptional beauty are found within the State. The yellow, brown and orange varieties from Pelican Point are to be exhibited at the Paris Exposition. The onyx industry was at a standstill during 1899, but a company has been recently incorporated to exploit the Bearnice onyx quarries in the Big Cottonwood mining district.

Washington.—There are two new companies in the field, which have been incorporated to work the so-called marble occurring on Greenway Mountain, near Valley, Stevens County. The stones produced from this district are known as "Light Landscape," "Autumn Tint," "Landscape Green," Royal Serpentine," and "Light Mottled," the latter three being the most beautiful. All five varieties are translucent phases of serpentine and are of remarkable beauty and rarity. The Landscape Green is a gray-green phase with an appreciable percentage of white calcite so grouped as to give the stone a landscape, or more properly, a mackerel-sky effect; the Royal Serpentine is of that unusual rich, yellow-green color and translucency characteristic of this hitherto rare stone. The Light Mottled is a nearly white, very translucent serpentine, faintly mottled and banded with pale green and strikingly resembles some varieties of onyx. The United States Marble Co. reports that the verde antique "vein" is 300 ft. wide at a minimum and has been traced half a mile, while its depth is known to be over 1,000 ft.

SANDSTONE.

There are several varieties of quartz and highly siliceous sedimentary rocks that are of industrial importance, and under this division will be discussed those that pertain more particularly to building and other structural purposes. The varieties of diatomaceous earth, flint and quartz, moulding sand, and tripoli will be found under the caption "Silica"—elsewhere in this volume.

The production of sandstone in the United States is mainly employed for constructional purposes, the chief variety being that commonly known as brownstone and bluestone, the lighter colored sandstones and quartzites being of lesser importance. The hard compact sandstone quarried in Ohio for the manufacture of grindstone will be discussed under the caption "Silica."

Sandstone to the approximate value of $5,600,000 was produced in the United States in 1899, against $4,800,000 in 1898, and the values of the bluestone production for the two years were $1,000,000 and $800,000.

Alabama.—Sandstones are quarried from the Cambrian and Sub-Carboniferous and the supply is practically unlimited.

Arizona.—In 1898 the sandstone production was used chiefly in the construction of railroad bridges. Sandstone comes mainly from one irregularly-worked quarry near Flagstaff and other intermittently-operated quarries near Phœnix and Tucson. The Flagstaff stone is quite widely known and is shipped to Chicago, California and Oregon to some extent.

California.—There are some 15 quarries of sandstone producing stone used for general building purposes. With the exception of the stone used for jetties, building sandstone is little worked in the State. Twenty-five or more quarries supply the road metal and the stones used are tough, blue sandstone, diorite, limestone, basalt, marble, jasper and granite. It is to be regretted that basalt, which abounds in California, is not more largely used for road metal. Sandstone, while often hard, is brittle, possesses no cementing power, and is to be discriminated against for this use when possible. The total output of sandstone in 1899 was 56,264 cu. ft., valued at $103,384, against 37,264 cu. ft. ($46,384) for the preceding year.

Colorado.—The best developed sections of Boulder County are about Lyons, Noland and Beach Hill, and are furnished transportation by the Stone Mountain Railroad, a standard gauge road, 5 miles long, extending from Noland to the quarry area about Beach Hill, and having curves of 45°.

This section turns out sandstone known to the trade under the general name of "Lyons stone." There are at present 20 shippers, employing some 218 hands. Several quarries are worked by lessees, who pay the owners 15 to 30c. per ton royalty, according to the quality of the stone marketed

Near the city of Boulder, the Woods Bros. quarry furnishes a compact, gray sandstone which is used extensively.

Larimer County possesses sandstone and granite quarries and is second in importance in the State.

Connecticut.—The brownstones of this State have long been known and widely used all over the country. While the business in this stone is not of former pro-

portions, it is unlikely that the unsatisfactory condition noted in 1898 will continue to exist indefinitely. In fact there has taken place a marked improvement during 1899. The construction of a sea wall at Saybrook Point is to be of brownstone from Shailerville, and greater activity in all branches of the trade, both in granite and sandstone, is reported from all parts of the State. The cause of the low ebb in the sandstone industry is to be ascribed to various causes. Among them may be mentioned the fact that this stone is lessening somewhat in popularity in this country, and the business had not, up to 1898, recovered from the general depression of the last few years. Sandstone employed in portions of buildings badly exposed to rains and frosts is rightfully falling into disuse more and more, and stones more resistant to our varying climatic conditions are taking its place. The invention and wide introduction of pneumatic tools, which permit a saving of about one-half over purely hand methods, has so cheapened the cost of ornamental and sculptural work that granite and other hard and expensive stones have become active competitors of the softer sandstones once so largely used for these purposes. The field of brownstone has also been intruded upon by fancy-colored brick and terra-cotta. Modern steel-frame buildings can be so economically and quickly built that facings of granite, which were too costly 10 years ago, are often used to the exclusion of cheaper sandstones. The neighboring State of Massachusetts has also appeared in the field as a competitor in this class of stones by exploiting anew the rich, red sandstones of the Connecticut Valley at Longmeadow.

Maryland.—Seneca red sandstones which are of Triassic age, are extensively quarried, and are well known outside the State. From other localities much sandstone is wrought for local use.

Massachusetts.—The sandstones of Massachusetts are a northerly extension of the Triassic beds of Connecticut. The principal quarries are situated in Hampden County. The Massachusetts sandstone is more compact, and the grains are more firmly cemented together than the series further south. As a result the absorptive power is less, and the compression strength is higher; two qualities which render it superior to the general run of Connecticut sandstones.

Minnesota.—Minnesota and South Dakota enjoy the possession of an exclusive line of building material in the so-called "jasper." This stone is a highly solidified sandstone or quartzite, which has great enduring qualities, but is worked with difficulty, owing to its uniform hardness.

Montana.—The sandstone quarried in this State is used for general building, for making fire-brick, paving and broken stone. The State Capitol is to be built of the gray sandstone quarried at Columbus, Yellowstone County. The output of sandstone for building purposes in 1899 was valued at $21,897, for flux and road metal at $12,750.

New Jersey.—The sandstones compare favorably with those of the same geological age in Connecticut. These are wrought in three principal areas; near Wilburtha; on the Delaware above Stockton and at Avondale. There are 14 quarries in operation in the Stockton beds, 9 in the Lockatong beds near Wilburtha and 25 in the Brunswick series at Avondale.

New York.—Sandstone is second in economic importance in New York State.

The sandstone wrought comes from many geological horizons—from the Potsdam formation near Potsdam, and from Fort Ann, where a hard quartzite is extracted for paving and road metal, and from the Hudson River groups which furnish stone mostly for local purposes. The "Medina red stone," "gray Medina" and "variegated" come from the Medina formation, where the most valuable sandstone of the State is quarried. The Hamilton group furnishes the celebrated New York bluestone used so generally for flagging and trimming as well as for building purposes. Ulster bluestone is reported to have three times the resistance to transverse strain possessed by granite or other building stones, making it especially valuable for lintel sills, etc. In 1897 there were 1,133 sandstone quarries in operation in the State, and 693 separate quarries of bluestone exclusively.

South Dakota.—Quartzite is the principal building stone in the State, and the greater part of it comes from Sioux Falls and Dell Rapids. During 1899 quartzite paving blocks were shipped to Chicago, Kansas City and St. Louis; the building stone found a market principally at Sioux Falls, S. D., and Sioux City, Minn.

Buff-colored and pearl-gray sandstone was quarried at Hot Springs in 1899. This stone has a crushing strength test of 6,914 lb. per sq. in.

Utah.—Sandstone is quarried at several points, notably along the Oregon Short Line Railroad at Thistle Junction, and near Red Narrows, Milk Fork, Tucker, Soldier Summit, Colton, Kyune and Nolan.

Washington.—At Tenina, there is the largest sandstone quarry in the State, the stone being marketed mostly in Portland, Ore.

Wisconsin.—Sandstones which come from the Potsdam and St. Peter's formations are very important building materials, and have a growing market. This State, in common with Maryland, has recently issued a volume* descriptive of its stone resources, which is of great value and should be in the hands of every one interested in United States building and ornamental stones. Many colored plates of prominent granites, rhyolite and sandstones are included, which are remarkable for the perfection with which color, grain and structure have been reproduced.

Wyoming.—Thirteen active quarries are in operation in this State, but there are many others worked intermittently which supply the local demand for stone. During 1899 the sandstone industry was relatively active. Considerable of the stone is used by the Oregon Short Line and Union Pacific Railroads. Kammerer sandstone is to be used in constructing the Saint Mary's Cathedral in Salt Lake City, Utah, an important piece of work, requiring 400 car loads of stone.

Road Metal.

During 1899 the spread of macadam and telford road building, which has been progressing in this country for the last five years was maintained, and the consumption of the various stones used for this purpose is becoming an important part, both in value and in quantity of the total production of stone in many States. Instruction is being given the citizens of this country as to the qualities

*Bulletin of the Wisconsin Geological and Natural History Survey, No. 14. Economic Series. No. 2, on the "Building and Ornamental Stones of Wisconsin," 1898.

on which good road stones depend. Massachusetts, Maryland, Wisconsin and the Department of Road Inquiry, established by the United States Government, are active in imparting information of this kind. There is pressing need that all States should follow the example of Maryland and instruct their citizens in this regard, and, at the same time, furnish imformation as to the kinds and distribution of stones suitable for this purpose within their boundaries. The aggregate saving to the people in any one year, if this were done, would more than equal the first cost of such an investigation. Much is being accomplished in the establishment of laboratories run by the States where stones may be tested as to their suitability for road metals, free of charge.

Laboratories for testing road stone will never be equipped as they ought to be until the crushing strength, as well as abrasion and cementation, be ascertained. Where a granite sometimes has as good an abrasion test as many diabases, it is inferior as a road metal, not only on account of less binding power, but because it is more easily crushed. Some diabase is known to have a crushing strength of 37,000 lb. to the sq. in., while granite seldom attains a resistance of 30,000 lb. to the sq. in. The difference in strength is reflected somewhat in the abrasive tests of these two classes of rocks, granite averaging considerably poorer than trap. This is, however, more largely due to the fact that while granite is harder than diabase, it is also more brittle, and its coarser fracture presents more angles to be broken off in the abrasive test. In testing a stone to ascertain its crushing strength it will be found in some instances that a 2-in. cube of each, for example, will offer the same resistance to rupture, but it will in all probability be true that, of two rectangular solids of the same stones, say 0·25 in. in thickness, the trap will have the greater crushing strength. This difference will theoretically reach its maximum in comparing an abnormally coarse-grained granite and an unusually fine-grained diabase, for the reason that in thin fragments of a very coarse stone, the strength due to any interlocking of the minerals composing it is destroyed, whereas in the diabase, owing to its fine grain, the interlocking arrangement of its minerals is still effective.

The maker of rock sections for microscopic study is very familiar with the difficulty due to this cause in cutting coarse rocks, like granite, in comparison with fine-grained rocks, like trap. When granites are crushed into grains made up partly of quartz and partly of feldspar, further subdivision is made easy, owing to the fact that the surface of junction (bounding walls) between the minerals is one of great weakness. Fragments of this kind increase in number with the coarseness of the stone. The fine, granitic material from the stone crusher is principally made up of fragments of feldspar and fragments of quartz: the former are easily broken on account of their excellent cleavage; the latter on account of their brittleness. Objections of this kind are minimized in diabases (fine-grained rocks without quartz) which will allow fine crushing without reduction to their individual minerals. Highway surfaces of granite, where exposed to the wear and tear of heavy traffic, soon become covered with two separate minerals, quartz and feldspar, both objectionable in themselves: trap on the contrary may be finely subdivided and still be in the condition of a rock with the

qualities inherent in the large fragments still present. In the case of granites the objections of this nature increase with their coarseness, and are the least in the fine-grained, aplitic phases.

It is accordingly more important to learn by a series of tests the crushing strength of the smaller fragments of stone used in making macadam roads than the test from a builder's standpoint.

Connecticut.—This State is rich in road stone of the highest class. Diabase in the form of great intrusive and extrusive sheets occurs in the belt of Triassic sandstones and shales occupying the Connecticut River valley, and several belts of this stone are continuous across the State from north to south. There are also many dikes of this stone scattered over the State well situated for the economic distribution of road metal. Large quantities are used at the present time for road metal, breakwaters and railroad ballast. Recent tests show that diabase has a crushing strength of 34,920 lb. per sq. in.

Maryland.—This State has recently taken the rank in systematically studying, as part of the work of the State Geological Survey, her resources in building and ornamental stones, and also the kinds and distribution of her available road metals. In 1898 there appeared Volume II. of the Survey, which presents the building stone resources of the State before the people, and in 1899 the volume devoted to road stones followed. This latter volume is devoted exclusively to a consideration of the State highways, and the part played by road metals is comprehensively dealt with.

In common with other States along the Atlantic, the older rocks comprising the core of the Appalachian Mountains, are extensively exposed in this State; the consequence is, as would be anticipated, that the greatest diversity exists in the types of road stones. Among the many varieties occurring, the State is able to secure the kinds of stone which practical and theoretical considerations have shown to be the best. Of the igneous rocks, traps have been shown to be superior, based on their wearing and cementation qualities. From data determined by the State testing machines, in a laboratory which has been recently fitted up patterned after the one in Massachusetts, it appears that the merits of these so-called traps decrease for road purposes as we pass from normal diabase through gabbro-diorite, gabbro and diorite.

As regards the investigations of road stones as a whole, the results, while not exhaustive, are somewhat at variance with accepted standards. The average tests of diabase agree very closely with the more extended observations on Massachusetts rocks of this kind, but in some instances limestones are classed with gabbro-diorites so far as wearing qualities are concerned. It is noticed, however, that the limestones having this unusual abrasive strength, carry an appreciable percentage of silica, which no doubt accounts for their superiority over those free from silica. So far as the granites and gneisses are concerned they are shown to be fully equal to and accord almost exactly with similar rocks from Massachusetts. The poorest rocks, all things considered, are the sandstones and most limestones. It is shown that so far as cementation is concerned, serpentines are the best, and that limestones approach the most nearly in this particular.

Massachusetts.—The Somerville district annually produces large amounts of

a hard blue slate which is used as telford foundation and macadamized roads.

This industry has already assumed great proportions as a result of the large amounts required for the rapid increase in road building in this State. Fine-grained, basic rocks, occurring as trap dikes and lava flows are most sought after for this purpose. The Connecticut Valley region and the eastern portion of the State are best supplied with this material. For a more complete description of Massachusetts road metal see THE MINERAL INDUSTRY, Vol. VII.

Michigan.—The region about Keeweenaw Point abounds in basic rocks adapted to the requirements of the best type of modern highway construction.

New Jersey.—Crushing tests of some New Jersey diabase show a strength of 37,000 lb. per sq. in., a result among the highest strength tests on record. The laboratory tests of road stones, made by the Massachusetts Highway Commission, credits Guttenberg, N. J., trap with the greatest resistance to abrasion (or the highest wearing qualities) of any stone thus far experimented with. Taking as a standard a coefficient of wear of 20, the Guttenberg diabase is 30 to 40.

For road making, trap rock is used almost exclusively, although the hard, blue argillite, occurring at Byram, is quarried for this purpose to a limited extent. The traps of New Jersey are obtained from the Triassic (Newark) sandstone and slate series of rocks into which the diabase has been injected. There are 37 quarries in trap rock in operation as follows: Six in the Palisades on the Hudson River; one in the Granton area; one at Snake Hill; 16 in the First Watchung Mountain; 5 in the Second Watchung trap sheet; 2 in the Third Watchung Mountain; 3 in the Rocky Hill trap sheet and 5 along the Delaware River.

New York.—New York is easily second in the production of road metal. Of this amount the great bulk is diabase or trap, while granite and other rocks are used. Trap of the first quality comes from Richmond and Rockland counties, the Palisades furnishing the best stone in the State. There were 44 quarries operated for road metal in 1897.

North Carolina.—There is considerable activity in quarrying stone for macadam, granites, gneisses and soft-shell limestone being used for this purpose.

Oregon.—Tertiary basalt is used in Eugene and Lane counties as a road metal, and about Portland there are a number of quarries operated for road stones. At Fisher's Landing, on the Columbia River, above Vancouver, is the largest quarry on the coast. This supplies basalt used for jetties and is being worked at present for a jetty at Gray's Harbor.

Rhode Island.—Trap dikes and other sources of road metal occur in various parts of the State, notably in the western and northwestern portions. Except in the large cities very little attention has been paid to the subject of road improvement. The town of Johnston supplies trap rock with a coefficient of wear of 21·41. The principal part of the stone used for macadam on the streets of Providence comes from this locality.

SULPHUR AND PYRITES.

THE production of sulphur in the United States in 1899 was derived from Utah, Nevada and Louisiana, and amounted to 1,565 long tons, against 2,726 in 1898. Utah contributed the chief amount, while the production from Nevada decreased to less than one-third of the previous year's output, and that of Louisiana consisted of a few tons only. There was some development of the sulphur deposits in California and Texas, but no production was reported from these States. The decreased production was due mainly to the continued heavy importation of Sicilian sulphur, and the growing tendency of sulphuric acid makers to utilize the cheaper pyritiferous ores in place of brimstone. The development of the sulphur industry on the Pacific coast is discouraged by the importation of Japanese brimstone, which can be delivered at San Francisco very cheaply.

The Nevada mines of the National Sulphur Co., of St. Louis, Mo., are located on the edge of Black Rock desert, 32 miles north of Humboldt House, on the Central Pacific Railroad.* The ore occurs in masses of various sizes up to several hundred pounds in weight, which are mixed with clay among heaps and layers of ashes and light gravel. The adjoining rocks are black, slaty marl and limestone. The product is mined by tunnels and drifts, run in from the level of the wagon road, which extends entirely around the hill at a good elevation. The sulphur is refined by dry steam in iron kettles, each of 4-tons' capacity. Colorado coal is used for fuel, and although of greater cost it gives better results than the local cedar wood. The refined product is shipped to powder factories and acid makers of San Francisco.

SULPHUR PRODUCTION, IMPORTS, AND CONSUMPTION IN THE UNITED STATES.

Year.	Production.			Imports.								Consumption. (a)	
	Sulphur.			Crude.		Flowers of Sulphur.		Refined.		Totals.			
	Long Tons.	Value.	Value per Ton	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value	Long Tons.	Value.	Long Tons.	Value.
1895	1,650	$26,400	$16·00	125,950	$1,598,148	581·00	$12,888	229·00	$4,378	126,760	$1,610,414	128,190	$1,633,129
1896	3,800	72,200	19·00	145,318	2,065,076	665·00	13,266	447·00	8,226	146,430	2,106,568	149,746	2,170,233
1897	1,690	34,814	20·60	138,846	2,442,420	319·00	7,950	148·00	3,387	139,313	2,453,757	140,809	2,484,589
1898	2,726	59,754	21·92	159,790	3,081,974	507·00	14,548	163·00	4,391	160,460	3,100,913	161,772	3,128,845
1899	1,565	33,535	21·46	142,234	2,484,791	336·00	9,917	184·00	4,519	143,754	2,499,227	144,842	2,522,008

(a) In calculating the consumption of sulphur the exports were taken into consideration, although they do not appear in the above table, viz.: 1895, 220 long tons ($3,685); 1896, 484 long tons ($8,535); 1897, 194 long tons ($3,982); 1898, 1,414 long tons ($31,822); and 1899, 477 long tons ($10,804).

* *The Engineering and Mining Journal*, July 15, 1899.

Prices.—Sicilian brimstone ruled steady throughout 1899, owing to the successful operation of the Anglo-Sicilian Sulphur Co. The average monthly price of seconds and thirds during 1898 and 1899 is given in the following table:

AVERAGE PRICE OF BRIMSTONE PER LONG TON IN NEW YORK.

Month.	1898.		1899.		Month.	1898.		1899.		Month.	1898.		1899.	
	Seconds.	Thirds.	Seconds.	Thirds.		Seconds.	Thirds.	Seconds.	Thirds.		Seconds.	Thirds.	Seconds.	Thirds.
January..	$21·44	$20·56	$21·41	$19·75	May.....	(a)	(a)	$20·44	$18·50	September	$20·56	$18·50	$21·15	$19·25
February	21·50	20·38	21·44	20·12	June.....	(a)	(a)	21·07	19·07	October...	22·40	19·70	21·75	19·50
March...	22·00	20·25	22·87	20·50	July.....	$21·40	$19·60	21·70	19·43	November.	21·81	19·81	21·70	19·70
April.....	26·20	24·50	21·93	19·80	August....	20·69	19·12	21·44	18·88	December.	21·20	19·00	21·40	19·40

(a) The quotations for May and June are not given here as they would vitiate the averages for the year. Average for 1898: Seconds, $21·92; thirds, $20·14. Average for 1899: Seconds, $21·46; thirds, $19·49.

WORLD'S PRODUCTION OF SULPHUR. (a) (IN METRIC TONS.)

Year.	Austria (d)	France. (c)	Hungary	Germany	Greece.	Italy. (b)	Japan.	Russia.	Spain.	Sweden	United States.
1894.....	76	851	93	2,168	1,946	405,781	18,787	0·1	3,417	36	448
1895.....	830	4,213	102	2,061	1,480	370,766	15,557	190·0	2,231	(d)	1,676
1896.....	643	9,720	138	2,963	1,540	426,353	12,540	437·0	1,800	77	3,861
1897.....	530	10,723	112	2,317	358	496,658	12,018	574·0	b 3,500	(d)	1,717
1898.....	496	9,818	93	1,954	135	502,351	(e)	(e)	3,100	50	2,770

(a) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to about 31,000 long tons in 1898 is not included. (b) Crude. (c) Raw mineral; limestone impregnated with sulphur. (d) Crude rock. (e) Statistics not yet published.

Canada.—Near New Ross, Nova Scotia, there is said to be a number of veins varying from small seams to 2 ft. in thickness, carrying 50% ore. The building of the Hervey Railway will afford an outlet for this material.

Chile.—Extensive deposits of sulphur exist about 110 miles north of Taltal, in the first range of the Andes. It is reported that an American syndicate is negotiating for these deposits.

Peru.—In the district of Sechura a large deposit of sulphur is said to exist. It is said, however, that the sulphur is contaminated with gypseous earth. On account of the remoteness of the field it is hardly possible that it will successfully compete with the Sicilian product.

Italy.—The occurrence, mining and treatment of Italian sulphur ores are given in detail in the thorough monograph by G. Aichino later in this chapter.

SHIPMENTS OF SULPHUR FORM SICILY TO THE UNITED STATES. (IN LONG TONS.)

Port.	1893.		1894.		1895.		1896.		1897.		1898.		1899.	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.
New York.....	29,146	14,250	33,150	13,725	35,888	19,975	50,557	17,796	57,174	13,300	49,614	22,475	56,746	26,650
Charleston.....	11,665	1,860	3,273	12,023	700	8,450	2,330	5,370	1,500	3,630	500	1,600	.....	.....
Philadelphia.....	1,900	6,260	350	5,050	1,200	7,150	500	5,500	199	5,210	1,200	5,400	2,740	8,000
Baltimore.....	2,050	7,900	600	14,700	1,100	8,620	3,650	10,500	3,798	10,033	2,350	12,015	3,800	8,600
Boston.....	500	.....	1,017	3,300	2,350	2,600	4,600	700	7,220	1,000	4,500	1,550	600	1,000
Savannah.....	3,450	1,880	5,695	4,100	3,784	800	8,370	1,025	4,700	.....	1,980	.....	.....	.....
Wilmington, N. C...	.....	1,140	.....	1,890	.....	650	1,260	1,400	.....	1,550	500	1,200	.....	.....
New Orleans.....	1,900	.....	2,400	.....	1,700	.....	2,100	.....	3,340	.....	500	2,000	.....	800
Other ports (a).....	.....	.....	800	3,700	1,880	2,380	7,975	1,290	4,883	600	18,159	200	19,505	.....
Totals.....	50,611	33,290	47,285	58,488	48,602	50,625	81,342	43,581	82,814	35,323	79,303	46,440	83,391	45,050

(a) Norfolk, Mobile, San Francisco, Portland, Me., and Portland, Ore.

TOTAL EXPORTS OF SULPHUR FROM SICILY SINCE 1891. (*a*) (IN LONG TONS.)

Country.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	1899.
United States.....	97,520	84,450	83,901	105,773	99,227	124,923	118,137	138,435	128,441
France.....	56,168	73,176	89,736	56,932	69,696	76,739	84,895	88,657	96,043
Italy.....	42,212	38,711	54,486	49,895	49,349	54,009	73,052	62,652	87,230
United Kingdom...	23,408	24,853	27,453	22,165	24,043	21,913	24,520	26,983	25,088
Greece and Turkey.	11,414	*b*14,845	*b*13,840	*b*16,870	*b*16,195	*b*18,556	*b*13,866	*b*24,906	*b* 18,656
Portugal.....	11,439	13,490	14,545	8,670	14,562	12,001	7,054	8,257	12,269
Russia.....	11,930	14,178	19,730	17,977	17,962	18,752	17,532	12,285	19,211
Germany.....	10,629	14,326	16,259	16,437	15,472	15,680	19,721	27.048	25,933
Austria.....	10,575	9,096	10,169	11,494	12,170	13,799	15,993	15,796	18,519
Turkey.....	3,000	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)
Spain.....	3,845	7,382	3,499	3,445	5,753	5,910	4,039	3,233	7,757
Belgium.....	5,089	5,133	4,358	5,644	6,410	7,527	9,253	8,402	7,481
Holland.....	.....	2,183	2,957	2,365	3,235	3,834	3,599	5,646	6,408
Sweden and Norway	2,252	4,561	6,579	7,887	5,730	14,540	11,226	12,331	12,476
Australia.....	.....	1,200	.....	.....	.....	.....	.....	.....	.....
Denmark.....	300	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)
Other countries....	3,542	3,152	1.680	3,376	7,782	8,562	7,651	12,791	13,569
Totals.....	293,323	309,536	349,192	328,930	347,636	396,745	410,538	447,324	479,031
Stock in Sicily at end of year.....	117,037	175,299	210,665	198,513	208,756	222,999	240,367	248,023	277,098

(*a*) From report of A. S. Malcomson, New York. (*b*) Exports to Greece and Turkey combined after 1891. (*c*) Included in exports to Sweden and Norway.

Stocks in Sicily on December 31, 1899, were 277,098 long tons, against 248,023 tons at the same period in 1898. Ocean freight rates were firm throughout 1899, and were 10s. to 11s. 6d. on contract to the Atlantic ports in the United States. The Anglo-Sicilian Sulphur Co., Ltd., has rescinded its contract with the millers and refiners of Catania, and is purchasing from outsiders. This company reports a net profit for the year ending July 31, 1899, of £93,232, of which a 6% dividend (£34,334) was paid on the preferred shares. Of the remainder £11,779 has been credited to the capital guarantee fund, £27,673 to the general reserve fund, and the balance, £19,444, was distributed in further dividends.

Japan.—The production of sulphur in Japan in 1899 was consumed for the most part by the local manufacturers of sulphuric acid, although several shipments were made to the United States. The Volcanic deposits present increasing difficulties in extracting the ore, and the average cost of production has been augmented.

Mexico.—The Mexican Sulphur Co. produced a small quantity of sulphur in Lower California, which was shipped to San Francisco.

Spain.—The sulphur deposits are in the Middle Eocene formation, and lie between Gador and Lorca in the Sierra Nevada, about 11 miles north of the port of Almeria. The formation consists of a succession of strata, which in descending order, consists of coarse grained limestone, limestone conglomerate and clay marls. The sulphur, occurring in the joints and cavities in the limestone, acts as a cementing material in the conglomerate, and appears in a network of veins in the marls. Between the marls and conglomerates there is frequently a band of almost pure sulphur of varying thickness. The sulphur zone at Gador is about 1,300 ft. in length, by 400 ft. in thickness, and 160 ft. in depth.

The average assay in sulphur is estimated at 15%. No uniform system of mining has been employed. As in Sicily, the ore is largely extracted by boys. Beehive furnaces are used for sublimation of the sulphur, which are somewhat striking on account of their size. There are 25 furnaces, 18 of which have a

capacity of 180 tons each, while seven take a charge of 240 tons. The larger furnaces measure 16·4 ft. in diameter and 16·4 ft. from the floor to the spring of arch, with a total height of 24·6 ft., and are operated like the Sicilian calcarone. The large furnaces produce about 12·5% of sulphur or about 30 tons per charge, with a reported loss of not over 1·5% of total charge, though probably this is exceeded. The operation occupies from 12 to 15 days. The annual production at Gador was about 4,500 tons of merchantable sulphur at a cost of $9·64 f. o. b. Almeria.*

PYRITES.

The production of pyrites in the United States in 1899 was a little less than that of 1898, owing to fires which destroyed a large portion of the plant at the sulphur mines in Virginia, and totally incapacitated the mines of the Pyrites Mining and Chemical Co. near by. The properties in Tennessee and New York States showed an increased production, although the total amount of the output is comparatively small. There have been no new developments whatever on the several mineral belts which produce pyrites along the Eastern coast line, but we hear of increased production along the Pacific coast mostly

PYRITES STATISTICS OF THE UNITED STATES. (a)
(IN TONS OF 2,240 LB.)

Year.	Production.		Imports.			Consumption.	
1895.....	107,371	$342,587	190,436	46%	673,812	297,787	$1,016,669
1896.....	109,282	292,626	199,678	47%	1,140,571	308,960	1,433,197
1897.....	133,368	404,699	259,546	47%	847,419	392,914	1,252,118
1898.....	191,160	589,329	171,879	47%	544,165	363,039	1,133,494
1899.....	178,408	583,323	310,615	47%	1,084,451	489,023	1,667,774

(a) These statistics do not include the auriferous pyrites used for the manufacture of sulphuric acid in Colorado.

for their copper products. It is supposed that the difficulties in the way of utilizing pyrites for the manufacture of sulphurous acid gas to be used by the sulphite pulp mills in northern New York have now been overcome, and that for the year 1900 several deposits which have awaited the perfection of a roasting process will come into prominence as producers for this special industry. In this connection it may be noted that the largest sulphite pulp mills in the United States are located in Maine, New Hampshire, Massachusetts and New York, and are within easy reach of a class of low grade copper ores, which, after concentration, it is expected will become valuable for the manufacture of sulphurous acid to supply this special industry.

Demand.—An unusual and steady demand for sulphur products for the manufacture of fertilizers, the refining of oils, and the unusual demands for chemical purposes, has led to an increased consumption both of brimstone and pyrites, so that nearly all stocks which had accumulated at the end of 1898 have been depleted, and the demands at the beginning of the year 1900 have tended to stiffen prices for both foreign and domestic ores, a part of which is due to the increase in ocean freights.

* Abstract from a paper by Arthur P. Wilson, in the *Transactions of the Institution of Mining Engineers*. April, 1899.

Prices.—American pyrites is now sold for the first time since it became an article of commerce, very nearly on the basis per unit as foreign ores, and for the latter part of the year prices were firm at 10, 11 and 12c. per unit of sulphur contained in ores which analyzed over 41%, as against the prices of the year before which ranged from 4 to 8c. per unit at the mines. This radical change, and the probability of a long continued trade on even terms with foreign ores is unusually gratifying.

New York.—The Stella mine at De Kalb junction, which was abandoned 14 years ago, was preparing to resume operation the early part of the year. On the property there is one shaft 500 ft. deep, and another 60 ft. It is proposed to sink the latter. The run of mine shows S, 33%; SiO_2, 32%; Fe, 32%; Cu, 0·04%, with traces of gold and silver. Wet concentration is said to bring the material up to 51% S, while 43% S has been obtained in unsatisfactory experiments with dry concentration.

Virginia.—(By W. H. Adams.)—For the coming year the most radical changes in the business of producing American pyrites will come from the withdrawal from the open markets of the largest producer, *viz.:* the Sulphur Mines Co., of Virginia, as this mine property has been sold to the Southern Fertilizer Combination, known as the Virginia-Carolina Chemical Co., and its entire output of ore will henceforth be diverted to the use of this company alone at its several works in the Southern States. It is expected that on April 1, 1900, a milling plant will be ready to run at these mines capable of turning out 300 tons per day of concentrated pyrites, say on an average of 44% sulphur. A plant of this character will admit of mining the entire body of mineralized slates on this property, nearly two miles in length and supposed to contain millions of tons. A plant of this character built with very modern convenience and designed by engineers who have studied this question closely, will, no doubt, be one of the great object lessons of the decade and prove of incalculable benefit to all low grade pyrites propositions along the Atlantic coast.

WORLD'S PRODUCTION OF PYRITES. (*a*) (IN METRIC TONS.)

Year.	Belgium.	Bosnia.	Canada.	France.	Germany.	Hungary.	Italy.	Newfoundland.
1894.....	3,050	.....	36,766	283,439	134,787	76,870	22,638	41,436
1895.....	3,510	.....	31,024	253,416	127,036	69,195	38,586	34,879
1896.....	2,560	2.000	30.586	282,064	129,168	52,697	45,728	27,712
1897.....	1,828	3,670	35.299	303.448	133,302	44,454	58,320	33,316
1898.....	147	.....	25,118	310.972	136,849	58,079	67,191	33,100

Year.	Norway.	Portugal.	Russia.	Spain. (*d*)	Sweden.	United Kingdom.	United States.	Totals.
1894.....	70,859	(*c*)	19,801	60,000	656	15,771	107,462	1,073,535
1895.....	61,994	(*c*)	11,042	60,267	221	9,193	107,871	1,007.732
1896.....	60,507	(*c*)	11,550	100,000	1,009	10,178	117,782	1,075.191
1897.....	94,484	210.265	19,380	217,545	517	10,753	123,468	1,295,046
1898.....	*b* 90,000	248,218	*b* 20,000	260,016	386	12,302	194,219	1,456,597

(*a*) From the official reports of the respective governments except the figures for Spain in 1897, which are taken from C. Le Neve Foster's report. (*b*) Estimated. (*c*) Not reported. Under Portugal is included only the output of pyrites carrying less than one per cent. of copper. Considerable amounts of the richer sulphides are also used in sulphuric acid manufacture. In completing the totals the output in 1894, 1895, and 1896 is estimated at 200,000 metric tons. (*d*) Does not include the production of copper-iron pyrites, from which the copper is extracted.

North Carolina.—As a coming addition to the sulphur products of the Southern States it may be noted that developments, now being made on a very large scale at the Union Copper Mine are expected to turn out through the new concentration mill which is being erected on this mine, a large tonnage of ores suitable for roasting, and possibly for the manufacture of sulphuric acid.

Newfoundland.—The Pilley's Island pyrite mine was closed down on November, 1899, and it is doubtful if it will again be reopened; 31,500 tons of ore were shipped from this property in 1899, of which 30,000 were exported to the United States and the balance shipped to Chatham, N. B.

France.—There are four pyrites deposits in France, from three of which the production has been insignificant. The Sain-Bel (Rhône) mine produced 310,000 metric tons in 1898, against 300,000 tons in the preceding year. The greater part of this output is consumed by the Société des Manufactures Saint-Gobain Chauny et Cirey, which owns the mine. A portion, however, is exported.

Portugal.—The production of pyrites in Portugal was 248,218 metric tons in 1898, against 210,265 tons in 1897. This ore contains less than 1% of copper and is derived mainly from the Mason and Barry mines. Large quantities of sulphide ores richer in copper are also used in the manufacture of sulphuric acid.

Spain.—The exports of iron pyrites as given by Señor Adriano Contreras, in the annual statistical number of the *Revista Minera,* was 319,285 metric tons in 1899, against 260,016 tons in 1898, showing a continued increase over previous years. In addition to this output there was a large exportation of cupriferous pyrites that was utilized for the manufacture of sulphuric acid as well as for the extraction of the copper contained.

CONSUMPTION OF SULPHUR IN THE UNITED STATES.

	1896.	1897.	1898.	1899.
Domestic production of sulphur	3,800	1,690	2,736	1,565
Imports of brimstone	145,318	138,846	159,790	143,234
Total	149,118	140,536	162,516	144,799
Exports of brimstone	484	194	1,414	477
Consumption	148,634	140,342	161,102	144,322
Sulphur contents at 98%	145,661	137,535	157,880	141,336
Domestic production of pyrites	109,282	133,368	191,160	178,408
Imports of pyrites	199,678	259,546	171,879	310,615
Sulphur in domestic pyrites at 44%	48,084	58,683	84,110	78,500
Sulphur in foreign pyrites at 47%	93,849	121,986	80,779	145,989
Total sulphur consumed in pyrites	141,933	180,669	164,889	224,489
Total sulphur consumed in brimstone	145,661	137,535	157,880	141,336
Grand total	287,594	318,204	322,769	365,825

In the above table the imports and exports are for crude sulphur or brimstone only; the figures of consumption, therefore, differ slightly from those given in the preceding table.

Sulphuric and Sulphurous Acids.

On account of the use of sulphuric acid in many manufacturing operations it may be considered the most important of all chemicals; in fact the statement has been made that the industrial progress of the nation is directly in propor-

tion to its consumption per capita of sulphuric acid. The greater part of the output of sulphuric acid produced yearly does not appear in the market, as consumers of large amounts generally make their own supply. If the entire amount of sulphur and pyrites consumed in the United States in 1899 were converted into sulphuric acid it would have produced 1,257,599 short tons of acid of 66°B. It is not fair, however, to assume that the consumption of brimstone and pyrites is utilized for sulphuric acid manufacture, as large quantities are converted directly to sulphurous acid and used as such in preparing sulphite pulp for the manufacture of paper. It is not possible to obtain statistics of the amounts of brimstone and pyrites used for this purpose, nor for the accurate production of sulphuric acid, though probably the greater part of pyrite consumption is utilized for acid manufacture. There was a considerable amount of sulphuric acid recovered as a by-product in roasting sulphide ores of zinc and copper and auriferous pyrites. The production of acid from this source amounted in 1899 to 59,572 short tons, against 47,558 tons in 1898. These figures were reduced to a common basis of 66°B. acid, whereas much the greater part of the production was marketed as chamber acid, which naturally increased the tonnage actually handled. The average price of sulphuric acid, concentrated and chamber, at New York in 1898 and 1899 is shown in the subjoined table.

Month.	Conc. Acid. 66° B per 100 Lb.		Chamber Acid. 50° B. per Ton.		Month.	Conc. Acid. 66° B. per 100 Lb.		Chamber Acid. 50° B. per Ton.	
	1898.	1899.	1898.	1899.		1898.	1899.	1898.	1899.
January.....	$1·00	$1·10	$7·25	$11·50	July.....	$1·10	$1·10	$11·55	$11·50
February.....	1·00	1·10	6·50	11·50	August.....	1·10	1·10	11·00	11·50
March.....	1·00	1·10	6·50	11·50	September...	1·10	1·10	11·50	11·50
April.....	1·02	1·10	7·50	11·50	October.....	1·10	1·10	11·50	11·50
May.....	1·10	1·10	11·50	11·50	November...	1·10	1·15	11·50	18 75
June.....	1·10	1·10	11·50	11·50	December....	1·10	1·20	11·50	16·00

The average price of 66° B. acid in 1898 was $1·07, and $1·11½ in 1899; 50° chamber acid averaged $9·98, and $12·06¼ in 1899.

There was a consolidation in 1899 of a number of Eastern sulphuric acid manufacturers forming the General Chemical Co., which was capitalized at $25,000,000 and began operations with $750,000 capital. The net profits for the ten months ending December 31, 1899, were $959,696.

The largest consumers of sulphuric acid in the United States are the Virginia-Carolina Chemical Co. and the American Agricultural Chemical Co., which use it in the manufacture of acid phosphate of lime; and the Standard Oil Co. which employs the acid for refining petroleum; probably 90% of the production in the United States is used for these purposes.

Liquid anhydrous sulphurous acid is manufactured by the Somerset Chemical Co., of Bound Brook, N. J. This acid sells in New York at $8 to $10 per 100 lb., and is shipped in cylinders containing 100, 200 and 500 lb. each. These cylinders are so constructed that either liquid or gaseous sulphurous acid can be drawn off, and they are worth respectively $12·50, $17·50 and $25 for the three sizes. The sulphurous acid is made by the Haenisch and Schroeder process, which is fully described in the special monograph by Dr. Franz Meyer later in this chapter.

Progress in the Sulphuric Acid Industry in 1899.

By John B. C. Kershaw.

The apparatus and process used for the manufacture of sulphuric acid, have been so fully dealt with in previous volumes of this Annual by Lunge, * Weidig,† and Falding,‡ that it is only necessary in this article to record the progress of the past year, and to direct attention to the more important contributions to the technical literature of the subject during 1899. The following notes will not, however, deal with concentrating apparatus, as a separate article, by F. J. Falding, is devoted to this subject in the present volume.

Forbes-Carpenter, in the last published report upon the chemical industry in Great Britain, gives some interesting notes upon the development and improvement to be observed in the sulphuric acid works under his inspection. These deal chiefly with the Lunge-Rohrmann "plate-towers" which are in operation at three works, viz.: at Briton Ferry, near Swansea: at Pontardawe: and at a sulphuric acid works in the Midlands. At the first-named works the chamber-space is 127,000 cu. ft., and 15 tons of 35% pyrites are burned per day. Allowing for the sulphur lost in the burnt pyrites, and in gas escapes, this equals a daily consumption of 10,060 lb. sulphur. The chamber-space is therefore only 12·6 cu. ft. per lb. of sulphur burnt per 24 hours. This result has been obtained by the use of a Lunge-Rohrmann tower containing 15 layers of 16 plates each. During 30 days' operation, this tower was found to have condensed 50 tons of 62% acid. The niter consumption remained stationary, after insertion of the tower in the chamber system. It would have been interesting to know exactly the percentage of niter upon the sulphur burned; and also what space was used at these works, before the installation of the Lunge-Rohrmann tower. In the absence of these figures one cannot assert positively that the improvement is wholly due to the latter. In the same report some figures relating to the progress of the Lunge-Rohrmann towers on the Continent of Europe are given. These towers are said to be in operation at 23 vitriol works, and to be in course of construction at five others. It is surprising to learn, however, that the chief use of these towers has been in connection with the production of another acid than sulphuric; no fewer than 200 plate towers having been erected for hydrochloric acid condensation in Germany and other Continental countries.§

Gilchrist, in May, 1899, read a useful paper upon the Manufacture of Sulphur Acid in the United States before the New York Section of the Society of Chemical Industry.‖

The Herreshoff mechanical dust burner is described, and it is stated that with this burner, pyrites dust containing 42% sulphur can be burned down to 2% sulphur. The burner is said to cost little for repairs. If this be correct, the problem of dust burning would appear to have been solved. Hitherto the difficulty has been that hand-worked shelf-burners were inefficient; and that mechanical burners were efficient but costly in repairs. The writer had at one time charge of a set of shelf dust-burners, hand-worked, which could never be got

* The Mineral Industry, Vol. IV., 1895, and Vol. VI., 1897. † *Ibid.*, Vol. VI., 1897. ‡ *Ibid.*, Vol. VII., 1898.
§ Chief Alkali Inspector's Report for 1898, United Kingdom.
‖ *Journal of the Society of Chemical Industry*, May, 1899.

to reduce the sulphur in the dust below 6 to 8%; and an excessive amount of air was admitted during the hand-raking of the dust from shelf to shelf. After a few months' trial, these burners were pulled down, and the pyrites dust was again used in the ordinary burners, after milling and formation into balls. This, the writer believes, is still the customary method of dust utilization in the United Kingdom.

With regard to Glover and Gay-Lussac towers, Gilchrist states that quartz is now being used for packing; and that the towers are being built larger. Two Gay-Lussac towers are also being used in many plants. This method of increasing the absorbing surface for nitrous gases, was adopted with advantage in the plant of which the writer had charge, many years ago. With regard to the size of the lead chambers, Gilchrist states that they are being built shorter, and that 60 ft. is now the mean length. This is a great reduction on the 120-ft. length customary in the older English works. "Plate-towers" are stated to be in use in many places in America; but their adoption is restricted by their high cost. Intermediate towers of other types are, however, common. These are packed with rings, bricks, or quartz, and require to be larger than the plate-towers to obtain equal efficiency. Some of the works using these intermediate towers are said to have reduced their chamber-space to 15 cu. ft. per lb. of sulphur per 24 hours. The fan system of working chambers is extending, and 25% increases of output are stated by Gilchrist to have been obtained in this way. The wear and tear on the lead chamber is, however, greatly increased, and the sides of the chamber may bulge, unless extra support has been provided when erecting them. The Pratt system of intermediate towers is stated to have led in some cases to a reduction of the chamber-space to 12 or 14 cu. ft. As already pointed out, the reduced chamber-space may not be entirely due to these additions to the plant; but may result in part from the more scientific supervision of the process.

According to Davis* a plant having none of these devices was, in 1894, working in England with only 14 cu. ft. per lb. sulphur burnt per 24 hours. The plant consisted of three chambers; the pyrites consumption was 63 tons per week, and the niter consumed was only 1·8%—presumably on the ore.

The remainder of Gilchrist's paper deals with the device known as a "Pipe-column," first employed by himself and Hacker in America in 1892. A detailed description of an acid-making plant at Norfolk, Va., provided with these columns, is given. The plant was built to produce 30 tons of acid per day, but by use of pipe-columns, the output has been increased one-third. The chamber-space is now between 10·87 and 12·5 cu. ft. per lb. of sulphur burnt, and the niter consumption is 4% on the sulphur. This latter, it will be seen, is high; and it confirms the opinion that beyond a certain point, increased output, whatever devices may be adopted to obtain it, necessitates increased consumption of niter.

Hasenclever, in his annual report upon the German Sulphuric Acid Industry,† states that the newer forms of acid chamber are hexagonal in section instead of square; this change leading to a saving in lead. The Glover and

* *Journal of the Society of Chemical Industry*, December, 1894. † *Moniteur Scientifique*, May, 1899.

Gay-Lussac towers are, for the same reason, being built with round instead of square sections. The elevation of the acid chambers on tall columns in order to save ground space, is also being discarded. The remainder of the report deals with the new "Catalytic" methods of acid production; but it contains no information beyond that given in the special articles by Lunge, and Falding, in Vols. VI. and VII. of this Annual.

During the past year the completed specifications of the English patents, Nos. 15,947-48-49-50, granted to the "Badische Anilin und Soda Fabrik" for improvements in the manufacture of sulphuric anhydride, have been published. The applications are all dated July 21, 1898. Patent No. 15,947 is an omnibus patent, and relates, firstly, to the removal of the pyrites dust, arsenic, phosphorus, and sulphuric acid, always present as impurities in the burner gases; secondly, to the regulation of the temperature at which the reaction between the oxygen and sulphur-dioxide occurs; and thirdly, to the arrangement of the platinized asbestos in the contact apparatus.

Patent No. 15,948 relates in greater detail to the first of these, and Patent No. 15,949 to the second. The last of these four patents relates to the method of arranging the platinized asbestos in the apparatus "so that it is capable of being effectively cooled, and that the layers do not press one upon the other, and are yet traversed completely by the gases passing through the apparatus." A detailed account of the apparatus and process employed by this company also appears in the journal named below.*

Lunge, in an address recently delivered before the Liverpool Section of the Society of Chemical Industry,† referred to the progress of these new methods of acid manufacture, and to the fact that it is now possible to obtain a condensation of 98% of the gases passing through the apparatus, as compared with 67% in 1897. The better control of the temperature at which the reaction occurs, and the removal of the deleterious impurities of the burner gases, are chiefly responsible for this great improvement in efficiency. According to the same authority, the removal of the heat of the reaction is now effected in such a manner that the entering gases are heated to the point necessary for the reaction. The contact apparatus, when once heated and started, is therefore automatic and continuous in action. In summing up the present and future prospects of this new method of manufacturing sulphuric acid, Lunge expressed himself as follows: ". . . there seems to be no doubt that the catalytic processes have a decided advantage over the chamber process, where acid of a higher concentration than Glover acid, or of special purity, is required. Proof of this is the fact that one after the other, the great German Color Works, where mainly strong acid is used, have introduced the new processes, or are making preparations for doing so. But even here some comfort remains to the owners of lead chambers, or glass or platinum retorts, and the like. Where the initial cost of plant has been written off to a great extent during former better years, and especially in small-sized works, they will in the face of the cost of new plant, and of the royalties to be paid for the processes, be able

* *Moniteur Scientifique*, September, 1899.

† Hurter Memorial Lecture, *Journal of the Society of Chemical Industry*, October, 1899.

to hold on to their existing plant for a number of years. The first to change will probably be those works where much waste acid is to be disposed of and reconcentrated, as, for instance, in the manufacture of explosives."

In bringing this report of Progress in the Sulphuric Acid Industry to a close, the writer may remark that the economy of many of the new devices for increasing the output of acid chambers is not yet proved, and that further information is required as to the best results obtained in the same set of chambers over long periods of time, with and without these additions to the plant. The amount of wear and tear on the chambers under each system of working is also exceedingly difficult to estimate; and it is evident that if this is proportionately increased, the only saving effected by increased output is in management and general charges. But the proportion of these debited to the sulphuric acid account in large works, is small; and the increased consumption of niter which in most cases seems to accompany increased output, may more than swallow up the sums saved under this head.

With regard to the use of fans and other devices for regulating the draft of acid chambers, the writer believes that in time their use will become general; not only for acid plants using the Gay-Lussac tower as a chimney, but also for those connected to smokestacks with considerable draft power. Many of the mysterious disturbances which at times affect the chamber process are undoubtedly due to irregularities in the pull on the chambers; and much more uniformity in chamber working may be expected when automatic dampers are the rule, and not, as at present, the exception, in sulphuric acid works.

Finally, with reference to the catalytic processes of acid manufacture, the position appears to be somewhat similar to that now obtaining in another branch of the chemical industry—the alkali industry. In this, the older Leblanc process for the production of alkalies and chlorine is challenged by the newer electrolytic processes; but the displacement of the older method of manufacture is slow and it seems probable that for many years to come the Leblanc Alkali Works will continue to operate, though with diminished profits.

Similarly, the adoption of the new catalytic methods of acid manufacture will be gradual, as Lunge has pointed out, and the chamber process, which has supplied the world with sulphuric acid for over 100 years, will still be carried on in the twentieth century in many localities.

LIQUID ANHYDROUS SULPHUROUS ACID (SO_2).

BY FRANZ MEYER.

THE series of American chemical industries which owe their origin to metallurgical processes, has recently been increased by a new process,—the manufacture of liquid anhydrous sulphurous acid (SO_2).

At the present time this acid is made in the United States by one concern only, the Somerset Chemical Co., at Bound Brook, N. J., which uses Haenisch and Schroeder's patented process. This company uses brimstone to obtain the necessary sulphur gases, but gases derived from roasting sulphides of any kind can be utilized advantageously as long as their content of sulphurous acid does

not fall below about 4%; in fact most of the large European sulphurous acid plants make use of the gases produced by roasting zinc blende.

The principle of the Haenisch-Schroeder process is as follows: The sulphurous acid contained in the roasting gases is absorbed by water, and from this weak aqueous solution the sulphurous acid is driven off by steam heat applied in so perfect a manner that its latent heat is largely utilized and only a relatively small quantity of fuel is required for the entire process. The apparatus and the manner of working is briefly outlined as follows: The roasting gases pass through a flue, having its cover cooled by flowing water, into the bottom of an absorbing tower and rise against a spray of water in which the sulphurous acid is absorbed. The spray is so regulated that the amount of water is just sufficient to take up all the sulphurous acid. The unabsorbed gases, consisting mainly of oxygen and nitrogen, are discharged from the top of the tower.

The aqueous solution of sulphurous acid, of a strength depending upon the temperature of the absorbing water and on the percentage of sulphurous acid contained in the roasting gases, is led through a system of surface- and jet-condensers in which it is preheated by a mixture of steam and sulphurous acid continuously developed from this solution in a subsequent state of the process. After leaving these condensers the solution enters an apparatus built on the principle of countercurrent, in which it is continuously heated by a stream of water freed from its sulphurous acid, and in which it is heated up to about 195° F., while the heating water flows to waste from this apparatus at about 140° F. Finally the solution enters a digester in which all of the sulphurous acid is driven off by direct steam. This can be performed only by raising the temperature of the solution to the boiling point, but as this temperature is only 17° F. above that of the preheated solution it takes a relatively small quantity of fuel to do the work. The boiling water, now entirely freed from its sulphurous acid, passes again through the countercurrent apparatus as described above.

The mixture of sulphurous acid and steam, as it is continuously developed in the countercurrent apparatus and in the digester, is passed through the above-mentioned system of jet- and surface-condensers but in the opposite direction to that taken by the cold water solution, the latter condensing the steam and thereby separating the sulphurous acid from the mixture.

This effect is accomplished principally in the jet-condensers into which a spray of the cold aqueous solution is led directly from the absorbing tower. In order to prevent the condensed water from carrying away too much sulphurous acid, the temperature in the apparatus is kept near the boiling point, and for a proper condensation of all the steam the distance which the mixture of sulphurous acid and steam has to travel must have sufficient length. To fulfill these conditions the jet-condensers are operated in the following manner: At the start much sulphurous acid is absorbed by the spray of cold aqueous solution, but as this meets the hot mixture of steam and sulphurous acid its temperature naturally rises until it finally reaches the boiling point. The content of sulphurous acid proportionately decreases until it is practically zero at 212° F. If, therefore, the spray is regulated so that the discharged solution shows from 203 to 212° F. it is evident that it will not carry away appreciable amounts of sulphurous acid.

For instance at 203° F. the content of sulphurous acid in the solution is only about 0·5% and this small percentage of sulphurous acid is caught by leading the solution into the digester.

From the jet-condensers the sulphurous acid gases, still containing a small amount of moisture, pass through the surface-condensers, in which they are cooled by the cold aqueous solution of sulphurous acid from the absorbing tower and thereby freed from moisture, the last trace of which is finally taken out by a spray of concentrated sulphuric acid. The pure dry sulphurous acid gas is liquefied by a compressor, acting in connection with a water-cooled condenser, and is stored in large wrought-iron vessels, equipped with pressure gauge, safety valve and an arrangement for withdrawing the liquid acid.

The pressure necessary for liquefying the gaseous sulphurous acid depends upon the temperature of the cooling water and is respectively: 0·53 atmospheres for 32° F.; 1·26 atmospheres for 50°F.; 2·24 atmospheres for 68° F.; 3·51 atmospheres for 86° F.; 5·15 atmospheres for 104° F.

The liquid acid is shipped in wrought-iron drums, each equipped with a special valve for the withdrawal of the acid either in liquid or gaseous form. If the vessels are placed horizontally, liquid sulphurous acid can be drawn off, and if the removal of gaseous acid is desired, the cylinders must be set upright. The handling and shipping of these drums is not connected with danger if ordinary care is taken with them, as they are subjected to a hydraulic pressure of 450 lb. per sq. in. before use, while the highest pressure exerted under atmospheric conditions is 80 lb. as stated above.

Liquid anhydrous sulphurous acid is used for various purposes in the arts and the following brief summary gives several of its more important applications:

For Refrigerating.—Liquid sulphurous acid for ice machines was used first in France in Pictet's process, which is based on the fact that liquefied sulphurous acid when evaporated under atmospheric pressure, creates a temperature of 14° F. and if this evaporation is done in pipes surrounded by brine, the latter will be cooled to this temperature and can therefore be used for producing cold air or ice.* The gaseous sulphurous acid is continuously liquefied by a compressor and is again evaporated, etc. As the anhydrous sulphurous acid is used repeatedly the refrigerating industry is not a large consumer of sulphurous acid, the small leakage losses only requiring replacement.

For Wood Pulp.—A large quantity of liquid anhydrous sulphurous acid is used by those European wood-pulp mills which employ the sulphite process and as its use for making bisulphite of lime solutions offers so many advantages over the use of roasting gases, it is probable that the paper manufacturers in this country will soon adopt the product for this purpose. With the aid of the liquid sulphurous acid, lyes of the same proportion of sulphurous acid and lime may be made at all times, a condition that is very difficult, if not impossible in hot weather, when roasting gases are used. Furthermore the apparatus for making bisulphite solutions with liquid sulphurous acid are much simpler and cheaper than those used with brimstone or pyrites.

For Beet Sugar Factories.—In this industry the liquid sulphurous acid is

* See article on "Liquid Air," elsewhere in this volume.

used for saturating sugar juices and its consumption for this purpose is increasing steadily. It does away with the sulphur burners and the loss of sulphurous acid connected with the use of brimstone. The supply of gaseous sulphurous acid can be stopped or started at any time by closing or opening the valve on the cylinder, and as the acid is always under pressure in the drums, the gas is forced through the sugar liquors without the need of a pump.

For Disinfecting.—The disinfecting properties of sulphurous acid have been known for a long time, and aqueous solutions of the gas, or solutions of bisulphite of lime, are generally employed for this purpose, but liquid sulphurous acid may be advantageously used in place of them as, for instance, in disinfecting beer and wine barrels. If it is preferred to use the chemicals mentioned above, they can easily be produced on the spot, by leading the gas from the cylinders into water or into milk of lime as long as it is absorbed. The use of liquid sulphurous acid for disinfecting rooms and clothing is undoubtedly preferable to that of burning sulphur, as it is neither inflammable nor does it destroy the materials to be disinfected by sublimed sulphur or by the formation of sulphuric acid.

For Bleacheries.—In this industry the use of liquid sulphurous acid offers similar advantages. For bleaching, the acid can be used either in gaseous or liquid form, and the bleachers are able to produce their own sodium sulphite by simply leading the gas into a solution of soda until it is saturated. As an anti-chlor the sodium sulphite is preferable to sodium hyposulphite, as it does not precipitate sulphur in the fiber.

For Glue and Gelatine Factories.—These plants are using aqueous solutions of sulphurous acid for disinfecting and bleaching purposes. A patented process invented by Dr. Max Schroeder, of Germany, is operated by the Somerset Chemical Co., at Bound Brook, N. J., in which the pure gaseous sulphurous acid is used for the manufacture of glue and gelatine. A light-colored, clear and sweet product results from this process, while the bone meal obtained as a by-product is a very valuable fertilizer.

These are the main applications of anhydrous liquid sulphurous acid. Of minor importance in the arts are its qualities as a solvent for many organic bodies and its reducing power for chemical and metallurgical processes.

A new use for liquid sulphurous acid is pointed out by some interesting and promising experiments, which Prof. Josse reports in the *Mittheilungen aus dem Maschinen-Laboratorium der Kgl. Technischen Hochschule zu Berlin,* II Heft, 1899. Based on the patents of Behrend and Zimmermann he experimented with liquid sulphurous acid as a medium for increasing the efficiency of steam engines by the utilization of the heat of the exhaust steam for evaporating the liquid sulphurous acid; the high-pressure vapors so created were used for driving an auxiliary engine, and finally after having thus generated power, were again liquefied in a cold-water surface-condenser and used over and over again.

Prof. Josse experimented with a 40 h. p. condensing engine and found that an added efficiency of 56% was gained by connecting a sulphurous acid motor to the engine, without the use of additional steam, a result which may revolutionize the construction of steam-power plants.

THE MANUFACTURE OF SULPHURIC ACID STRONGER THAN CHAMBER ACID.

BY F. J. FALDING.

THE generic term sulphuric acid covers (*a*) a series of the hydrates of sulphur trioxide (SO_3), or certain definite combinations of water and SO_3, which possess certain physical characteristics that entitle them to be considered as definite chemical compounds, and (*b*) a series of solutions of sulphur trioxide (SO_3) in water, of convenient strengths for various uses in the arts.

It appears an easy matter to avoid the complicated and indefinite terminology that has been applied to these various hydrates and solutions, by the use of a simple expression of their values in terms of percentages by weight of sulphuric anhydride or sulphur trioxide (SO_3) contained in such hydrate or solution.

This simple method of expressing the values of the acids would probably have been used had sulphuric acid always been prepared by direct synthesis from water and sulphur trioxide, but until a comparatively recent date, the foundation of the manufacture of all commercial sulphuric acid was the chamber process, and, as a natural result, the nomenclature of these hydrates and solutions represents the history and evolution of the various manufacturing processes.

The terms in use are not exact, and often cover a wide range, so that there is a tendency toward confusion. In different countries, and even in different factories in the same country, the expressions do not represent the same percentage solution of sulphur trioxide.

Chamber Acid, Pan Acid, Oil of Vitriol, Concentrated, Extra Concentrated, Distilled, Nordhausen, Fuming, Solid Oleum, and even acids designated by the degrees of various hydrometric scales, which represent various values or degrees of strength in their places of manufacture, can all be ultimately expressed with absolute simplicity and accuracy in terms of percentage contents by weight of sulphur trioxide.

The following table exemplifies the above characteristics and gives some of the hydrates and principal commercial solutions.

The first sulphuric acid was probably distilled as trioxide from the metallic sulphates and was prepared synthetically by the addition of water. While it is very probable that in the near future all sulphuric acid will be made synthetically by the contact or catalytic process, up to the present time almost the entire manufacture of sulphuric acid has been by the chamber process and its adjuncts, the Nordhausen and freezing process.

An examination of the diagram discloses, with tolerable accuracy, the various methods employed in the manufacture of the different hydrates and solutions.

These operations are roughly divided as follows:

I. CHAMBER PROCESS.—(*a*) Purification of chamber acid.

(*b*) Evaporation or distillation. 1. Entirely, or in part by heat of burner gas. 2. Entirely by extraneous heat.

(*c*) Auxiliary processes. 1. Freezing process for production of monohydrate. 2. Nordhausen process for the production of "fuming" or acids stronger than monohydrate.

THE PRINCIPAL HYDRATES AND COMMERCIAL SOLUTIONS OF SULPHUR TRIOXIDE, OR SULPHURIC ANHYDRIDE (SO_3).

(Molecular Weight, $S=32+O_3=48=80$).

Names of Acids and Solutions.	Per Cent. by Weight of		Per Cent. by Weight of		United States Beaumé.	European Beaumé.	Twaddell.	Specific Gravity at 60° F.	Physical Description and Condition, etc.
	H_2O.	SO_3.	H_2O.	H_2SO_4.					
The chamber acids (commercial solutions)	51·54	48·46	40·68	59·37	48·0	47·8	99·0	1·495	
	49·24	50·76	37·82	62·18	50·0	49·8	106·0	1·526	
	46·80	53·20	34·83	65·17	52·0	51·9	111·8	1·559	
	44·28	55·72	31·74	68·26	54·0	53·8	111·6	1·593	
The second, or triple hydrate ($H_2SO_4+2H_2O$, or SO_3+3H_2O)	40·30	59·70	26·87	73 13	57·4	57·1	131·0	1·655	On mixing strong vitriol with water down to this point a contraction from 100 volumes to 92·14 volumes takes place.
The Glover tower and lead-pan acids (commercial solutions)	36·65	63·35	22·40	77·60	60·0	59·7	141·2	1·706	
	33·59	66·41	18·65	81·35	62·0	61·7	149·4	1·747	
	30·07	69·93	14·33	85·67	64·0	63·7	158·0	1·790	
The first, or double hydrate ($H_2SO_4+H_2O$, or SO_3+2H_2O)	31·04	68·96	15·52	84·48	63·5	63·2	156·0	1·780	Large, clear, hexagonal columnar crystals with six-sided end faces; melt at 80° C. (Lunge).
U. S. oil of vitriol	23·67	76·33	6·50	93·50 (93·43)	66·0	65·7	167·0	1·835	Thick, oily liquid—United States commercial oil of vitriol, or concentrated sulphuric acid.
Rectified, or extra concentrated acids	21·96	78·04	4·40	95·60	.....	65·85	168·0	1·840	Hydrometers should not be used for stronger solutions.
	21·67	78·33	4·05	95·95	.....	.....	.....	1·8405	The so-called "98 per cent." acid used for making "mixed" sulphuric and nitric acid for manufacturers of high explosives.
	20·82	79·18	3·00	97·00	.....	.....	.....	1·8410	
Distilled acid	19·59	80·41	1·50	98·50	.....	.....	.....	1·8420	(Richter.)
Monohydrate (H_2SO_4, or SO_3+H_2O)	18·37	81·63	0·00	100·00	.....	.....	.....	1·8384 1·8372	(Lunge) (Richter) Limpid, colorless, oily fluid; solidifies at about 0° C. into large plate-shaped crystals; at moderate temperatures it dissociates into H_2O+SO_3, and consequently, cannot be obtained by boiling down its solutions. It must be prepared by freezing from its strongest solution (Lunge process), or by synthesis from water and sulphur trioxide (H_2O+SO_3).
		SO_3 found by Titration.	Per Cent. by Weight of						
			SO_3.	H_2SO_4.					
Fuming, or Nordhausen acids	16·53	83·47	10·0	90·0	.....	.....	.....	1·8850	The Nordhausen, or fuming acids, are thick, oily liquids, strongly fuming in air, formerly prepared by dehydrating ferrous sulphate. They are now prepared by the direct synthesis of water or dilute acid and SO_2—the sulphur trioxide being produced from "burner [gas" by catalysis.]
	14·69	85·31	20·0	80·0	.....	.....	.....	.....	
	14·46	85·54	.....	.....	.....	.....	.....	1·9250	(Lunge & Naef.)
	14·44	85·56	21·4	78·6	.....	.....	.....	Crystal	An intermediate hydrate consisting of thin transparent prism crystals; fumes in air; melt at 26° C. (Lunge).
	12·86	87·14	30·0	70·0	.....	.....	.....	.....	
	12·54	87·46	.....	.....	.....	.....	.....	1·9362	
	12·00	88·00	.....	.....	.....	.....	.....	Crystal	
	11·02	88·98	40·0	60·0	.....	.....	.....	.....	
Pyrosulphuric, or Disulphuric acid, or solid oleum ($H_2S_2O_7$, $H_2SO_4+SO_3$, or H_2O+2SO_3)	10·11	89·89	45·0	55·0	.....	.....	.....	Crystal	Transparent crystalline mass, melting at 35° C.; decomposes at moderate heat into H_2SO_4 and SO_3. Is sold in Europe as an article of commerce for producing the monohydrate and fuming acids.
Solutions	9·18	90·82	50·0	50·0	.....	.....	.....	.....	
	7·85	92·65	60·0	40·0	.....	.....	.....	.....	
	5·51	94·49	70·0	30·0	.....	.....	.....	.....	
Intermediate hydrate ($H_2SO_4+2SO_3$, H_2O+4SO_3)	5·31	94·69	71·0	29·0	.....	.....	.....	.....	(R. Weber.)
Solutions	3·67	96·33	80·0	20·0	.....	.....	.....	.....	Sulphur 40% + oxygen 60% by weight. When absolutely pure and free from water, it is a colorless mobile liquid of sp. gr. 1·940 at 16° C. When cooled, solidifies in long, transparent prismatic crystals resembling pure nitrate of soda. When impure and containing a little water, it is a mass of opaque white asbestos-like crystals, melting at about 50° C. In the melted state it is less oily than oil of vitriol, and if pure, colorless. (Weber.)
Sulphur trioxide, or sulphuric anhydride (SO_3)	1·84	98·16	90·0	10·0	....	.....	.....	.....	
	0·00	100·00	100·0	0·0	.....	.....	.....	.....	

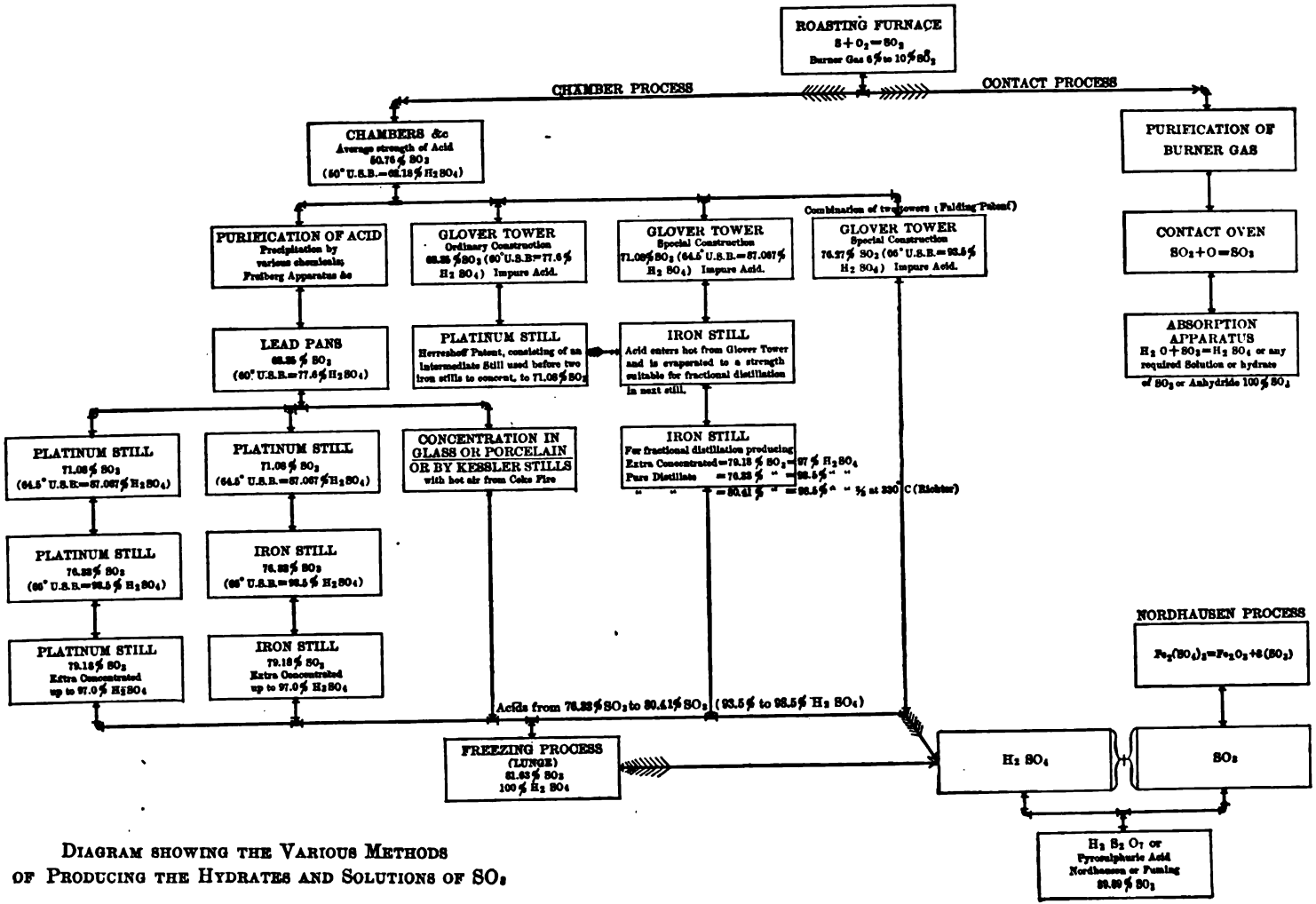

ROASTING FURNACE
$S + O_2 = SO_2$
Burner Gas 6% to 10% SO_2
CHAMBER PROCESS
CONTACT PROCESS
CHAMBERS &c
Average strength of Acid
50.76% SO_3
(50° U.S.B. = 66.18% H_2SO_4)
PURIFICATION OF BURNER GAS
CONTACT OVEN
$SO_2 + O = SO_3$
ABSORPTION APPARATUS
$H_2O + SO_3 = H_2SO_4$ or any required Solution or hydrate of SO_3 or Anhydride 100% SO_3
PURIFICATION OF ACID
Precipitation by various chemicals, Freiberg Apparatus &c
GLOVER TOWER
Ordinary Construction
66.26% SO_3 (60° U.S.B. = 77.6% H_2SO_4) Impure Acid.
GLOVER TOWER
Special Construction
71.08% SO_3 (64.5° U.S.B. = 87.067% H_2SO_4) Impure Acid.
GLOVER TOWER
Special Construction
76.27% SO_3 (66° U.S.B. = 93.5% H_2SO_4) Impure Acid.
Combination of two towers (Falding Patent)
LEAD PANS
66.26% SO_3
(60° U.S.B. = 77.6% H_2SO_4)
PLATINUM STILL
Herreshoff Patent, consisting of an intermediate Still used before two iron stills to concent. to 71.08% SO_3
IRON STILL
Acid enters hot from Glover Tower and is evaporated to a strength suitable for fractional distillation in next still.
PLATINUM STILL
71.08% SO_3
(64.5° U.S.B. = 87.067% H_2SO_4)
PLATINUM STILL
71.08% SO_3
(64.5° U.S.B. = 87.067% H_2SO_4)
CONCENTRATION IN GLASS OR PORCELAIN OR BY KESSLER STILLS
with hot air from Coke Fire
IRON STILL
For fractional distillation producing
Extra Concentrated = 79.18% SO_3 = 97% H_2SO_4
Pure Distillate = 76.23% " = 93.5% " "
" " = 80.41% " = 98.5% " ½ at 330° C (Richter)
PLATINUM STILL
76.23% SO_3
(66° U.S.B. = 93.5% H_2SO_4)
IRON STILL
76.23% SO_3
(66° U.S.B. = 93.5% H_2SO_4)
PLATINUM STILL
79.18% SO_3
Extra Concentrated up to 97.0% H_2SO_4
IRON STILL
79.18% SO_3
Extra Concentrated up to 97.0% H_2SO_4
Acids from 76.23% SO_3 to 80.41% SO_3 (93.5% to 98.5% H_2SO_4)
NORDHAUSEN PROCESS
$Fe_2(SO_4)_3 = Fe_2O_3 + 3(SO_3)$
FREEZING PROCESS
(LUNGE)
81.63% SO_3
100% H_2SO_4
H_2SO_4
SO_3
$H_2S_2O_7$ or Pyrosulphuric Acid Nordhausen or Fuming
60.80% SO_3

DIAGRAM SHOWING THE VARIOUS METHODS OF PRODUCING THE HYDRATES AND SOLUTIONS OF SO_3

II. CONTACT PROCESS.—(*a*) Purification of gas.

(*b*) Direct conversion of SO_2 into SO_3 by catalysis.

(*c*) Collection of the SO_3 or its dilution with water to form hydrates or solutions of any required strength.

Within the limits of a paper of this character it is impossible to describe fully all the apparatus employed in the various processes. In many cases the difference in form, material of construction, and other details are but slight.

It is intended to emphasize such changes in the processes or in methods of construction as seem to constitute more or less radical improvements, and especially to emphasize points of difference between the practice in the United States and in Europe, where there appears to be a distinct advantage in favor of one or the other.

The manufacture of chamber acid has already been discussed in THE MINERAL INDUSTRY,* and to take up the matter of its concentration by evaporation, approximately in chronological order, brings us first to the subject of purification.

The Purification of Chamber Acid.—With the exception of lead sulphate, nearly all the impurities of chamber and sulphuric acid are brought into the system by the burner gas, and depend largely on the raw material of manufacture.

These impurities consist of various oxides and acids of arsenic, antimony, selenium, thallium, lead, iron, zinc, copper, lime, aluminum, mercury, fluorine, together with nitrous and nitric acid, nitric oxide and organic matter.

All of these impurities will hardly exist in acid at the same time; some are not deleterious for many uses in the arts, and most of them are mechanically deposited by sedimentation in the Glover tower and the first chamber, so that when acid is made from brimstone or from a reasonably pure pyrite, it is possible by a judicious selection of the acid formed in the back chambers to secure a sufficiently pure acid for the most exacting manufacturing purposes, and to collect the more impure acid for less exacting uses.

There is a great advantage in the cost of production of acid when obtained by using the waste gas from desulphurizing the more impure metallic sulphide ores, especially zinc blende, arsenical copper sulphides and "fine" ore of various kinds; therefore, the more and more exacting requirements of customers as regards purity, make it desirable to purify the entire output of acid.

As stated above, the Glover tower and sedimentation in the first chamber take care of many of these impurities, still some remain in solution or in such a form that they will not precipitate, and these require a chemical treatment for their removal. The most troublesome of these impurities are arsenic and selenium. The latter can be readily precipitated by the addition of small quantities of an aqueous solution of potassium chlorate. Arsenic, however, usually exists, not only in larger quantity, but unless practically eliminated at this stage of the concentration, it will contaminate all products manufactured from the acid, rendering such products unsalable or making it necessary to purify them in turn.

* THE MINERAL INDUSTRY, Vol. VII., p. 651.

Comparatively small quantities of acid can be freed from arsenic by the method described by Dr. Stahl,* but for plants of large size I know of no better nor cheaper method of purification for the entire output of a plant than the Freiberg system of the precipitation of the arsenic as sulphide by means of sulphuretted hydrogen (H_2S), or rather a modification of this system used in this country, in which the sulphuretted hydrogen is a by-product, as is the case where the chemical works include the treatment of gas liquor for aqua ammonia or ammonium sulphate. Under these conditions this process is quite cheap. When sulphuretted hydrogen is not available from such a source, the process is still cheap. The sulphuretted hydrogen gas in this case is prepared from sulphuric acid, H_2SO_4, and ferrous sulphide, FeS. The ferrous sulphide is readily prepared on a large scale from pyrites by treatment in a water-jacketed cupola.

The construction of the Freiberg system that is used in the United States differs in several respects from the process abroad.

In Europe the process consists principally of a tower in which the descending acid meets the ascending sulphuretted hydrogen gas. As the acid flows from the tower it is pumped into a vacuum filter, which removes the precipitated arsenic sulphide.† In America the acid flows by gravity from the chambers into a series of lead-lined, gas-tight boxes or tanks connected with a tower similar in construction to the Freiberg tower.

The gas is forced through the acid in the boxes by means of a perforated pipe on the bottom and only enough acid is run down the tower to prevent the escape of any of the gas. In this way, economy in pumping is effected. The pipes for the sulphuretted hydrogen gas are connected to a sufficient number of boxes in such a manner that the gas can be made to pass through any one of the boxes first, and any one of the boxes may be disconnected from the series at any time. In this way when box 1 in the series has had sufficient treatment by the gas, it may be cut out, and box 2 takes its place, and so on. The box so cut out is allowed to settle for from 24 to 48 hours, and the arsenic sulphide is so completely precipitated that probably three-fourths or more of the supernatant acid may be decanted in a pure state, and will not require further treatment. Each series of boxes is provided with two simple gravity filters, which consist of a lead-lined box filled with broken quartz of graduated sizes. The acid which remains in the box after the clarified acid has been decanted, is run by gravity over one of the filters, and another pumping thereby saved. In the course of a few days or weeks, when one filter has become foul, the other filter is put into service and the foul one cleansed.

With apparatus arranged in this way, the purification of sulphuric acid by the removal of arsenic becomes an inexpensive matter, and the output of the Freiberg filter thereby very largely increased.

Pan Concentration or Evaporation to 60·7° B. (144 Tw. 64·43% SO_3).—The acid from the filter is run into a succession of lead pans arranged so as to furnish a flow by gravity from one to another. In these pans it is concentrated to 60·7° B. (144 Tw. 64·43% SO_3). If carried beyond this degree of concentration

* Lunge, *Sulphuric Acid and Alkali*, Vol. III., 2d Edition, p. 786. † *Ibid.*, Vol. I., 2d Edition, pp. 642 to 649.

the acid acts too strongly on the lead, and the loss of acid by evaporation is considerable.

Formerly these pans were placed over the brimstone or pyrites burners, the waste heat being utilized for concentration. On account of the dirt arising from the use of pyrites, and on account of the difficulty of making repairs to the pans when so placed, and the serious damage to the burners by escaping acid, this plan has been generally discarded. The introduction of the Glover tower, moreover, utilized the waste heat of the burners to better advantage.

The pans are now fired separately by coal, gas or oil, or, in exceptional cases by steam coils. They are of two general types; the one is heated at the bottom, and the other is arched over, and heated by the gas which passes over the surface of the acid.*

Concentration of Pan Acid.—The pan acid 60·7° U. S. B. (144 Tw. 64·43% SO_3) is further concentrated in glass, porcelain or platinum apparatus. While glass retorts are now made better and cheaper, and the percentage of breakage has been greatly reduced, still the use of glass concentration has been abandoned in the United States. When employed abroad the acid is finally concentrated in glass to 66·° U. S. B. (167 Tw.,76·33% SO_3). No progress has been made in this country with the use of the porcelain dishes of Negrier and other systems.†

In platinum, however, large quantities of acid have been concentrated. The tendency is to use platinum for a partial concentration to about 64·5 to 65° U. S. B. (71·08% SO_3) and to make the final concentration in iron stills. The acid of these higher strengths has but little effect on the iron.

At first the iron stills were of the same size as the ordinary circular dish still of platinum, and had a similar water-cooled lead hood. Usually in a set of three or four stills in series, the first one or two were of platinum, and in such a set the pan acid was finally concentrated as desired to 76·33% SO_3 or to 79·18% SO_3 (93·5 or 97% H_2SO_4). Later, oblong stills were constructed of larger size, and with cast-iron domes, sometimes lined with firebrick. These stills gave good results in economy and capacity.

The above method of concentration does not represent the best American practice, and except as to first cost of plant it is doubtful whether the substitution of iron stills for some of the platinum ones has effected any very considerable economy.

Modern American Practice.—The general introduction of the Glover tower early called attention to the fact that by its use the whole output of chamber acid could be concentrated from 50° B. to 62° B. (50·76 to 66·41% SO_3). The acid so obtained, however, was too impure for general use, as an article of commerce.

The advantages of the tower method of concentration are great, the heat generated by the combustion of the brimstone or pyrite is sufficient for the entire evaporation, and all the steam thereby produced is returned to the system, where needed in the leaden chambers, and thus the usual cost of water evaporation for that special purpose is saved; furthermore, all weak acid distillates are returned to the system without loss or additional cost for concentration.

* Lunge, *Sulphuric Acid and Alkali*, 2d Edition, pp. 659–680. † *Ibid.*, p. 693.

The only drawback of this method is the impurity of the acid produced and the difficulty encountered in its purification by precipitation at such density. With such a preponderance of advantage, it is not to be wondered at that efforts were made in various directions toward the production of a commercial concentration by means of the Glover tower.

The first successful efforts date from 1886, when J. B. F. Herreshoff patented an improved construction of the Glover tower.* By this construction many impurities formerly derived from the materials of construction employed were eliminated from the acid, and a higher concentration in the tower became possible.

The next step also was made by Herreshoff,† who began operating with Glover tower acid of ordinary strength, 60° U. S. B., and first concentrated this acid in platinum to about 86% H_2SO_4 (about 64·25° U. S. B.). Then in an iron still, further concentrated it to about 95% H_2SO_4, or some degree of strength higher than 93·5% H_2SO_4 (U. S. B. 66°), and finally in a second iron still, evaporated therefrom a pure distillate of 93·5% H_2SO_4, and recovered an impure concentrate of about 98% H_2SO_4.

In 1894 the writer patented‡ a process of concentration which consisted of the exposure of the de nitrated Glover tower acid to the direct action of the hot burner gas in an independent concentration tower or apparatus. In practice, when two or more chamber systems can be worked together, a special concentration tower is unnecessary, as the Glover tower of one system can be used entirely for denitration and concentration up to 62° B., and the Glover tower of the other can be used exclusively for the concentration, up to 66°B. (76·33% SO_3), of the denitrated product of first tower. The gases from both towers, the one nitrous and the other not, are collected, thoroughly mixed by fans and distributed between the two or more sets of chambers. The Glover towers are of special construction with a lining of the best quality Volvic Lava, and a packing of selected quartz.

The tower acid produced in this manner is, of course, impure, but is quite cheap and suitable for many purposes, such as oil refining, etc.

For pure acid the process is modified as follows: The first platinum still of the Herreshoff patent is discarded, and the Glover tower acid is concentrated to 64·5° U. S. B. (71·08% SO_3). It then runs through a platinum tube directly into iron stills without any appreciable loss of heat. These stills are either constructed as solid cast-iron flasks about 8 or 9 ft. long by 2 to 3 ft. wide, or else are built with flat tops, and are set in the still bench so as to be entirely surrounded by the fire space (Figs. 1, 2, and 3). When so constructed and properly fed with acid which contains the full heat of the Glover tower, but very little additional fuel suffices to complete the process. The products are distillates of great purity, and the process is so manipulated that the impurities, including the arsenic, remain in the stills as mud. The stills, of course, have to be periodically cleaned out, although in some instances they are provided with "mud drums." The whole bench is so arranged that a still

* United States Patent No. 335,699. † United States Patents Nos. 357,528 and 369,790.
‡ United States Patent No. 541,041.

can be withdrawn, and another substituted in a very short time. The one withdrawn is then cleaned, overhauled, and made ready in turn to replace the still in commission. In this way the cost of pure high-grade acids is greatly reduced, and the life of the iron stills when properly cast is remarkable for its length.

The acids so prepared are usually very pure distillates, 60° U. S. B. (63·35% SO_3), 66° U. S. B. (76·33% SO_3), and impure concentrates of 79·18 to 80·00% SO_3 (97 to 98% H_2SO_4), although by a mere change in the firing of the stills a pure distillate of about 80·40% SO_3 (98·5% H_2SO_4), can also be prepared at will, as Richter* has already pointed out.

FIG. 1.

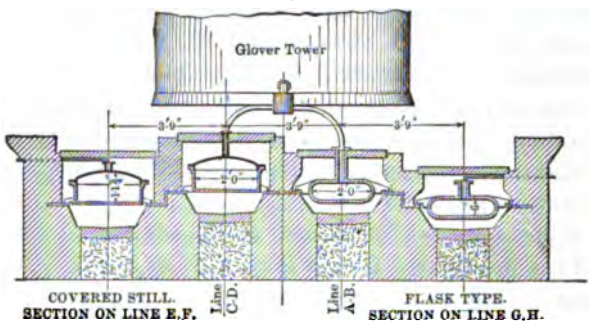


FIG. 2.

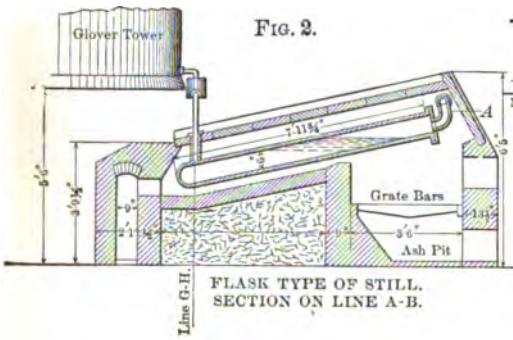
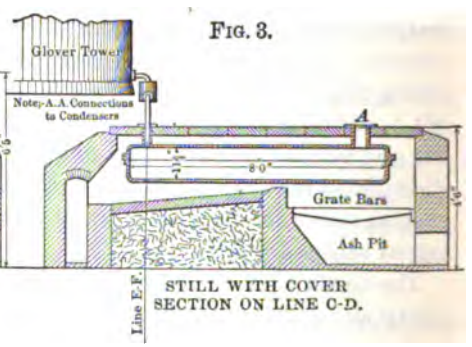


FIG. 3.



IRON STILLS FOR CONCENTRATING SULPHURIC ACID.

As by this method the original heat of combustion in the furnaces is carefully conserved throughout the whole process; and, as all distillates are returned to the chambers except such as are of commercial strength; and as the resulting acids are of great purity without any additional cost for purification, it would seem as if the progress of the economical concentration of acids by evaporation or distillation had attained its highest development. A few pounds more coal may be saved or a slightly longer life be secured for some of the apparatus, but no stronger acid can be obtained, as the application of further heat tends to dissociate the water and sulphur trioxide.

No mention has been made of the Kessler apparatus for concentration by the heat of a coke fire, as the few installations which were made in the United States did not seem to be easily handled, and did not give the hoped-for

* Richter, *Inorganic Chemistry*, 3d American Edition, p. 192.

economical results, and their use is not likely to become general. To obtain the actual monohydrate or stronger acids than 98·5% H_2SO_4, recourse must be had either to the Lunge freezing process,* whereby crystals of the actual monohydrate are recovered from the high-grade distillates or concentrates; or these high-grade acids are fortified synthetically by the addition of SO_3, prepared by the Nordhausen process.†

Neither of these processes has been exploited in this country, and on account of their cost they are being supplanted in Europe by the direct preparation of acids of all strengths by the catalytic or contact processes.

The Contact Process.—The present practice begins with the burner gas, which need not contain more than 6% of SO_2, and may be produced from zinc-blende, pyrites, brimstone or any raw material, which may be used to produce SO_2 for the chamber process. There are three distinct stages:

(1) Cooling, cleaning and drying the gas.

(2) Submitting the gas under proper conditions of temperature to the action of the contact mass, with a consequent conversion of SO_2 into SO_3.

(3) Recovering the SO_3 either as sulphur trioxide or as a hydrate or solution of the trioxide, by its absorption in water or weaker acid.

While many of the details of this process will undoubtedly be simplified and improved, both in regard to the contact mass and to the apparatus, the process itself is now far beyond the experimental stage.

The cost of production of 50° U. S. B. acid by this process, compared with its cost when made by a modern set of chambers, shows a fair margin of profit in favor of the contact process, and for acids above 60° B. this margin increases rapidly with the strength. In addition to these advantages, the smaller cost of plant and less ground space required are decidedly in favor of the contact process. Furthermore, in the contact process the same plant which produces 50° U. S. B. acid can, with very slight modification, be made to produce 66° U. S. B. —95%, 96%, 97%, 98%, or 99% H_2SO_4, the actual monohydrate, any of the fuming or Nordhausen acids, or the actual anhydride, without additional labor cost, and it is evident not only that these acids, which were formerly so costly, can in this way, be manufactured very cheaply, but also that the saving on the capital account will be very great, as it does away with the concentration plant, the freezing plant for the monohydrate, and the Nordhausen and various processes for the production of sulphuric anhydride.

In the Hurter Memorial Lecture delivered before the Liverpool Section of the Society of Chemical Industry, October 4, 1899, Prof. Lunge, who was formerly rather skeptical in regard to this process, in reviewing recent changes in the chemical industry of the world, states:

"We naturally begin with sulphuric acid, and here we are at once confronted with the greatest revolution that has taken place since that acid became a commercial product, a revolution beside which the invention of the Gay-Lussac and Glover tower, let alone that of the plate columns, and the like, sinks into insignificance. I mean, of course, the total abolition of the vitriol chamber, and even of the use of nitrous fumes as oxygen carriers, by the use of the

* Lunge, *Sulphuric Acid and Alkali.* Vol. I., 2d Edition, p. 739. † *Ibid.*, p. 839.

catalytic power of platinum, perhaps also by that of ferric oxide, and other substances . . .

"As a measure of the progress made, I may remind you that, as I wrote at the end of the year, 1897, in THE MINERAL INDUSTRY, Vol. VI., page 131:

"Probably 67% must be considered very good work, and the remaining 33% of sulphurous acid must be sent to the lead chambers together with better gas from other burners. In lieu of 67% we must *now* speak of 98%."

Catalysis is defined by Berzelius as "a decomposition and new combination produced among the proximate and elementary principles of one or more compounds by virtue of the mere presence of a substance or substances which do not of themselves enter into the reaction." It is believed, however, that bodies which cause catalysis do in some way take part in the chemical reactions involved, though they are in the course of it always brought back to their original condition. "Platinum black . . . absorbs 800 times its volume of oxygen from the air and in virtue thereof is a most active oxidizing agent which in general acts catalytically, because the black, after having given up its oxygen to the oxidizable substances present, at once takes up a fresh supply from the atmosphere."—Encyclopedia Britannica, XIX, 191.

The observed phenomena of catalysis as applied to the conversion of SO_2 into SO_3 are as follows:

(1) If pure gaseous SO_2 with its theoretical volume of oxygen (one-half a volume SO_2+O) or a mixture of atmospheric air with SO_2 are heated to a temperature of about 250° C. (482° F.) and brought into intimate contact with finely divided platinum, ferric oxide or other catalytic material, a more or less complete oxidation to SO_3 will take place, *providing the temperature of the reaction during the contact is not allowed to rise to the point of dissociation—that is, to the point where* SO_3 *will break up again into* SO_2+O. The SO_3 vapor can then be collected by cooling or as a solution in H_2O.

(2) The activity of the catalytic agent or contact mass, at least so far as regards platinum, seems to vary directly with its surface exposed. At first platinum wire was used, afterward platinum sponge—platinized pumice—platinized asbestos (Winkler patents) and finally a platinum mass consisting of crystallization of a double salt of platinum and a water soluble sulphate.* "A contact body for use in catalytic processes comprising salt crusts, containing a platinum salt distributed through a mass of one or more other soluble salts which serve as a carrier or vehicle therefor."

By the use of this last-mentioned contact substance the amount of platinum required for the process has been reduced to less than one-half of the amount required when the substance used is asbestos platinized by means of platinum black. Probably partly because of the extremely fine division of the platinum and partly because the associated sulphate also of itself possesses catalytic properties.

Other phenomena connected with the catalytic activity of platinum are set out by Lunge,† and are connected with the method of the preparation of the platinum black used.

* Grillo-Schroeder United States Patents, Nos. 636,924 and 636,925.

† *Sulphuric Acid and Alkali*, 2d Edition, pp. 863 and 864.

(3) The gaseous mixture of atmospheric air and sulphurous acid gas must be free from all impurities other than nitrogen. Arsenious acid or other arsenic compounds, fluorine, silicon and flue-dust, especially interfere with the reaction, probably by coating the contact mass and reducing the exposed surface of the active substance.

In this reference it is claimed under the Grillo-Schroeder patents that in case of weakening of the activity of the contact mass by any carelessness in purification of the gas that the contact gas can, on account of its complete solubility, be readily regenerated and without loss of platinum.

The nature of the catalytic reaction does not seem to be entirely understood. There are two theories in regard to it which after all are rather interdependent the one on the other. The first theory refers to the electro-chemical condition of the gases and their increased readiness to unite under a polarized condition brought about by the presence of the catalytic agent.

W. R. Grove* says: "I am strongly disposed to consider that the facts of catalysis depend upon voltaic action, to generate which three heterogeneous substances are always necessary."

The second theory refers to the thermo-chemical conditions and is that the catalytic substance carries off the heat of the reaction either directly or by its conversion into thermo-electrical energy.

Richter† says: "The catalytic action of many metals, *e.g.*, platinum, in the reaction $SO_2+O=SO_3$ is very likely due to their conducting off the heat. Another case is the greater reactivity of many bodies when they form a galvanic chain, because in this way the chemical energy is removed, as electricity, *e.g.*, the union of hydrogen and oxygen at ordinary temperatures due to the formation of a polarization current."

In this reference a brief mention of some of the facts of the thermo-chemistry of the oxidization of sulphur will be found interesting.

Sulphur in its formation appears to form an exception to the general rule of thermo-chemistry; that where two or more compounds are possible as the products of chemical combination, that product will be formed which produces the greatest heat in the reaction: for example, C+O can form CO or CO_2, and CO_2 is the usual product of complete combustion. S+O can form SO_2, and SO_3; yet SO_2 is the usual product of combustion.

The reason for this is that the heat of combination of S and O_3 is so great as to cause the dissociation of the SO_3 into SO_2 and O or, in other words, SO_3 at the temperature of its production is not stable unless some means can be devised to carry off the heat of the reaction effectively.

Many facts point to the probability that conversely as the intermediate product CO is formed in the combustion of carbon before the final production of CO_2, so is the first product of combustion of sulphur SO_3 which, on account of the high temperature, is then broken up into SO_2+O.

It is a well-known fact that in burning pyrites the burner gas contains percentages of SO_3 varying from 2 to 20% of the total sulphur oxides. This fact has been generally explained as being due to the presence in the burners

* *Correlation of Forces*, p. 6. † *Inorganic Chemistry*, 3d American Edition, p. 189.

of ferric oxides and their catalytic action on the gases. But the undoubted presence of varying percentages of SO_3 in the burner gas resulting from the combustion of brimstone cannot be accounted for in this way—and it would seem therefore as if the observations of Lunge, Fortmann and others, closely approximate the truth—and therefore that if means could be found to get rid of the excess of heat of combustion in the furnace itself the direct product of such combustion would be SO_3.

Lunge states as follows: "By later experiments in my laboratory it was found that in roasting pyrites by itself 5·05% SO_3 was formed when passing the gases through a layer of red-hot pyrites cinders, 15·8% of the total sulphur reappeared as SO_3. On burning brimstone it was found that even here 2·48 to 2·80% of the sulphur was converted into SO_3."

This result is confirmed by Hempel,* who further states that "when effecting the combustion under a pressure of 40 or 50 atmospheres about one-half of the sulphur was converted into SO_3."

Fortmann† states "that the whole of the anhydride fumes appear at the moment the pyrites take fire." Any one who has been accustomed to make the Reich's test for the determination of SO_2 in burner gas is familiar with the fact of the presence of increased amount of fumes SO_3 in the gas when the burners are in bad condition and cool.

The catalytic or contact processes for the manufacture of sulphuric acid which are in actual practical operation are largely due to the discoveries of two German chemists, Cl. Winkler and M. Schroeder. Reference to Cl. Winkler's discovery of platinized asbestos, which first rendered this process practical, is made in the quotation from Hasenclever in a previous paper.‡

Following Winkler's discovery Dr. Schroeder by his process of the manufacture of sulphurous anhydride made pure sulphurous acid available at a reasonable cost. In connection with the Badische Anilin- und Soda-fabrik, Dr. Schroeder first manufactured sulphuric anhydride catalytically from the pure sulphurous acid gas mixed first with pure oxygen and later with atmospheric air. Dr. Schroeder's last discovery (disclosed by the Grillo-Schroeder patents) of a soluble contact mass consisting of a double salt of platinum and an earthy or metallic salt soluble in water, whereby the quantity of platinum required is reduced, seems to be the latest practical advance.

The contact process is exploited practically by the following concerns:

1. The Badische Anilin- und Soda-fabrik.—This concern has manufactured sulphuric acid for several years in this way, as a more or less secret process, to the entire exclusion of chambers. They use platinized asbestos as a catalytic agent and lay special stress on the necessity for purifying the gas; upon their method of securing such purification; upon some of the details of the construction of the apparatus used by them, and upon their method of regulating the temperature of the reaction.

2. The Farbewerke, vorm. Meister, Lucius u. Bruning, of Hoechst-am-Main.—This concern also use platinized asbestos as contact mass. Their

* *Berichte der Deutscher Chem. Gesellschaft*, 1890, p. 1455. † *Dingler's Journal*, CLXXXVII., p. 155.
‡ THE MINERAL INDUSTRY, Vol VII., p. 695, "Manufacture of Chamber Sulphuric Acid."

patent relates to the method of utilizing the heat of the reaction for bringing up the burner gas to the initial temperature required for the reaction.

3. The Actien-Gesellschaft für Zink-Industrie at Oberhausen (Grillo-Schroeder).—This concern uses a new contact mass whereby it is claimed that less than one-half the platinum is required and that subsequent loss of platinum is practically nothing. Some eight or nine large concerns in Europe are erecting plants under the Grillo-Schroeder patents. In America the rights have been acquired by the New Jersey Zinc Co., and several plants are in course of erection or about to be erected in the United States, both by the New Jersey Zinc Co. itself at Mineral Point, Wis., and at Newark, N. J., and its licensees.

4. The Verein Chemischer Fabriken in Mannheim.*—This patent contains the following statements: "If burner gases containing arsenic are passed through a layer of material containing ferric or certain other oxides (preferably pyrites cinder) heated to red heat, the arsenic will be removed." In their Patent No. 107,995 they claim: "(1) process of manufacture of sulphuric acid by passing the gases over red-hot (glühende) cinder in such a way that the reaction takes place in immediate connection with the roasting process, so that the gases pass through the contact mass (pyrites cinder) with a temperature acquired by the roasting process; (2) the use of dry air for the burners; (3) the combination of (1) and (2) in such a way that the cinder used as contact substance is taken fresh from the burners before it has lost any of its capacity for the reaction." They state that whereas Lunge in his experiments of 1877 and 1883 could obtain a yield of only 16 to 18%, they are able to obtain a yield of at least four times that, in fact approaching the theoretical. In a still later patent they admit air to the burner gas to reduce the percentage of SO_2.

Of course the cheapness in the first cost of contact mass makes this process attractive. It does not appear, however, as yet to be in practical shape, inasmuch as the reaction obtained is not complete, and its operation is yet in the experimental stage. The three former concerns are obtaining and are willing to guarantee a total yield of from 95 to 98% of the *theoretical*, which in this respect also places the process easily on a par with the chamber process.

YIELDS OF SULPHURIC ACID SOLUTIONS, IN POUNDS AVOIRDUPOIS OR KILOS.
(Based on 100 lb. or kilos. of Available Sulphur or Sulphur actually contained in the Burner Gas as Oxides.)

Percentum of Theoretical.	Sulphuric Anhydride. SO_3 100%.	SO_3—60% / H_2SO_4—40% / SO_2—32·05%.	SO_3—40% / H_2SO_4—60% / SO_2—38·98%.	SO_3—20% / H_2SO_4—80% / SO_2—45·31%.	Monohydrate. H_2SO_4—100%. SO_2—81·632%.	Distilled. H_2SO_4—98·5%. SO_2—80·41%.	Extra Concentrated. H_2SO_4—97%. SO_2—79·18%.	Oil of Vitriol. 66° U. S. Bé. H_2SO_4—96·5%. SO_2—78·33%.	Glover Tower Acid. 62° U. S. Bé. H_2SO_4—81·35%. SO_2—66·41%.	Pan Acid. 60° U. S. Bé. H_2SO_4—77·6%. SO_2—63·35%.	Chamber Acid. 52° U. S. Bé. H_2SO_4—62·16%. SO_2—50·76%.
100	250·00	269·88	280·96	298·05	306·25	310·91	315·73	327·53	376·45	394·63	492·51
99	247·50	267·13	278·15	290·12	303·19	307·80	312·56	324·26	372·69	390·70	487·59
98	245·00	264·43	275·34	287·19	300·13	304·69	309·41	320·99	368·93	386·76	482·67
97	242·50	261·74	272·53	284·26	297·06	301·58	306·25	317·72	365·17	382·81	477·74
96	240·00	259·04	269·72	281·33	294·00	298·47	303·09	314·44	361·40	378·86	472·82
95	237·50	256·34	266·91	278·40	290·94	295·36	299·93	311·16	357·64	374·92	467·89

* German Patent No. 106,715.

The Sulphur Industry of Italy.

By Giovanni Aichino.

In Italy there are two classes of sulphur deposits, *viz.*, (1) those which have and show direct connection with volcanoes, and (2) those which occur as beds, sheets, or even as masses in stratified formations. The former are generally known as solfataric deposits, being the direct result of the last period of volcanic action; the others are known as sedimentary deposits. In Italy both kinds are found, but the only ones of commercial value belong to the second class. The solfatara of Pozzuoli (near Naples), furnishes an irregular and very limited output, and Volcano, one of the Lipari Islands, used to produce a little before the eruption in 1888. The only solfatarian deposits of practical importance are those in the vicinity of Viterbo, where there is, below a stratum of ash and volcanic scoria, a sheet of basaltic lava covering a deposit of sulphur ore. The latter is a whitish, light, porous rock impregnated with sulphur, and intersected by veinlets of it. Below this stratum, which is known as "marmorone," is a bed of cemented "lapilli," impregnated with sulphur. A test of the best "marmorone" showed 71% and of the lapilli 38% S, but in practice these figures were never realized. These deposits were worked for some time past, and then neglected. In 1891, a new company (Société Civile des Soufres Romains) took over one of these mines (the Latera), but in 1896 the mine was again abandoned. The average yield of the ore (which was treated from the beginning by steam apparatus), was at first 11·5%, falling off to only 6% in 1895. Perhaps it would be feasible to concentrate the mineral mechanically, but under existing trade circumstances the exploitation of these deposits is not at all likely.

The Sulphur Deposits of Sicily.—The deposits of Sicily are found in the sedimentary formations. The sulphur ore is always accompanied by gypsum; and strictly speaking, we ought to say that the sulphur is only accessory to the extensive formation of gypsum, since most writers refer the genesis of the sulphur to the gypsum. The formation which is called in Italy "Gessoso-solfifera," belongs geologically to the upper part of the Miocene. It occurs between the tripolis of the Sarmatian and the foraminiferous white marl of the lower Pliocene, and is extremely bare of fossils, which are found only in some beds of small extent. In addition to the gypsum and sulphur, the formation chiefly consists of clays, more or less bituminous and saline, associated with the gypsum often but slightly coherent, and a concretionary limestone forming large beds ("calcare siliceo"), between the gypsum and the tripolis. This calcarious rock plays a prominent part in the series, as it almost always accompanies the deposits of sulphur. Sometimes it is compact and very hard, with nodules of silica, and in other instances it is meal-like and friable. It is widespread throughout the sulphur regions, forming long hillocks of a peculiar character. Its thickness, which reaches sometimes 50 or 60 m., varies considerably, and generally is inversely to that of the gypsum.

Gypsum is the most common and characteristic rock of the formation; it presents itself in a great variety of structure (crystalline, saccharoidal,

amorphous, schistose, arenaceous, etc.), and occupies a vast area in the island, attaining its greatest development in the region extending from the vicinity of Catania to Gibellina, Castelvetrano and the southern coast of the island. It is chiefly in this region that the mines occur. The thickness of the gypsum is often a very few meters only, but in many cases it reaches 20 m., and sometimes as high as 80 m. As previously stated, it frequently alternates with bituminous clays ("tufi") and then the whole formation may attain even greater thickness, exceeding 400 m. as at Comitini.

The gangue of the sulphur is calcarious, argillaceous, gypseous, or mixed; but from an industrial standpoint, it may be considered chiefly of limestone more or less marly, a gangue of clay or gypsum being far less common.

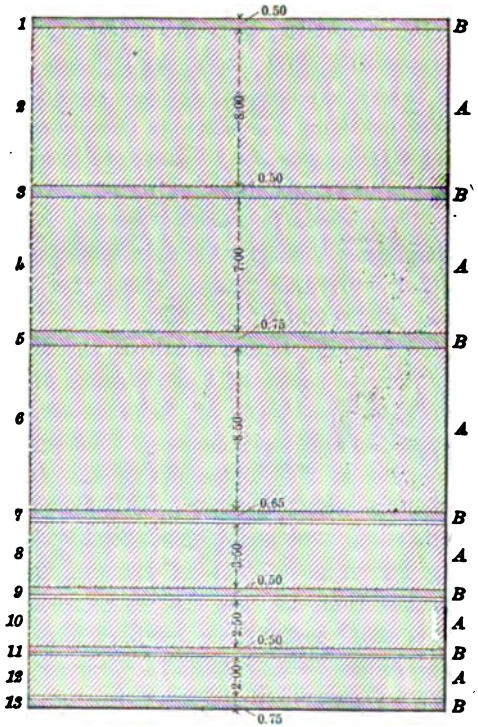


FIG. 1.—STRATIFIED DEPOSIT AT SOLFARA GRANDE DI SOMMÀTINO.

(Dimensions in meters.) *AA*, Sulphur ore. *BB*, Barren layers.

Sometimes the sulphur is disseminated in the limestone in small geodes; at other times it is found intersecting the limestone in thin seams, *i.e.*, from 4 to 5 mm. up to 2 or 3 cm. in thickness; this is known as mineral of a "listata," or "soriata" structure.

The deposit is usually formed of several beds of mineral, more or less numerous and more or less rich, which occur between sterile beds of marly limestone or clay, the thickness of these barren strata varying from a few centimeters to several meters; in some instances it is so small that the deposit might be considered as one single layer. The barren beds generally have no very great

extension and disappear at the edges of the deposits. Fig. 1 is an example of a deposit with numerous layers which exists in the Solfara Grande di Sommatino—the layers reading downward are as follows: (1) tufa, 0·5 m.; (2) ore, 8 m.; (3) sterile, 0·5 m.; (4) ore, 7 m.; (5) tufa, 0·7 m.; (6) ore, 8·5 m.; (7) marly gypsum and seam of tufa, 0·65 m.; (8) ore, 3·5 m.; (9) marly gypsum and seam of tufa, 0·5 m.; (10) ore, 2·5 m.; (11) marly gypsum and seam of tufa, 0·5 m.; (12) ore, 2 m.; (13) tufa and marly gypsum, 0·75 m. (bottom of the deposit.)

The rocks do not form a regular continuous series. The formation Gessoso-solfifera may be best described as being formed by a series of calcarious beds, marls, black clay, gypsum and sandstone.

The sulphur is accompanied by a number of minerals, sometimes finely crystallized; they are calcite, aragonite, gypsum, celestite, quartz, chalcedony and opal. The mineral, especially that with argillaceous gangue, also contains bituminous substances, which are sometimes very abundant, particularly in the marls which separate the ore from the silicious limestone.

Atmospheric agencies acting upon the sulphur ore produce a hydrated calcium sulphate, with a loose, granular structure, of light yellow or gray color, locally known as "briscale." This forms to some extent the cap of the deposits, and is of great importance in prospecting. Strictly speaking, the sulphur ore does not form true strata, but lenticular masses, that, when of large extension and moderate thickness may be spoken of as beds.

The sulphur deposits of Sicily spread south of the Madonie Mountains, and occupy almost entirely the provinces of Caltanissetta and Girgenti and a part of that of Catania. In the provinces of Palermo and Trapani (at Lercara and Gibellina) there are other deposits isolated from the main group.

Causes acting subsequent to the formation of the deposits made numerous distinct groups out of those which were originally more or less connected, and produced variable inclinations and great stratigraphic derangements, the latter without much folding but a good deal of faulting and displacement. The deposits with a dip of less than 15° are few in number; about one-third have dips from 15 to 30°; a little more than one-fifth dips from 60 to 90°. See Figs. 2, 3, 4, 5 and 6, for sections of some of these deposits.

The mineral is very irregularly distributed in the deposits, the thickness, richness and number of the beds varying considerably. Their general average as to thickness has been found to be from 6 to 7 m.; the maximum is found in the groups of Colle Groce (Lercara) where it exceeds 50 m. On the other hand there are groups of mines where it does not exceed 3 m.; in some mines it varies from 2 to 20 and even 30 m.

The largest number of mineral-bearing beds is at Solfara Grande di Sommatino, where there are seven, which have a total thickness of 40 m. Ordinarily there will be found two or three beds, each from 2 to 3 m. thick.

Practically, the tenor of the mineral varies from 8 to 40% S; below 8% it is now considered worthless; more than 40% is exceptional, though in certain mines 90 and even 95% has been produced from the "Sterro" (fines). Mineral containing from 8 to 16% S is considered as poor; that with 16 to 24% as

FIG. 2.—SOLFARA CUMMINELLI (CALTANISSETTA).

1, Tortonian marls; 2, tripoli; 3, limestone; 4, ore; 5, gypsum; 6, marls; 7, tripoli.

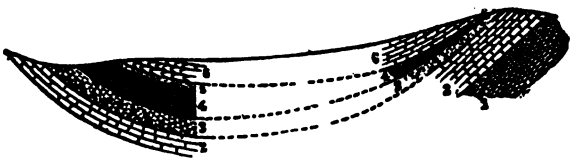


FIG. 3.—SOLFARA GROTTACALD (VALGUARNER).

1, Tortonian marls; 2, silicious limestone; 3, gypsum; 4, ore; 5, tufa; 6, "trubi."

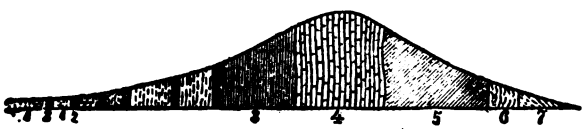


FIG. 4.—SOLFARA OF FIUME RIESI.

1, Tortonian marls; 2, sandstone; 3, tripoli; 4, silicious limestone; 5, ore; 6, gypsum; 7, marls.

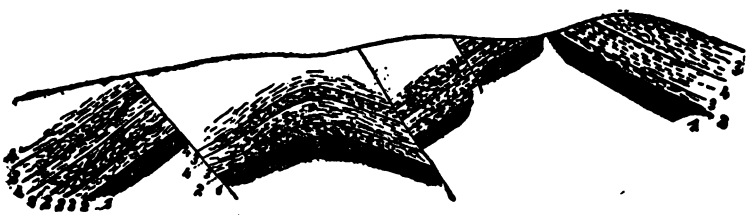


FIG. 5.—SOLFARA GASPA (VILLAROSA).

1, Tripoli; 2, ore; 3, gypsum; 4, gypseous marls.

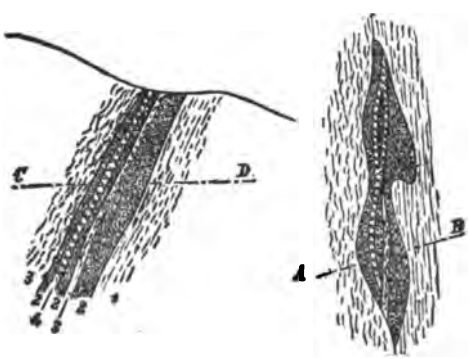


FIG. 6.—SOLFARA GIOVANNELLO (CASTELLARMINI).

1, Tortonian marls; 2, ore; 3, gypsum; 4, limestone.

good; while that with 24 to 40% is regarded as rich. The general average of all the deposits has been estimated at from 22 to 24%.

Generally the thickness and richness vary in each deposit according to the dip and often according to the strike. This great variability with insufficient exploration does not permit any reliable estimates as to the quantity of sulphur contained in the Sicilian deposits. However according to estimates by Messrs. Mottura and Conti, in 1885, there were at least 50,000,000 tons in sight; and according to M. Baldacci, there should have been at that time at least 42,000,000 tons. Since then it is estimated that about 8,000,000 tons have been taken out, so that there would still remain to-day 42,000,000 tons according to the former estimate, or 34,000,000 according to the latter.

The Sulphur Deposits of the Italian Peninsula.—The sedimentary deposits which are being worked on the Italian Peninsula, are of much inferior importance to those of Sicily with which they are identical in age and essential characteristics. The most important among them are in Romagna and Marche in Central Italy. The "Gessoso-solfifera" formation extends east of the Apennines, from Forli to Macerata. The formation is nearly always conformable with the Sarmatian tripolis, while the gypsum is accompanied by clays more or less bituminous, and by a silicious marly limestone similar to that in Sicily. The mineral bed varies in thickness from 1 to 4 m. and in some exceptional instances reaches 15 m. The average sulphur content of the mineral is from 18 to 20%, its yield with the "calcarone" being 12%.

Turning south, we find mines in the Province of Avellino, in the Sabbato Valley, where there is only one deposit, of an average thickness of 8 m., the mineral averaging 33% S. In Calabria, there are a few mines in the region of Cottone. These deposits are identical with those in Sicily, but their exploitation has not yet assumed much importance. The gangue of the mineral often consists of hard marl. The poorest ore worked yields 8% S with the "carcarone," and the richest yields as high as 25%.

Exploitation of the Mines.—The oldest historical document mentioning sulphur mines in Sicily dates back to the twelfth century, although it is fair to presume that the deposits were worked a long time previous to that date. But it was only during the first half of the nineteenth century that Sicily began to occupy a prominent place among the sulphur-producing countries. In 1832 a heavy demand from abroad resulted in the exportation of 31,818 tons; in the succeeding year the quantity exported reached 39,349 tons, and two years later 49,083 tons. Since then the exportation has increased rapidly. The statistics from 1860 to 1899 pertaining to this industry are set forth in the accompanying table.

The working of the deposits in Romagna and Marche began in the first half of the nineteenth century, but it was not until the second half that it amounted to much. The maximum production was reached in 1878 with 39,017 tons of sulphur, since which time it has gradually fallen off.

If anything has to be said in a general way in regard to the working of the sulphur mines in Sicily, it must be admitted that it has always been done in a deplorable manner, technically and industrially, and it has been only at the cost

of great effort, and remarkable perseverance on the part of the "Real Corpo delle Miniere," that some improvements have been introduced especially in order to increase the safety of mining, but which, being merely an application of rational methods of working, could not fail to be beneficial from an economical standpoint.

SULPHUR INDUSTRY FROM 1860 TO 1899.

	Sicily.				Italy.				
Year.	Number of Mines in Operation.	Production. Metric Tons.	Total Value Lire.	Shipped to the Italian Peninsula and to Foreign Countries.	Number of Mines in Operation.	Production. Metric Tons.	Total Value Lire.	Exports. Metric Tons.	Average Value per M. Ton at Seaport. Lire.
1860.....	300	150,000	18,000,000	143,783	309	157,599	19,134,000	.....	120·00
1861.....	320	160,000	19,200,000	156,645	329	165,383	20,080,330	.....	120·00
1862.....	320	160,000	19,200,000	155,330	331	165,485	20,064,125	143,324	120·00
1863.....	350	175,000	21,000,000	169,956	359	182,571	22,145,088	147,035	120·00
1864.....	365	172,618	20,365,353	155,678	376	180,610	21,424,198	139,841	117·40
1865.....	365	162,905	18,891,974	159,657	377	171,587	19,463,767	138,282	112·90
1866.....	297	190,123	22,912,043	184,173	309	198,304	23,728,324	179,110	120·51
1867.....	297	189,733	23,137,065	187,881	309	199,172	24,046,370	192,320	121·94
1868.....	270	189,344	23,362,087	171,901	284	201,333	24,985,273	175,387	123·38
1869.....	322	180,086	23,498,768	170,902	338	200,719	26,278,315	170,142	130·48
1870.....	322	180,199	21,778,161	165,562	338	203,974	24,366,408	172,752	120·86
1871.....	259	176,504	22,693,130	158,976	274	199,728	25,230,050	171,236	128·57
1872.....	297	218,863	27,705,358	185,232	315	239,167	29,773,324	182,185	126·64
1873.....	307	246,815	31,198,680	243,595	324	274,401	34,030,986	202,376	126·40
1874.....	254	214,442	30,486,560	185,731	271	251,259	35,804,078	173,360	142·17
1875.....	204	173,423	24,459,370	222,449	219	207,420	28,821,032	215,144	141·64
1876.....	256	239,221	28,706,544	208,234	269	276,041	33,017,540	194,786	120·00
1877.....	272	217,947	21,903,354	214,890	289	260,325	26,861,724	208,622	100·50
1878.....	292	255,025	21,298,848	235,706	312	305,142	30,552,671	218,326	99·20
1879.....	275	329,984	32,142,607	300,026	299	376,316	36,477,537	242,271	97·41
1880.....	250	312,921	31,402,533	285,103	280	359,663	36,465,593	287,149	100·35
1881.....	299	323,151	37,250,889	297,778	320	373,160	41,907,966	289,365	115·30
1882.....	329	394,087	41,379,135	316,322	352	445,918	46,642,589	273,347	105·00
1883.....	344	391,689	37,210,465	357,974	367	446,506	42,393,199	288,381	95·00
1884.....	369	367,712	32,726,368	338,214	393	411,037	36,522,099	277,210	89·00
1885.....	346	377,194	31,307,102	333,519	374	425,547	34,964,129	289,257	83·00
1886.....	373	326,657	24,891,263	347,336	403	374,343	27,962,282	300,081	76·30
1887.....	336	300,757	20,905,739	329,123	364	342,215	23,694,194	279,628	69·50
1888.....	337	322,042	21,512,405	308,075	362	376,538	25,013,014	323,790	66·80
1889.....	394	327,672	21,744,198	371,652	419	371,494	24,651,876	331,902	65·36
1890.....	480	328,024	25,461,037	364,363	504	369,239	28,265,291	328,708	77·62
1891.....	581	347,568	40,175,385	306,158	611	395,528	44,525,456	269,376	112·57
1892.....	657	374,359	35,626,249	323,932	689	418,535	39,221,065	291,081	95·17
1893.....	569	374,840	27,119,674	367,774	604	417,671	29,616,675	310,867	72·35
1894.....	487	366,185	23,285,704	348,406	524	405,781	25,267,955	299,090	63·59
1895.....	432	352,908	19,653,446	364,417	456	381,197	21,027,176	317,566	55·69
1896.....	490	405,628	28,240,871	406,630	509	440,581	30,163,891	356,370	69·62
1897.....	642	495,201	42,953,418	425,369	668	500,442	45,418,797	358,932	90·56
1898.....	695	482,198	46,605,392	458,916	720	506,805	48,922,442	405,823	96·62
1899.....	....	.....	.....	485,946	.....	.....	.....	.....	.....

From the technical as well as from the industrial standpoint the chief difficulty is the fact that in Sicily the sulphur mines belong to the owner of the soil surface, and consequently the field of exploitation is determined by the limits of the landed properties and the mines must be arranged accordingly. There are some large properties of sufficient area, but in general they are too small for mining requirements and moreover the lines do not in any way, except by chance, conform to the lay of mineral deposits. If the number of working mines be compared with the total production, it can be seen readily of how little importance most of the mines are. The following table will show the conditions prevailing in 1890:

Production per Mine.	No. of Mines.	Total Production.
0— 100 tons.	152	7,444
100— 500 "	195	50,237
500— 1,000 "	56	40,572
1,000— 2,000 "	37	51,266
2,000— 3,000 "	16	37,681
3,000— 4,000 "	10	34,311
4,000— 5,000 "	4	17,869
5,000— 6,000 "	2	11,114
6,000— 7,000 "	....	.....
7,000— 8,000 "	3	22,779
8,000— 9,000 "	1	8,790
9,000-10,000 "	1	9,765
10,000 and above.	3	86,196
Total.....	480	328,024

Owners of sulphur mines in Sicily either work them themselves, or lease them on royalty to a "gabellotto." The latter method, however, is more common and about three-fourths of the mines are now leased. According to the terms of the lease, the "gabellotto" undertakes, at his own risk, the working of the mine for a term of years, and binds himself to reserve a part of the sulphur taken out for the owner. The lease is usually for a period of 9 to 12 years, but may, in rare cases, run for 20 years.

The quantity reserved for the owner ranges all the way from 3 to 10%, and at times even as high as 40% of the gross production. The lease also specifies the method to be employed in the working of the mine, with a view of having the latter revert to its owner in good condition at the expiration of the lease. Under such conditions it is hardly to be expected that the mine will be operated according to the principles of intelligent mining. Even the organization of labor, in which piece-work plays an important part, is changing conditions from bad to worse, rendering it particularly difficult to establish systematic management. Hence, the working, which spoils the ore, at the same time endangers the mine.

It is no wonder that we can only confirm, more or less what an Italian engineer, Sig. Fabri, said some 25 years ago: "Properly speaking we cannot give any name to the method or to the general disposition of work in Sicily, because in most cases the principles of scientific mining are not practiced."

Exploring is done by means of inclined galleries ("buchi," "scale," "discenderie"), which always have a steep pitch, sometimes as steep as 40 or 45°. These galleries start from the outcropping of the "briscale," and are excavated along the latter, or if the pitch is too steep, along the hanging or foot wall. Besides the "briscale," which is nothing more than the outcrop, there are no surface indications to aid in exploration. A knowledge of the geological conditions does not afford much information as to the presence or absence of sulphur; and, indeed, the knowledge that we have is very scanty and inadequate.

After the prospecting galleries have been driven, exploitation begins and is carried on without any attention to the preparatory work. Mining is done almost exclusively with pillar and stall workings, and this system is usually stipulated for in the terms of the lease, which in many cases also fix the dimensions of the pillars and galleries.

Excavation is carried on along the dip and strike of the bed, but reaches its greatest development in the richer and easily mined parts of the deposit.

The galleries are from 2 to 2·5 m. wide, some are even 4 m. and more, according to the nature of the deposit. Their height varies in conformity with the thickness of the bed and the character of the wall rocks. If the latter are sound, the entire bed may be taken out, provided it is not more than 4 or 5 m. thick, but if the walls are weak, it is seldom that more than 2 m. of the bed is excavated, leaving a portion of it on the roof and floor as a protection ("archi" and "pasture"). The pillars are of various dimensions even in one and the same bed. Where the dip does not exceed 35°, they are commonly square and average from 3 to 6 m. on the side. If the dip is steeper they are oblong, the longer axis following the dip.

In horizontal or but slightly inclined deposits, after the prospecting galleries have reached them, a main working gallery is driven in both directions, and from the latter at suitable intervals cross drifts are run (downward if there is any dip), which are afterward connected. In this way a checker-work of openings is formed. If the bed is of considerable thickness, mining begins on the upper part and is afterward carried down so long as the height of the excavation does not become excessive; that is, not over 4 to 5 m. If the thickness is such that it cannot be at once mined to its entire height, all mineral in excess of that excavated is left in the roof or at the floor and later is worked in successive floors, from the upper part downward. In such cases the greatest irregularity prevails in the shape of the workings, as usually no attention is paid, while working out one floor, to the position and size of the pillars previously left on an upper floor. When a barren parting is met, which separates the ore into two portions, the two parts are mined successively only when the barren is very thick.

Beds steeper than 45° are mined in horizontal sections. If the bed is not too large in proportion to the stability of the roof and the side, drifts can be cut through its entire length, the rock in the roof and sides serving as casing. The different levels are separated by a layer of ore from 2·5 to 3 m. in thickness and often only 2 m. The depth of the excavation is very variable, sometimes reaching 10 m. Openings are left in the tops and bottoms of the galleries for communication between the different levels. If in a very steep bed, the thickness is too great to allow of working it by a single gallery, it is divided into horizontal sections, one above another, and a gallery is run in each of these.

Under such conditions, when the surveillance of the Government engineers was less efficient than at present, mining was in a disordered state and was, indeed, a dangerous occupation.

Each level was worked as if there were no others above or below. In one mine (Cozzo Disi), the ore bed, which on a length of 60 m. was 30 m. thick, was worked by 32 galleries in vertical succession.

The superposition of so many galleries is now prohibited, and further changes have taken place in the past few years, which add to the safety of mining. The practice of stowing is now coming into use, although it was entirely unknown a short time ago. There are, too, a few managers who possess sufficient

foresight to recognize the value of modern principles in mining, as well as the courage to put them in practice. Innovations, however, are always met by protests on the part of the mine owners whose vision never extends beyond the present.

The ore is usually loosened and broken up by blasting powder, but picks are used where the deposit is very rich.

When a mine is first opened the walls are generally firm and require no artificial support; where weaker rocks are met they are consolidated by plastering or lining with masonry. Timber is never used for this purpose, if it can be dispensed with, on account of its high cost. In the course of time, the walls become weak and repairs must be kept up continuously at an ever increasing expense.

In regard to conveying and raising the ore, it is gratifying to find that real progress has been made. Previous to 20 years ago, this was done almost entirely on the backs of laborers; but over 30% of the ore is now raised by mechanical means. In 1898 there were operated 27 pits and 23 inclines with mechanical appliances, by which more than 34% of the total production was raised. In some mines where the ore is hoisted by machinery, the conveying in the interior is still done by hand even under extremely difficult conditions. Thus, the ore may be raised by this means from one level to another often an height of 50 m. In mines where there is no hoisting machinery, the ore carriers ("carusi") may have to climb a vertical distance of 200 m. with their load; little attention is paid in such cases to choosing the most direct or safest passage; in fact, up to a few years ago and until the Government interfered, the carriers often made use of the ventilating shafts for passing in and out of the mines. The usual load for an adult is 60 kg., for a boy 20 to 30 kg. The number of boys under 15 years of age employed in this exhaustive labor is now steadily decreasing.

Although a few of the mines are worked down to 200 m. or more, the average depth is not over 100 m. as is shown in the statistics of the year 1890, when out of a total sulphur output of 328,024 tons, 243,167 tons were taken from mines whose lowest levels were 105 m. or less.

In general the mines are not very wet, but owing to poor management, water often becomes a serious hindrance to work. The total amount of water pumped out in 1898 was 3,679,410 cu. m., of which only 2,667,370 cu. m. was done by direct action and piston pumps mechanically driven; the remainder by hand pumps (211 in number) and by buckets attached to the ore-hoisting apparatus. Hand pumps are still used in the small mines and in the deeper levels of the larger ones to which steam facilities have not yet been extended. The amount of water pumped out by hand in 1896 was 1,039,069 cu. m.

No effort is made to secure ventilation beyond the construction of at least two air shafts. In such a network of galleries, there can hardly be a sufficient circulation of air without artificial aid, and consequently the temperature is often very high.

The air becomes vitiated by breathing and combustion, and is also charged with the carbon dioxide that emanates from the bituminous marls accompanying the ore and with hydrogen disulphide as well. Fortunately fire damp is

rarely encountered, but sulphur dioxide, which is often developed, may, in case of fire, be the cause of great disaster. According to law, whenever the presence of explosive gas is ascertained, a forced circulation of air should be maintained. Safety lamps are but little used, except under particularly dangerous conditions, though they are employed by the inspectors when entering the mines.

Second and Third Phases of Exploitation.—After opening the galleries as above outlined, which constitutes the first phase of exploitation, the pillars are taken down and the ore removed. The remainder of the deposit, after the walls have settled, may be mined by opening a new system of galleries, a very dangerous and very difficult operation. The amount of ore remaining in the mine after the first exploitation, may be estimated at an average of more than one-half of the total deposit, there being about that amount in the pillars and walls. In general practice from 20 to 25% of the deposit is never removed.

Exploitation of Sulphur Mines in Romagna.—Greater difficulties are encountered in working the mines of Romagna than those of Sicily, but it is accomplished with a degree of skill not found in the latter. Their depth is considerably greater; in some groups it averages 270 m., while the lowest levels in certain mines are nearly 350 m. below the surface. It is not necessary to enter into the details of exploitation; sufficient to state that various systems are employed according to the nature of the deposit, such as open cutting, pillar and stall and drifting. Stowing is extensively practiced, the material when not already present being brought from the outside. The walls are supported by timber, and masonry is sometimes used where special care is needed. The conditions of the mines as regards water are variable but the means of drainage are always adequate. In some mines from 200 to 300 cu. m. of water are pumped out daily and even double that amount at certain times. Ventilation is secured by inclined galleries and shafts of which there are never less than three. The ore is hoisted by steam while the miners enter by inclined galleries and in some mines by shafts, using the hoisting apparatus or ladders for going up and down. Ventilation is assisted by fans sometimes operated by hand and sometimes by engines. Each mine is provided with safety lamps of the well-known protector pattern.

Fires and Explosions in Mines.—Fires are frequently started by the overheated particles that fly from the hard rock when struck by steel tools, but the more common cause, especially in Sicily, is the heat generated by the roofs falling in. When once started they may be rapidly spread by the sulphurous dust which often causes very dangerous explosions. At the beginning of a fire, buckets of water and pumps are brought into play; if these means do not suffice, attempts are made to stifle it. This is relatively an easy matter in the mines of Romagna as the fire can be readily isolated, but in Sicily, where the deposit of sulphur is usually cut through and through by galleries, it is often necessary to abandon the whole mine and to close all air passages. Generally the fire will burn out in a month or less. Some mines burn for much longer time, one in particular at Sommatino, the "Solfara Grande" has been on fire for over 17 years. The conditions are sometimes such that an immense "calarone" results, which fuses the sulphur in the gangue and deposits it in

accessible portions of the mine. In 1898 sulphur to the amount of 3,954 tons was taken from this source, amounting to 0·82% of the total production of sulphur.

It is of course impossible to enter a burning mine without some protection against breathing the sulphur dioxide gas. The simplest protector is a large sponge moistened with water and held close to the mouth; but this is serviceable only for a very limited time. For a longer period the Galibert sack is used in Sicily, while on the Continent an apparatus invented by Pirazzoli in 1882 consisting of a box with a moistened sponge and two valves through which the air enters and escapes, is extensively employed. With the aid of the latter a person can remain in the vitiated air of a burning mine for 20 to 40 minutes.

Extraction of Sulphur from the Ore.—Owing to the slight difference in the specific gravity of sulphur and its gangue, the usual mechanical contrivances for effecting a separation cannot be employed, and even hand picking is inef-

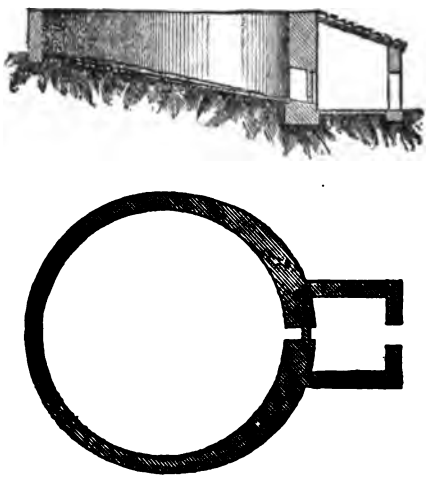


FIG. 7.—CALCARONE—VERTICAL AND HORIZONTAL SECTIONS.

fectual in most cases. The methods that are available are liquation, distillation and solution.

Liquation, the only method now in use, is capable of several modifications according to the means employed to produce it:

(1) Heat generated from using a part of the sulphur as fuel; (2) heat generated from other fuels; (3) hot steam currents; (4) saline solutions which boil at a higher temperature than the melting point of sulphur.

1. *Calcarelle-Calcarone.*—The primitive method for extracting sulphur from the ore in Sicily was by means of the calcarelle, but this has been replaced by that of the calcarone which differs principally in the larger dimensions that are given to the kiln. The calcarone (see Fig. 7) consists of a circular or elliptical wall of ordinary masonry, sometimes let down for a distance in the ground, with a bottom inclining 10 to 15° toward the front. The inner surface of the wall is covered with a thin coat of plaster. The thickness of the wall varies

in different parts, being greater near the front; for a calcarone of 10 m. diameter and 2·5 m. depth the walls should be 40 or 50 cm. thick at the rear and 1 to 1·2 m. at the front. A floor of masonry is sometimes made but often the ore is laid directly on the ground, care being taken to have the lower layer finely pulverized and firmly packed. On the level with the floor and at the front is an opening 1·2 to 1·8 m. in length by 0·25 to 0·9 m. in width through which the fused sulphur is tapped off.

In charging the kiln it is usual to place the coarsest ore on the bottom and the finest on the outside next to the walls. Very fine material is mixed with water and shaped into briquettes. The pile of ore when completed extends slightly above the walls and is covered with slag from preceding operations. The opening in front is filled by thin masonry in which small channels are left that are afterward plugged with clay. Fire is started by throwing burning straw or fagots of wood into a small opening or chimney left in the back part of the heap. After fire is started all air passages are closed, and the calcarone is carefully watched until the process of extraction has been completed.

As fast as the fusion progresses the molten sulphur trickles down through the ore and collects on the bottom where it is tapped off by one of the channels left in the walled opening. The first run is made in 8 to 20 days after starting the fire and after that at regular intervals of 6 to 8 hours; if the calcarone is large and the ore rich, the sulphur may run continuously. The wooden moulds in which the sulphur is allowed to harden are in the form of a steep truncated pyramid with a rectangular or parallelopiped base and will hold 60 to 80 kg. of sulphur. In general it takes one to three months after commencing to charge the calcarone before the extraction is completed. For a calcarone of 250 to 350 cu. m. capacity 30 to 35 days are usually necessary, 50 to 60 days for one of 1,000 to 1,200 cu. m., and 80 to 90 days for one of 2,000 to 2,500 cu. m. A further period of 10 to 30 days is allowed for cooling off; a hastening of this operation results in the loss of sulphur through the formation of sulphur dioxide.

It will be readily observed that successful management of the calcarone is not an easy matter. A great source of trouble lies in the difficulty of maintaining a suitable temperature; if it falls too low, a rehardening of the sulphur takes place, while too high a heat means an increased consumption of sulphur and an inferior product. It is only after repeated experiments that the type of calcarone is found that is best suited to the conditions and not until then can satisfactory results be attained.

The two chief and inevitable drawbacks in this process are the loss of a considerable part of the sulphur and the vitiation of the air by sulphur dioxide, which is so destructive to vegetation. The consumption of sulphur may be stated in general as one-third of the total amount in the ore—a quantity larger by half than the theoretical amount necessary for fusion. Incomplete separation from the gangue, volatilization and escape of heat by radiation are the principal sources of loss. Poorer grades of ore which contain less than 10% S do not pay for treatment.

As regards size the calcarone may vary greatly; the smallest have a capacity

of 250 to 300 cu. m., while some as large as 5,000 cu. m. have been built. Those of medium content, say 1,500 cu. m., are to be preferred, being more easily managed than the larger ones and at the same time more economical. A calcarone of 2,000 cu. m. makes but one fusion in a year, one of 900 cu. m. from three to four and one of 100 cu. m. makes as many as ten fusions. The average cost of treating a ton of ore by this method is about 0·70 lire (about 14c.).

Numerous experiments have been made with a view of finding some substitute for the calcarone that would lessen the cost of production; very few of these, however, have survived the test of practice. It can still be stated that the old method holds first place, although it is gradually being replaced by the furnace and steam methods. Of the total amount of sulphur produced in Sicily in 1890, 80% was recovered by the calcarone; in 1891, 71·05%; in 1892, 65%; in 1893, 66·35%; in 1894, 62·53%; in 1895, 55·99%; in 1896, 52·74%.

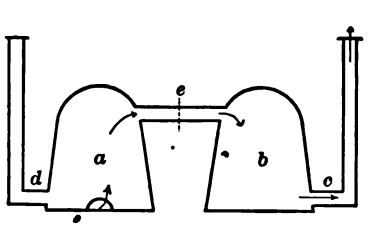


FIG. 8.—OUTLINE SKETCH SHOWING THE MANNER OF OPERATING.

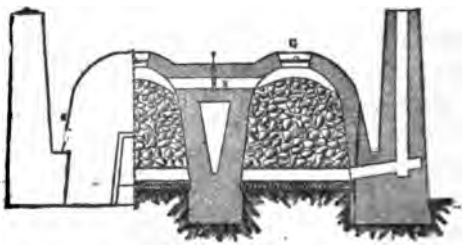


FIG. 9.—VERTICAL SECTION, *a, b, c, d.*

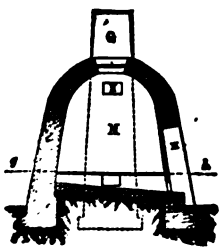


FIG. 10.—VERTICAL SECTION, *e, f.*

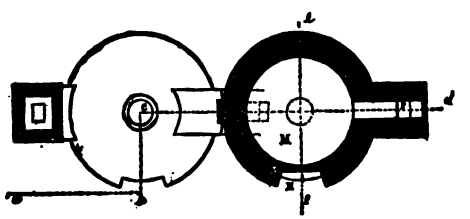


FIG. 11.—PLAN AND HORIZONTAL SECTION, *g, h.*

FIGS. 8-11.—THE GILL FURNACE FOR SULPHUR.

2. *Gill Furnace.*—The Gill furnace which was patented by Robert Gill in 1880 is in its primitive form a modification of the calcarone. Its use has been slowly but steadily growing in the last few years, until now it has become an important factor in the sulphur industry. The amount of sulphur recovered by it was, in 1890, 12% of the total product; in 1891, 16·50%; in 1892, 20·15%; in 1893, 24·20%; in 1894, 25·72%; in 1895, 30·97%; in 1896, 34·09%; in 1897, 38·51%; and in 1898, 40·06%.

The furnace consists of two connecting chambers, one of which when in operation contains the residue of a former fusion, while the other chamber is filled with the ore. Figs. 8, 9, 10 and 11 show the outlines and details of the furnace. The chambers, which are circular in horizontal section with a

diameter that gradually decreases toward the top, communicate at the upper end by a flue which holds a valve or damper, while at the bottom they are connected by another flue with the chimney. Each chamber has an opening that is walled up with brick during operations. Air enters by openings left at the bottom of the chamber containing the residue, where it is heated and passes through the flue into the second chamber, causing a combustion of part of the ore and consequent liquation of the rest; it then escapes with the other gases through the chimney; the chimney adjacent to the first chamber is kept closed. The molten sulphur is tapped off every two or three hours. When the fusion has been completed the valve in the connecting pipe is closed and the residue is drawn from the first chamber, which is then charged with ore. The charging is made through the door and also through an opening in the crown of the furnace. After the chamber has been filled the adjacent chimney is opened, communication is re-established between the two chambers and operations are begun in the manner previously described. The flow of gas may be controlled by a damper in the chimney or by placing a cover over the top.

It has been found advantageous to modify the original type of the Gill furnace by increasing the number of chambers. Where there are only two

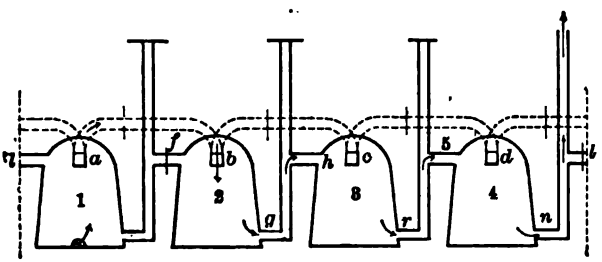


FIG. 12.—THE GILL FURNACE FOR SULPHUR WITH FOUR CHAMBERS.

chambers, the operations have to be suspended while one is being emptied, and they must also be of small size. Indeed in a furnace of two large chambers the sulphur as it melts, runs down and coming in contact with the cooler layers of ore on the bottom becomes chilled and hardens, thus forming a more or less impervious mass which interrupts the flow of air and sulphur. Also, at the close of the operations, the gases are liable to be too highly heated and to carry off both sublimed sulphur and sulphur dioxide. By increasing the number of chambers a larger percentage of the heat is made effective, the size of the chambers may be increased, while the loss by oxidation and sublimation is considerably lessened.

The furnace is seldom constructed with six chambers and usually with not more than four. Fig. 12 shows one of four chambers numbered consecutively 1, 2, 3 and 4 and will aid in explaining the method of operation. Air enters through the bottom of No. 1, which in this case is supposed to contain the residue, becomes heated, and passes into No. 2, which contains ore in process of fusion. After supporting combustion here it passes, in company with the waste gases into chambers Nos. 3 and 4, where it gives up its heat to the fresh charges and then escapes through the chimney. As soon as the fusion in No.

2 has been completed, No. 1 is isolated, emptied of its residue, recharged and then connected by a flue with No. 4. Fusion is now advanced to No. 3, while No. 2 serves as an air warmer.

Although this style of furnace has many advantages and its adoption is becoming more and more general, the results in some cases are not entirely successful. The principal difficulty is in securing sufficient draught to prevent stoppage of the air and gases through the long and devious course in which their temperature is considerably lowered. On this account it has been found sometimes necessary to break the continuity and couple the chambers together in pairs, each pair operating independently.

For the construction of a Gill furnace a few general principles can be given, but such details as relate to arrangement of the chambers and flues must be left to the builder as they depend largely on local conditions. The height of the chambers is usually equal to the diameter of the base or may be slightly in excess of the latter. They are constructed of stone laid in mortar in two courses, a space being left between for a filling of some non-conductor of heat as spent ore. The size of the chambers varies according to the character of the ore, at first it did not usually exceed 5 cu. m., but this limit has been extended after experiments and chambers are now built with a capacity of 10, 20, 30 or even 90 cu. m.; the largest chambers are preferred when the ore is sufficiently compact. The connecting flues are square, measuring 30 or 40 cm. on the side. The chimneys are built to a height of 1·5 to 2 m. above the top of the chambers.

In charging the furnace it is necessary first to spread a coarse layer of ore loosely over the bottom so as to allow an easy passage for the influx of air, but after this the rest of the ore may be dumped in at the top. The amount of time consumed in making a fusion is not entirely dependent upon the size of the chamber—a small chamber cannot be operated very much faster than a large one—but it is influenced by atmospheric conditions and the character of the ore. It is possible to push the fusion by increasing the draught, though if this is carried too far, a loss of sulphur and a poor quality of product will result. Compact or poor ores take longer to fuse than porous or rich ones. In general each chamber in a two-chambered furnace will make from 45 to 90 fusions in a year, a four-chambered furnace from 24 to 60.

The greatest advantage the Gill furnace possesses over the calcarone is the saving in the amount of sulphur consumed, a result of utilizing the spent ore for fuel. In actual practice the output of the former is from 15 to 20% more from the same quantity of ore, an economy sufficient to pay for the first cost of the plant and the increased expense of operation. It also takes the ore just as it comes from the mine while with the calcarone large stock piles are accumulated in which, after a time, considerable capital is tied up to the detriment of the industry, especially in Sicily.

Recent experiments have been made toward effecting further economy by attaching condensation chambers to the furnace, through which the gases are directed before passing out into the air. The results have not been particularly encouraging, it being impossible to save more than 0·5% of the

weight of the ore in flour sulphur in a two-chambered furnace; we do not know how far this may be due to the insufficient dimensions of the condensation chambers.

3. *Fusion by Steam.*—The quantity of sulphur recovered by this process is over 12% of the total production. It has not been so generally introduced as the Gill furnace on account of the greater cost of plant. The use of steam for extracting sulphur from the ore was first proposed by Joseph Gill, and is a logical solution of the many difficulties surrounding the usual methods. By employing steam as a vehicle of heat the temperature can be carried to the desired degree and maintained there for a time sufficient to complete the fusion, a difficulty which has not been satisfactorily solved with the furnace, and a saving in the consumption of fuel can be effected with a fair extraction of sulphur. In regard to the latter point, however, it is necessary to state that after fusion by steam 8 to 15% S is still left in the ore, so that from ores of 25% S or less a relatively smaller amount is recovered than by the calcarone or the Gill furnace. For rich ores, on the other hand, the return is greater by this process; and ores containing much moisture or accompanied by gypsum can be treated with advantage, as in a furnace when such ores are treated a large part of the heat is consumed in driving off the water. Fusion by steam furnishes also a better quality of product than the calcarone, irrespective of the character of the ore. It is more rapid of operation, is independent of weather conditions and emits no sulphur dioxide fumes. These features may often be of sufficient importance to more than balance the larger cost of treatment per ton of ore.

Gill took out the first patent for his steam process in 1859, but it does not appear to have been given a trial. In 1865 Emile Thomas patented a method very similar to that of Gill, although seemingly original with him, and it is from the resultant experiments that the real source of the steam method is to be found. These experiments were carried out in Sicily in 1868, and since then, extraction by steam has been slowly but gradually developing, and in the past few years has attained considerable importance.

The first apparatus was of the vertical type similar to that shown in Fig. 13. This was adopted in many of the mines in Sicily, but has since been abandoned except in the Lercara mines, where the results were particularly favorable. The apparatus consists of a stationary sheet-iron kettle, cylindrical or slightly conical in shape, and standing in a vertical position. The diameter is about 1·2 m., the height 3·2 m., and is supported by four iron pillars. The bottom is fastened by bolts after packing the joint with rubber. On the inside of the cylinder is a truncated cone of perforated sheet-iron with a mean diameter of about 1 m., having the top covered with a lid and the bottom closed by two gratings which can be turned on hinges. An iron receiver of the same shape as the kettle is fastened to the latter by bolts, or rests upon a truck beneath. Steam circulates between the metal cone and the outer plates and also around the receiver when this is jacketed.

For an apparatus of the dimensions given the charge of ore is about 3,500 kg. The steam pressure should register 3·5 to 4 atmospheres after one hour's

work and this pressure must be maintained for three-quarters of an hour. The first influx of steam coming in contact with the ore condenses immediately and falls into the receiver, and as the sulphur melts it also collects in the receiver below the water. When the fusion is complete, the steam is turned off and the sulphur allowed to run out of the receiver into a stone basin, going from there to the moulds. The cover of the boiler is next removed, the receiver unbolted and withdrawn. By unhinging the bottom of the cone, the spent ore is dropped onto a truck and removed.

A fusion takes from two and one-half to three hours; and in practice about seven fusions may be counted on in a run of 24 hours. The dimensions of the kettle were increased at some of the mines so as to give it a capacity of 5 tons of ore. Under favorable conditions about 105 kg. of dry steam are consumed per ton of ore treated. The expense of equipping a plant with three kettles and the necessary steam generating apparatus is about 30,000 fr. On

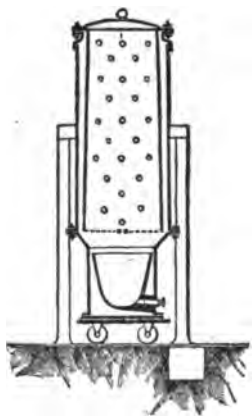


FIG. 13.—VERTICAL STEAM APPARATUS FOR SULPHUR.

the average it costs about 1·8 fr. to treat a ton of ore, considerably more than by the furnace or kiln method.

Another apparatus more commonly used than the above is that invented by Gritti in 1877, shown in Figs. 14, 15 and 16. It consists of a cast or sheet-iron cylinder which may be rotated about an axis so as to assume a horizontal or a vertical position. One end of the cylinder is closed, while the cover for the other end is removable and is fastened by bolts (with the interposition of a lead ring). A diaphragm of perforated metal divides the cylinder into two unequal parts of which the smaller serves during fusion as a receptacle for the molten sulphur. The larger chamber receives the ore which is held in place by another diaphragm of similar construction. When filled the kettle is brought into a nearly horizontal position and the cover is bolted over the open end. Steam is fed through the cover which is also provided with an exhaust pipe. Sometimes, the inlet and outlet of the steam are in the cylindrical portion of the kettle, near the axle of rotation. The sulphur is drawn off by a stop-cock at the bottom. The pressure should register from 3·5 to 4 atmospheres. After fusion the cylinder is again brought into a vertical position, the cover is re-

moved and the residue dumped out. To facilitate the discharge of the residue, the interior of the cylinder is sometimes cased with sheet-iron which, being movable, permits of the fused masses to be easily detached from the surface. The interior diameter of the cylinder is about 1·2 m., the length from 4 to 4·5 m. It will hold 5 to 5·5 tons of ore and can be relied upon to make seven

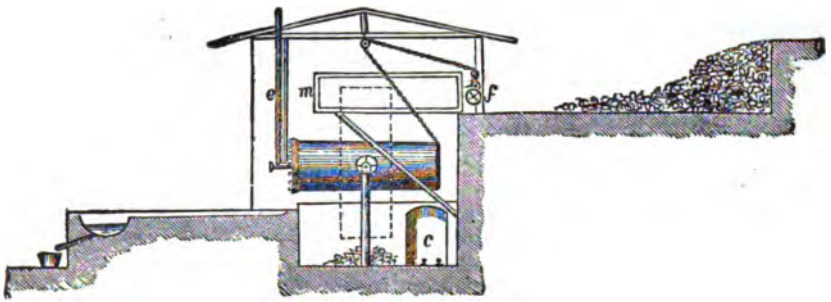
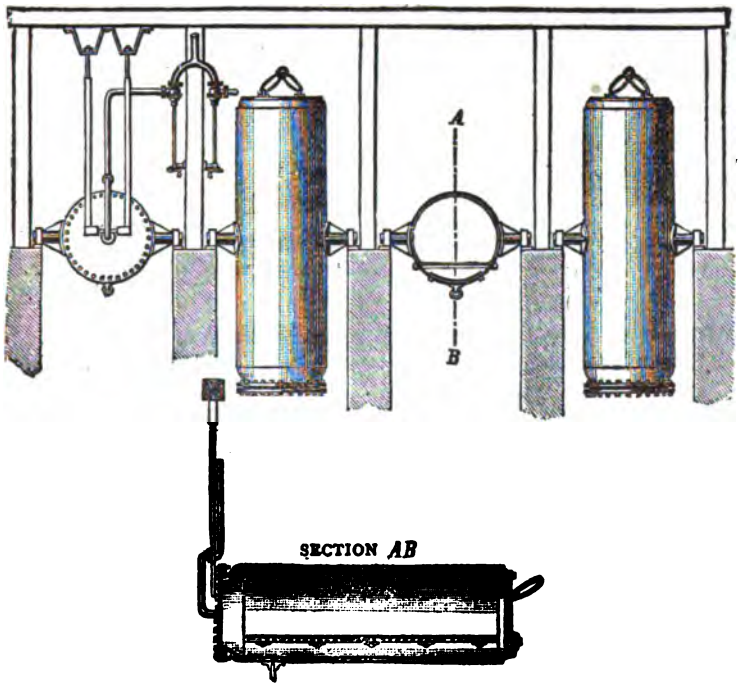


FIG. 14.—SECTION SHOWING THE GRAVITY ARRANGEMENT.



FIGS. 15 AND 16.—TOP VIEW OF ARRANGEMENT AND LONGITUDINAL SECTION OF A CYLINDRICAL KETTLE.

GRITTI'S APPARATUS FOR SULPHUR.

fusions in 24 hours. A plant supplied with four kettles requires a force of 30 workmen (including 16 boys for carrying ore). About 20 cu. m. of water and 2·3 tons of coal will be consumed in each 24 hours. The cost of treating a ton of ore in such a plant is about 1·36 fr. Where only two kettles are in operation the cost will be increased to about 1·60 fr.

Fine or argillaceous ores cannot be advantageously treated by steam, as they are too compact to allow of its circulation. Attempts have been made to avoid this difficulty by dividing the charge of ore into several portions, placing each portion in a separate sheet-iron compartment. It has also been proposed to connect the tap pipe with the bottom of each compartment so that the sulphur may be drawn off directly without passing through the rest of the charge.

The horizontal type of kettle is shown in Figs. 17 and 18. This appears to have some advantages over the rotary form but it has only been adopted in practice very lately and is much less common. It is composed of a cast or sheet-iron cylinder inclined 8 to 10° from the horizontal. The steam feed and exhaust

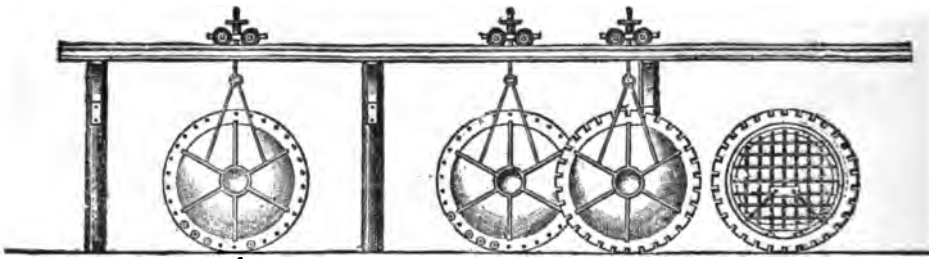


FIG. 17.—VIEW OF END, SHOWING METHOD OF REMOVING HEAD.

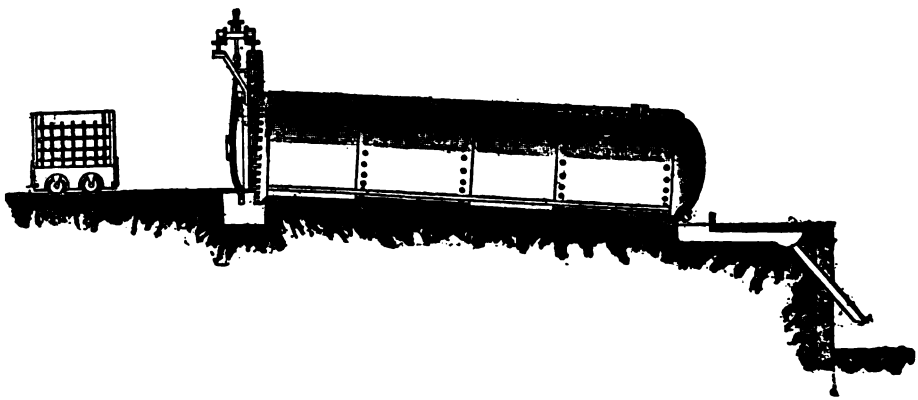


FIG. 18.—LONGITUDINAL SECTION OF CYLINDER.
HORIZONTAL APPARATUS FOR SULPHUR.

pipes are at the back or at the middle portion. The sulphur is run off at the bottom and near the back end. The cover is attached to wheels which run on a track above. Boxes of perforated sheet-iron or woven iron bands supplied with trucks (usually four in number), are used to hold the ore and are run in and out of the kettle on rails.

At the "Solfara Grande" of Sommatino, where this system has been in use since 1891, the interior diameter of the cylinder is 1·18 m., the length 5·15 m. The ore boxes have a capacity of three-fourths of a cu. m. (1,100 kg.). A kettle makes 15 or 16 fusions a day, requiring about 60 kg. of coal for each fusion or 13·6 kg. per ton of ore. The consumption of water is 8·4 cu. m. per day.

The first cost of a plant with one change of trucks, eight in all, exclusive of the steam generator, is about 5,000 fr. The expense of treating a ton of ore on the same basis of calculation as previously used is 1·41 fr.

As compared with the rotary kettle this type has the advantage of being more easily and quickly operated; the loss of heat by radiation is also considerably less. On the other hand its capacity is not used to such good advantage, as some of the space is occupied by the trucks, although the loss is not so considerable as one would suppose.

The same difficulty of treating fine ores mentioned in discussing the rotary kettle is met with here, and has not as yet been satisfactorily settled, notwithstanding a great many experiments have been made. It has been proposed to divide the boxes into compartments and also to sink perforated tubes into the ore so as to permit the better circulation of steam.

4. *Fusion in a Saline Solution.*—In 1865 Thomas proposed to extract sulphur from its gangue by immersing the mineral in a solution of some salt heated to a sufficient temperature. Balard in 1867 advanced the same idea, and Dépérais in 1868 secured a patent in Italy for a process in which the solution was calcium chloride. The price of the salt at that time was too high, however, to permit of its use. Attempts were again made in Sicily in 1882 by the De la Tour du Breuil Brothers and met with some success, but they have never been followed up.

Extraction of Sulphur by Distillation.—This method is now no longer used, although at one time it was employed quite generally on the Continent. In Sicily it never got beyond the experimental stage.

As this paper is only concerned with the sulphur industry of Italy it will not be necessary to include the description of the Dubois furnace which has been employed at Marseilles and has been tried at Livourne.* The method is suited both for the treatment of ore and for the refining of the crude sulphur. It is widely different from any of the methods previously described.

Extraction of Sulphur by Solution.—Attempts have been made to extract sulphur by solution by various inventors, at different epochs and in different countries. It has found some application here and there, but its existence has been more or less ephemeral. The solution used is the bisulphide of carbon. In Italy it has been tried repeatedly; the last attempts were made at the Talacchio mine in 1876 and 1877, a description of which by Niccoli will be found in the *Relazione sul Servizio minerario nel* 1877.

Refining of Sulphur.—In Romagna refining has been carried on for a long time, but in Sicily the first refinery does not date back more than 25 years. This is partly due to the fact that the product of the Sicilian mines is considerably purer and of better quality than that recovered in Romagna. Of late years, however, refineries have been multiplying on the island and now about a tenth of the production is refined sulphur.

The apparatus invented by Zanolini has been adopted most widely in Italy. (See Fig. 19.) Six, eight or ten retorts of elliptic conchoidal shape *s*, are arranged side by side in a long reverberatory furnace. They are con-

* *Rivista del Servizio Minerario*, 1889.

nected by a cast-iron pipe *t*, with a cylinder *C* of the same metal, which acts as a receiver for the sulphur condensed in the pipe. The vault supporting the retorts is cut through at several places to allow the flames to pass up around the feed kettles *f*, in which the sulphur is melted. Each kettle connects with its retort by a short pipe, which may be closed by a valve or plug. The fire-place is at *m*, the hot gases of combustion passing to the space around the kettles by the flues *n*. The residue is withdrawn through a door *p*. The retorts have a diameter of 1·05 m., with a vertical axis of 0·58 m., and will hold 450 to 500 kg. of sulphur. It takes about 12 hours to refine a charge; consequently a furnace with eight retorts will handle 7 to 8 tons of crude sulphur, of which sometimes only 1 to 6% is lost in the refining.

A mixture of wood and lignite is used for heating the furnaces in Romagna. Wood is more suited to the purpose, but owing to its scarcity it has been largely replaced by lignite. In Sicily they burn wood and coal in equal parts. The fuel consumption reduced to terms of coal is in a general way 10% of the weight of the sulphur refined.

The furnace has been modified somewhat in many of the refineries on the Con-

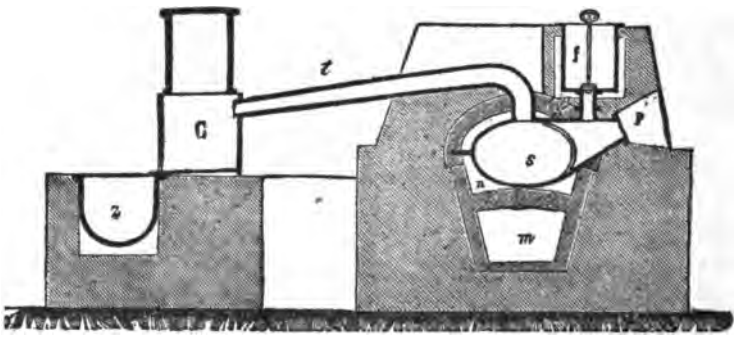


FIG. 19.—ZANOLINI'S APPARATUS FOR REFINING SULPHUR.

tinent. The retorts—more elongated than those described—are placed in separate compartments, thus securing a more uniform distribution of heat and at the same time permitting one or more to lie idle if necessary without disturbing the operation.

For producing flowers of sulphur the same type of retort is used, though of smaller capacity. The settling chambers are rectangular, have an arched roof and measure about 8×9×1·50 m. At Catane, where the largest works in Sicily are located, each furnace treats from 6 to 12 tons of crude sulphur a week, according to the number of retorts and capacity of the chambers. An operation lasts from six to seven days. The mean consumption of coal is 25% by weight of flowers produced. One-tenth of the product is agglomerated or melted sulphur ("colaticci").

Experiments have been carried on in a refinery at Pesaro with a view of accelerating the condensation of the sulphur. Cold air pipes are introduced into the retort chambers. Apparently they have met with good success; the

consumption of fuel is reduced one-fourth, while the service of the retort is largely increased.

Sulphur in Commerce.—Crude sulphur in the warehouses of Sicilian forwarding ports is graded according to its purity and also in a way according to the method employed in its extraction. Both the refined article and the better qualities of the crude are ground and shipped in sacks. A large number of steam mills for grinding are located at the principal export centers. In 1898 a total of 37 mills produced 92,413 tons of ground sulphur, of which 72,313 tons were crude. The mills at Catane accounted for 84,970 tons of the total. The eleven mills in Romagna and Marche (mostly attached to the refineries) in the same year ground 18,658 tons.

Grading is done by simple inspection without any attempt to sample or assay the product. Three qualities are recognized: "prima," "seconda," and "terza." Light-colored sulphurs, "bianchi," come within the first two, and darker varieties, "scuri," within the last. The second and third qualities are each again subdivided into "vantaggiata," "buona" and "corrente." The first quality is in color a beautiful canary yellow and nearly chemically pure; while "seconda vantaggiata" is but slightly inferior; "prima" sulphurs are often selected from the second class. "Seconda buona" has a fine chrome-yellow color, but is less pure than either of the former; "seconda corrente" of dirty yellow color is still less pure and homogeneous; "terza" is chocolate brown on the exterior, shading to greenish brown within, and is of coarse variable structure. In the American export trade there are two special classes, "seconda uso America" (best seconds), which is a mixture of "seconda corrente" and "terza vantaggiata," and "terza uso America" (best thirds), a mixture of "terza vantaggiata" and "terza buona." This system of classification, in which differences of color and structure are made the standards, is the source of many abuses, as the differences of classes do not correspond in a fair proportion to the differences in quality. Some analyses by Vasapolli, of the School of Mines at Caltanissetta, will illustrate this point:

	% S.		% S.
Seconda vantaggiata (calcarone extraction)...	99·82	Terza vantaggiata.....	99·64
Seconda vantaggiata (steam extraction).....	99·85	Terza corrente.....	99·58
Seconda buona (calcarone extraction).....	99·75	Sulphur refined in st cks ("cannoli").....	99·91
Seconda corrente.....	99·70		

It is to be observed that the variances of 2 or 3% spoken of by refiners and sulphuric acid manufacturers do not occur. The difference in aspect, which is often considerable between the several grades, is sometimes as much the result of the temperature and course of the fusion as of the proportion and nature of the impurities. The amount of the latter in refined sulphurs as shown by numerous analyses varies from 0·01 to 0·04%.

The refineries of Romagna prepare in addition to the regular grades the special products known as "solfo lavato" or "fiori neutri," "solfo acido" and "solfo ramato." "Solfo lavato" is obtained by washing the flowers of sulphur in stone vats with water. The washing is kept up until all traces of acidity disappear, and the powder is then dried in chambers warmed by the waste gases of sublimation furnaces. The product is used in pharmacy, also for the manu-

facture of powders and for vulcanizing rubber. "Solfo acido" (used in grape culture) has an acidity of about 2·2 per 1,000; it is prepared by mixing flowers of varying acidity (1·5 to 3 per 1,000) until the right proportion is reached. "Solfo ramato" is a mixture of sulphur with 2 to 5% of copper sulphate.

Condition of the Sulphur Industry.—In spite of the importance of the sulphur industry of Sicily and the fact that it has practically a monopoly of this article so necessary to modern civilization, it has passed through many severe crises, and no one would venture to say at the present time that it may not encounter many more. From 1860, when Sicily produced 150,000 tons, the output increased by gradual steps to 218,863 tons in 1872. In the following year it jumped to 246,815 tons, and in 1875 as suddenly fell off to 173,423 tons. It reached the total of 329,984 tons in 1879, and since that time has remained above the 300,000 figure. The output of 394,087 tons in the year 1882 remained the largest recorded until 1896, when 405,628 tons were produced, and this mark was again exceeded by the production of 1897, which was 475,000 tons, by that of 1898, 482,198 tons, and by that of 1899 about 520,000 tons. In 1900 it is expected that the production will reach 550,000 tons. The average prices have likewise experienced great fluctuations with a tendency toward steady decline. Up to the year 1877 the annual average price per ton at forwarding points was never less than 100 fr. and reached 142·17 fr. in 1874. In 1878 it was 99·20 fr. and in 1879 97·41 fr.; it again passed 100 fr. in the three subsequent years, and again fell to still lower figures. After reaching 112·57 fr. in 1891 the price fell to 95·17 fr. in 1892, 72·35 fr. in 1893, 63·59 fr. in 1894 and to the extremely low mark of 55·69 fr. in 1895.

The first crisis extending from 1886 to 1890, and that beginning in the year 1893, attracted the notice of the Government to the necessity of remedial measures. Numerous plans were proposed, many of which were contradictory and inapplicable and arose from a misunderstanding of the true conditions. For explaining such rapid and extensive fluctuations of prices the first causes which suggest themselves are naturally overproduction and foreign competition. There is no doubt that the first of these causes does exist, but even a hasty examination of the question shows that it is entirely out of proportion to the effects produced. The fact is that the lowering of prices has continued steadily at a time when the excess of production over consumption has decreased. Up to the year 1882 the excess of production over exports, which had always been less than 30,000 tons, increased to 77,000 tons and the price fell off by 15·30 fr. In the following year the excess continued to increase while the price dropped 22 fr. From 1886 to 1889, on the other hand, the exportation was over 139,000 tons more than the output, yet the price continued to drop and was lessened in all about 18 fr., a result that cannot be accounted for by the assumption that a large amount of stocks was thrown on the market. Competition has also been used as an explanation, but its importance is exaggerated, Italy is still the great source of sulphur and has no serious competitor in the markets of the world. Whether the product recovered by the Chance process in soda manufacture will prove a rival in the future cannot be foretold, but at present there is no danger from this source. Pyrites is much more to be

feared, as it is only a matter of time when the United States, which is the greatest customer of the Italian mines, will, like Europe, use the sulphides entirely for sulphuric acid manufacture. This means a large decrease in the exports in the near future unless compensated for by new applications of sulphur in the industries. This, however, is a future danger and has had little to do with the crisis. The kernel of the whole matter is this; the conditions of the sulphur industry in Sicily are such that any slump in the trade due to overproduction is greatly exaggerated. The market is feverish, uncertain and ever responsive to pressure from without. Lack of capital among the producers renders them victims of speculators who, in turn, to save themselves are often compelled to lower prices.

Looking at the question from every standpoint, whether one sides with those who propose to lessen the price of sulphur in order to maintain or increase the exportations, or with those who believe that the price should be increased even at the expense of a smaller output, it is apparent that the home trade in sulphur must be restricted by regulations. Attempts to bring the producers voluntarily into a trust having failed, the Government was asked to compel such a union, but it was unwilling to enter upon a plan so antagonistic to the principles of liberty in order to achieve a doubtful result. It has limited itself to granting financial aid, to furnishing facilities for the erection of general warehouses and to the abolition of export duties. Following this, some of the leading houses in Sicily united with English capitalists to form the Anglo-Sicilian Sulphur Co., Ltd., whose purpose was to gain control of as much of the Sicilian product as possible in order to market it under the best conditions and to maintain the prices at a reasonable point. The company secured by contracts, dating five years from August 1, 1896, and renewable under certain conditions for an equal period, the entire production of each producer at a fixed price, which was considerably above the average for the previous years. The output of each mine for the first year was not to be exceeded thereafter; the company also assumed the right to still further limit the production under agreement to keep the surplus in storage to the credit of the producer. A number of the English alkali manufacturers, represented by Mr. Chance, agreed to limit the output of sulphur by the soda process to 40,000 tons per annum. The prices fixed by the contract are: for "seconda vantaggiata," 82·65 fr.; for "terza vantaggiata," 80·45 fr.; and for "terza buona" and "correnta" 78·25 fr., as compared with an average of 60 fr. in 1895.

Although the advantages to the producer are evident, the company has not been able to secure control of four-fifths of the Sicilian production, which was desired, and consequently it has had to meet a strong competition from the outside producers. The final outcome will be anxiously awaited. It is possible that at the end of the five years the production may have been so stimulated by the favorable prices as to produce a crisis more disastrous than any that have gone before. The increased export demands, the abolition of export duties and the formation of the Anglo-Sicilian Co. have led to increased activity in mining. New deposits have been opened up and abandoned mines are again being worked.

From 1896 to 1899 the production of Sicily has increased over 119,000 tons, and the exportation about 80,000 tons. The prices, as already stated, have been in the last three years very satisfactory when compared with those of the years 1893-96; but in the same time, the stocks are also increased over 80,000 tons, and are now estimated (end of December, 1899) at 799,000 tons.

The betterment of the market has had a favorable effect on the mines of Romagna and Marche, although not so much as it would be expected. Many of the mines were closed and the others cannot easily increase their production. Progress was limited chiefly to the mines of Marche and more particularly to that of Cabernardi, which belongs to the "Schewefelgewerkschaft Italia." This company, although organized but a few years ago, has attained considerable importance under the favoring influences of a progressive management and a rich and extensive deposit. It produces about 7,000 tons of sulphur per annum, which it is able to refine in its own works. This and the S. Lorenze Co., of Zolfinelli, which has had a long career, control the only mines in Marche that are worked at present. The mines in Romagna, formerly the most important on the Continent, have lately fallen into great disorder; however, if the present conditions of the market continue they may yet have a good future. Marche may also become an important producer, as it contains many large deposits which as yet have not been explored. Hitherto the production of sulphur in these two districts has been under 20,000 tons per annum, but it is expected to increase in the future even to double that amount.

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TALC AND SOAPSTONE.

THE production of fibrous talc in 1899 increased above the output in the previous year, although that was in excess of past years, owing to the increased consumption of paper in the many editions of daily newspapers published during the Spanish-American war. Producers, however, were not fortunate in maintaining prices, as on the average these declined from $5·21 per ton at the mine to $4·77. Production, as in previous years, was confined almost entirely to St. Lawrence County, N. Y. The attempted combination of North Carolina producers resulted in little increased activity if any.

Soapstone.—Soapstone was produced to a greater extent than in previous years, but unlike fibrous talc the price increased from $8·41 to $9·41 per short

STATISTICS OF FIBROUS TALC AND SOAPSTONE IN THE UNITED STATES.
(In tons of 2,000 lb.)

Year.	Production.									Imports. (*b*)		
	Fibrous Talc.			Common Talc.			Soapstone.			Tons.	Value.	Per Ton
	Tons.	Value.	Per Ton	Tons.	Value.	Per Ton	Tons.	Value. (*a*)	Per Ton			
1895....	40,000	$320,000	$8·00	9,068	$86,099	$9 50	13.070	$130,700	$10·00	3,165	$26,843	$8·48
1896....	51,816	256,080	4·94	7,098	63,585	8·97	14,350	143,500	10·00	1,950	18,693	9·60
1897....	52,836	283,686	5·37	10,164	90,908	8·94	16,904	169,040	10·00	799	8,423	10·54
1898	54,807	285,759	5·21	9,112	78,645	8·63	18,862	158,635	8·41	445	5,536	10·70
1899....	57,120	272,595	4·77	6,671	51,763	7·71	20,011	189,504	9·47	254	3,534	13·91

(*a*) In reporting the value of their output of soapstone producers generally give the figures for the finished articles which they make. Since a varying proportion of labor enters into these, such figures are apt to be misleading. Few producers are able to name a value of the crude stone, or roughly dressed blocks. We have therefore valued the material arbitrarily at this stage at $10 per ton, except in 1898 when a large production of inferior stone reduced the average. The value in 1899 is that reported by the producers. (*b*) Talc, ground, powdered or prepared.

ton at the mine. The Santa Catalina Island soapstone deposit in California has been exhausted practically so that little production was recorded for 1899 from this source. Production in New Hampshire, owing to the exhaustion of the Francestown quarry, fell off considerably, but the Virginia mines still produce their usual amount. North Carolina produces a certain amount of soapstone, which is used as a principal ingredient of complexion powders.

Canada.—The production of soapstone in Canada for 1899 was 450 short tons, valued at $1,060. The production has been uneven and the quantity small.

Foliated talc of fine quality is found near Mantem in Styria, and near Pinerola, southwest of Turin. In the former deposits, talc occurs along the contact of Silurian limestone and graywacke, both of which have undergone dynamical disturbances. The deposits are worked by adits driven at successive levels, the strata here being bent into sharp, anticlinal folds or saddles. The talc, after being taken out, is sorted and graded, the fine white being most valuable, and the yellow, green and gray varieties less so. The thickness of the seam does not exceed 3 m. and averages much less.*

* *Zeitschrift für Praktische Geologie*, February, 1900.

TIN.

No tin was discovered in the United States in 1899, so, as in 1898, there was no production. The Pennsylvania Smelting Co. was organized in Pittsburg by parties who had been interested in the Pennsylvania Lead Co. prior to its absorption by the American Smelting Co., to erect a plant to recover the tin and dross from tin plates and to refine tin and lead.

According to a pamphlet issued by the New England Free Trade League the only mills in the country which are not comprised in the combination of tin-plate manufacturers is the New York Enamelling & Stamping Co., who have four mills at Granite City, near St. Louis; the Alcania Tin Plate Co., Avonmore, Pa., with two mills; the Lalance Grosjean Co., with five or six mills for plates for granite ware, eight mills building at Muskegon, Mich., and two or three mills at Washington, Pa.

The following plants went out of business: The Baltimore Tin Plate Co., Baltimore, Md., two mills; Britton Rolling Mill Co., Cleveland, O., three mills; Cincinnati Rolling Mill & Tin Plate Co., Cincinnati, O., four mills; Cumberland Steel & Tin Plate Co., Cumberland, Md., five mills; Hamilton & Co., West Newton, Pa., two mills; Marshall Bros. & Co., Philadelphia, Pa., six mills; Ohio River Sheet & Tin Plate Co., Rochester, Pa., two mills; Somers' Bros., Brooklyn, N. Y., eight mills; Stickney Iron Co., Baltimore, Md., two mills.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1894.....	39,266,628	$5,944,065	1896.....	44,639,394	$5,848,933	1898.....	62,748,399	$8,770,221
1895.....	54,252,045	7,405,619	1897.....	55,172,571	7,415,983	1899.....	71,948,407	16,746,117

CONSUMPTION OF TIN IN ENGLAND, AMERICA, AND HOLLAND. (IN LONG TONS.) (a)

	1894.	1895.	1896.	1897.	1898.	1899.
Consumption—						
Deliveries from London after deducting all shipments to America	18,050	17,222	19,015	17.118	15,472	13,519
Deliveries from Holland after deducting exports to London and America	8,207	9,029	10,150	11,680	12,130	13,094
English consumed at home	2,314	968	5,000	4,400	4,600	4,400
Exports of English, minus quantity shipped to America	5,686	5,530				
American consumption of all sorts	16,650	22,500	29,500	22,500	28,500	31,500
Billiton sent to other ports than Holland	862	1,532	1,608	1,370	2,120	1,930
Straits direct to Continent, less re-exports to America and England	11,795	7,622	11,400	9,170	12,740	6,470
Bolivian delivered from Liverpool	3,328	4,099	4,071	3,888	3,874	3,690
Bolivian delivered from Continent	.....	.....	210	1,908	1,000	813
Totals	66,817	68,522	71,949	71,284	80,436	75,416

(a) From the annual metal circular of William Sargant & Sons.

STOCKS OF TIN IN ENGLAND, AMERICA AND HOLLAND. (IN LONG TONS.) (*a*)

	1894.	1895.	1896.	1897.	1898.	1899.
Stocks, December 31—						
Stock of foreign in London	8,985	13,539	18,097	15,146	8,110	5,486
Foreign landing in London	1,189	1,025	1,174	678	165	1,212
Straits afloat for London, including wire advices	3,778	2,825	2,792	2,500	1,050	2,900
Australian afloat for London, including wire advices	650	825	525	600	400	450
Banka on warrants in Holland	1,006	1,154	1,616	2,877	2,228	1,160
Billiton in Holland	1,632	1,507	1,688	1,328	1,086	470
Billiton afloat for Holland	1,129	1,097	1,742	1,193	1,323	1,050
Straits stock in Holland	767	377	789	377	454	100
Straits afloat for Holland	.....	.....	950	100	215	.....
Straits afloat for Continent	.....	.....	650	600	560	450
Bolivian in Liverpool	494	492	250	710	300	550
Total stocks	19,630	22,741	30,223	26,104	15,840	13,828
Estimated stock in America and quantity floating	3,820	6,823	3,995	4,500	4,300	2,500
Grand totals	23,450	29,564	34,148	30,604	20,140	16,328
Trading Company's reserves of unsold Banka stock in Holland	5,770	6,140	5,958	4,388	3,213	4,353

(*a*) From the annual metal circular of William Sargant & Sons.

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (IN LONG TONS.) (*a*)

	1893.	1894.	1895.	1896.	1897.	1898.	1899.
English production	8,837	8,328	6,648	4,887	4,453	4,648	4,400
Straits shipments to Europe and America	39,670	45,640	47,840	47,180	41,700	43,350	44,460
Australian shipments to Europe and America	5,579	5,324	4,992	4,320	3,466	2,420	3,337
Banka sales in Holland	5,244	5,626	6,221	6,735	8,900	9,038	9,066
Sales of Singkep in Holland	*Nil.*	261	644	889	800	*Nil.*	*Nil.*
Billiton sales in Java and Holland	5,462	4,735	4,539	5,040	5,100	5,342	5,057
Bolivian arrivals on Continent	.....	.....	.....	210	1,208	1,000	813
Bolivian arrivals in England	2,909	3,482	4,097	3,829	4,298	3,464	3,940
Straits shipments to India and China	4,285	4,655	4,674	6,418	3,214	2,551	1,484
Totals in long tons	71,986	78,551	79,655	79,108	73,139	71,763	72,557
Totals in metric tons	73,138	79,808	80,980	80,374	74,309	72,911	73,718

(*a*) This table is based on the statistics compiled by William Sargant & Co., but the figures of English production are taken from the British blue-book, except for 1899, where Sargant's have been used. The Straits shipments to India and China are as reported by Messrs. Boustead & Co. This table does not include the production of Germany, Austria, Spain, Portugal, and various other countries.

PRODUCTION OF TIN IN THE WORLD. (*n*)

Year.	Australia. (*a*)			Austria. (*b*)			Banka and Billiton. (*c*)	Bolivia. (*d*)	England. (*e*)			Germany. (*f*)		
	Met. Tons.	Value. (*o*)	Per M Ton.	Met. Tons.	Value. (*o*)	Per M Ton.	Metric Tons.	Metric Tons.	Met. Tons.	Value. (*o*)	Per M Ton.	Met. Tons.	Value. (*o*)	Per M Ton.
1894..	2,653	$897,225	$338	80	$38,812	$485	10,527	3,538	8,461	$3,023,500	$357	896	$271,804	$303
1895..	2,295	680,400	304	60	24,257	404	10,932	4,166	6,755	2,333,900	331	884	266,292	301
1896..	1,737	496,060	286	54	18,260	341	11,968	4,104	4,915	1,538,390	313	826	238,017	288
1897..	1,159	350,640	303	48	16,806	350	14,224	5,594	4,524	1,456,680	322	929	287,829	310
1898..	908	302,325	333	48	19,074	397	14,610	4,535	4,722	1,729,060	366	993	372,252	375

Year.	Japan. (*g*)	Mexico. (*h*)	Portugal. (*i*)	Russia. (*j*)			Singkep. (*k*)	Straits Settlements. (*l*)	Tasmania. (*m*)			Total Met. Tons.
	Met. Tons	Met. Tons.	Met. Tons.	Met. Tons.	Value (*o*)	Per M Ton.	Met. Tons.	Met. Tons.	Met. Tons.	Value. (*o*)	Per M Ton.	
1894	39	42	26	4	$1,440	$360	265	51,100	3,195	$1,348,000	$422	80,826
1895	48	39	3	21	5,068	241	654	53,354	4,265	1,301,350	305	83,416
1896	50	5	6	2	476	238	852	53,964	4,759	1,327,369	290	83,227
1897	(*p*)	1	1	(*p*)	(*p*)	(*p*)	813	45,632	3,484	749,970	302	75,409
1898	(*p*)	*Nil.*	*Nil.*	(*p*)	(*p*)	(*p*)	(*p*)	46,635	2,088	706,810	346	74,489

(*a*) From *Report of Secretary of Mines and Agriculture of New South Wales*, which is the only Australian colony that produces metallic tin. Tin ore is also exported from New South Wales. Tin ore is produced in

Victoria, Queensland, South Australia, and Western Australia, but its metal contents are reported in the productions of other countries.

(*b*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.* The tin production of Australia is derived partly from domestic ores, and partly from Bolivian ores and impure tin from the East imported for refining.

(*c*) Total sales in Holland and Java as reported by William Sargant & Co.

(*d*) Exports of tin and tin in ore from Bolivia to England and the Continent. Some of this tin may be produced in Peru. It is all experted through Peruvian and Chilean ports.

(*e*) From *Mineral Statistics of the United Kingdom.* These figures give the amount of tin estimated to be obtainable in smelting the ore of domestic production. They differ considerably from the figures used by William Sargant & Co., which are possibly obtained directly from the smelters.

(*f*) From *Vierteljahrs und Monatshefe zur Statistik des Deutschen Reichs.* By far the larger part of this production is from Bolivian ores.

(*g*) From the *Resume Statisque de l'Empire du Japon.*

(*h*) According to export returns. Small amounts produced by natives for domestic consumption are not included.

(*i*) From official returns furnished by the Portuguese Government. The figures for 1895–1896 include tin ore.

(*j*) From Russian official reports.

(*k*) The figures for 1891–98 are for fiscal years, as reported by O. H. Van der Wyck in *Mineral Resources of the United States* for 1895; the figures for subsequent years represent sales in Holland, reported by De Mouchy & Havelaar and W. Sargant & Co.

(*l*) Shipments from the Straits to Europe and America, as reported by William Sargant & Co., and to India and China by Boustead & Co., except the figure for 1896, which is taken from the *Report of the Governor of the Straits Settlements.*

(*m*) From *Report of the Secretary of Mines.*

(*n*) It will be observed that this table differs from the statistics usually referred to for the world's production of tin, namely, those of William Sargant & Co., and of the *Metallgesellschaft Frankfurt-am-Main.* This is because (1) they reckon Australian shipments to Europe and America, instead of the Australian production, whereby a good deal of Australian tin consumed at home is not included; (2) shipments from the Straits to India and China and the production of several minor countries are not included. A small amount of t n produced in Spain is necessarily omitted, owing to the absence of statistics for that country.

(*o*) In cases where the statistics in the above table have been taken from official sources, wherein values of product have been stated, a calculation of unit values has been made. This developed some amazing differences, which are undoubtedly due to making value calculations at different stages of the product; *i.e.*, more or less cost of carriage is included.

(*p*) Statistics not yet published.

EXPORTS OF TIN PLATES FROM THE UNITED KINGDOM.

(In tons of 2,246 lb.)

	1893.	1894.	1895.	1896.	1897.	1898.	1899.
To the United States.....	255,583	226,880	222,901	113,049	85,472	65,337	63,547
To other countries.....	123,650	127,048	143,061	153,914	185,758	186,432	193,082
Totals.....	379,233	353,928	365,962	266,963	271,230	251,769	256,629

Australia.—On the northern coast of New South Wales, between the Clarence and Richmond Rivers, alluvial deposits occur in sand dunes or terraces outcropping the low-lying, swampy ground, and striking almost parallel with the sea. The dunes are between 200 and 300 ft. wide and 10 to 20 ft. high and run in parallel rows several miles inland. A section of one of the dunes discloses a capping of 10 to 15 ft. of white beach sand covered with spear grass, then several feet of black sand, and about a foot of auriferous and stanniferous sand forming the wash dirt, and lying on a bottom of reddish brown sandstone. The wash is characterized by its dark black color, high specific gravity (4·42) and the fineness of its component particles; the whole frequently passing through a screen with 3,000 holes to the square inch. The wash has a complicated composition, but its principal constituents are titanic iron, zircons, quartz, cassiterite, garnets, gold, platinum and the platinoid metals. The tinstone is generally in a remarkably fine state of division and easily passes through a screen having 8,000 holes per square inch. These concentrates are often highly auriferous. A sample from the Richmond River gave by analysis: Gold, 70 oz. per ton; platinum, 36 oz.; iridium, 8 oz.; iridosmium, 6 oz., with over four ounces of the platinoid metals and 25% of metallic tin.

The Herberton field in Queensland which had produced about $1,000,000 worth of stream tin from 1883 to 1894 and nearly $5,000,000 of lode tin in the

same period, has attracted some attention during the past year, although it had been practically abandoned. The deepest shaft is 600 ft. and only one hoisting plant is on the field. The tin lodes are said to be high grade, nothing under 10% having been worked in the past.

A discovery of lode tin was made at Sebastapol near Beechworth, Victoria. The outcrop was 4 ft. wide and said to be rich.

Banka.—The mines at Banka have been under government control since 1832. Their output in 1897 was 9,307 tons, rising to 10,220 tons in the succeeding year. The mines at Billiton, managed by the Billiton Maatschappj, founded in 1860, produced 5,565 tons of tin in 1897, and 5,778 tons in 1898; 8,223 laborers are employed, and the company has declared handsome dividends notwithstanding heavy taxation.

France.—The tin deposits of Montebras* are on the contact of granite and feldspathic porphyry. A considerable quantity of ore has already been exposed, being augmented constantly by development work. The ore is said to average 4%, which would be above the grade of other European deposits. Operations were suspended during the fall in the price of tin some years ago. Resumption of work is expected and is recommended.

Great Britain.—The Dolcoath Mine, Ltd., earned in the latter half of 1899 £31,066 5s. 5d. as against £22,458 0s. 4d. in the previous half year. The ore averaged 56 lb. of black tin against 56·57 lb. in the previous half year, being much lower than any former return. In September, 35,383 new shares were issued and 2s. 6d. per share paid on allotment. The whole issue up to date including 150,000 vendors' shares, amounts to 323,383 shares; £40,536 7s. 10d. remain to be called up on issued shares; 26,017 shares remain for future issue.

The average value per ton of ore was £1 19s. 8·39d., while the total cost including Lords royalties of 3s. 7d., amounted to £1 5s. 2d. Dividends at the rate of 13·33% were paid during the year 1899. The mine is said to have improved at several points. The statistics of production since the company was formed is set forth in the following table:

Six Months.	Tin Ore Crushed.	Block Tin Sold. T. c. q. lb.	Produce per Ton of Ore.	Average Value per Ton of Ore.	Average Price per Ton of Black Tin.	Amount Realized.
	Tons.		Lb.	£ s. d.	£ s. d.	£ s. d.
December 31, 1895.....	28,717	1,015 5 1 16	79·19	1 7 8·36	39 3 5	39,769 2 5
June 30, 1896.....	30,015	1,029 19 1 10	76·86	1 5 6·33	37 4 0	38,311 6 5
December 31, 1896.....	33,712	1,009 7 0 25	67.06	1 1 10·20	36 9 2	36,330 18 2
June 30, 1897.....	35,879	1,021 17 0 22	63·79	1 1 7·27	37 18 7	38,760 7 2
December 31, 1897.....	37,686	1,073 2 1 9	63·78	1 1 6·78	37 17 4	40,636 19 8
June 30, 1898.....	38,089	1,140 7 2 22	67·07	1 3 9·94	39 16 4	45,221 15 1
December 31, 1898.....	40.606	1,162 0 0 0	64·10	1 6 10·09	46 18 0	54,497 6 3
June 30, 1899.....	41,101	1,037 19 1 23	56·57	1 13 8·21	66 12 7	69,222 12 0
December 31, 1899.....	41,639	1,040 18 1 10	56·00	1 19 8·39	79 7 11	82,651 16 0

Java.—The exports of tin from Java and Madura in long tons are reported as follows:

	1894.	1895.	1896.	1897.	1898.
Private.....	5,768	4,526	4,825	5,436	5,194
Government.....	6,789	6,950	6,707	8,137	8,934

* *La Metallurgie*, March 7, 1900.

Tasmania.—This colony has long been celebrated for its tin deposits, and since 1880 has exported considerably over 79,093 tons of tin, valued at nearly £7,000,000. Most of it was obtained from alluvial workings, but they are rapidly being exhausted. The island, however, possesses very extensive and valuable deposits in stockwerks and similar formations.

The total expenses at the Mt. Bischoff mine during a period of six months ending June 30, 1899, were $1.261 per ton of 2,240 lb. divided as follows: Mining, including new work, maintenance and other expenses, 57c.; hauling, filling and emptying trucks, 12c.; crushing, dressing and maintenance of plant, 22·1c.; slime sheds, 3c.; ring tail sheds, 4c.; management and supervision, 11·5c.; plant, including machinery, 2·2c.; development and prospecting work, 4·5c.; contracts and sundries, 1c.; stores, 8·8c. Product during the period was 950 tons of black tin. The mine is worked in open faces. The first, called the brown face is a semi-circular quarry over 700 ft. in length. The ore, worked down from the benches, is oxide of iron containing nearly 3% of tin oxide. Below this gossan, tin and iron pyrites come in, with some veins of cassiterite. The second or Slaughter face is also gossan but denser. This face is opened up about 50 ft. in length and about 250 ft. in depth. The body here is about 260 ft. wide. The third or white face is alluvial, 1,500 ft. in length and 400 ft. wide. It is opened up to an average of 30 ft. deep and contains an average of 3·75% of tin oxide. Underlying this is a bed of spathic iron boulders, pyrrhotite, blende and clay, which, however, contains little or no tin.

There are 75 stamps on the property. The crushed ore is classified into sands and slimes, the former being treated on 30 double compartment Harz jigs while the latter pass to 39 convex rotating tables 10 to 15 ft. in diameter. Each table takes about 0·13 h. p. to drive it and treat about 900 lb. of slime per hour. The jig tailings are treated on 15 concave buddles 20 ft. in diameter. Of the tin saved the jigs are to be credited with 75%, the slime tables with 15 to 20%, and the concave buddles about 4%. The final tailings which go to waste do not run over 0·5% Sn. The dressed ore is shipped to the seaboard and smelted in reverberatory furnaces, the loss in operation being about 4·5%.

The Briseis mine was offered to the British public early in December. It had been reported favorably upon by H. W. Ferd Kayser of the Mt. Bischoff mine, and R. F. Moore of the South Tharsis Mining Co. Mr. Kayser says in his report: "In a report issued by the directors of the Briseis Co. to the shareholders at the end of 1889, the length of the lead is given as three-quarters of a mile from the Krushka Brothers' boundary line to the head of the lead, the width being put down at eight chains, narrowing however toward the bottom, and therefore only fixing a five-chain average, with a depth of 130 to 60 ft. During my visit, and from the closest observation, I also have formed an opinion on these points, and, while agreeing with the length given, I put down six chains as the average width and 100 ft. for the average height. It appears to me from the surface indications that the lead is widening out in the flat into the shape of a basin, which, if found to be true, would increase the width, but taking the figures as above stated to be correct, a calculation of the dirt, allowing only 1% of ore (which I believe to be under the average on account of the very

rich dirt met with in all the deep ground) would give a result of over £4,000,000 with tin at £100 per ton."

Mr. Kayser's report was fully confirmed by Mr. Moore's.

Water is scarce at present, but it is estimated that an expenditure of $110,000 will furnish an ample supply. The capital is fixed at $3,000,000, of which the nominal vendors, the Venture Corporation, are to receive $2,550,000, leaving $450,000 as working capital. The mine, previous to the option being granted to A. E. Langford, had produced tin to the value of $201,800, but only $7,500 had been paid in dividends. The consideration to be paid to the Melbourne Co. was $750,000 cash and 25% of the capital of the new company.

Another discovery of tin was made at the Renison Bell mine at North Dundas, on the West coast. A face of gossan 40 ft. wide was opened up which is said to show 9% metallic tin. Other samples went 7·5%, 1·5%, and 1·2%. The Emu Bay Railway is expected to run through this section of the country.

Tin was found in the schistose formation near Mt. Lyell. A sample taken from a trench 33 ft. wide is said to have given a return of 7·42% Sn.

The Tin Markets in 1899.

New York.—The year just ended will long be a memorable one in the annals of the tin trade. The advance in values which took place in 1898 continued at a much accelerated pace amid violent fluctuations. The consumption was very large. Production increased but slightly. Speculation was rampant. The main features of the market were an increasing consumption in the face of constantly advancing prices, and the daring and successful speculative operations carried on in London. In spite of the great stimulus of high prices, production did not increase much. Last year 68,800 tons were produced, and we estimate this year's production to be about 71,500 tons. The average price of tin in New York during 1898 was 15·70c., while this year it was about 29c. Our market has, as heretofore, followed that in London, though at frequent periods supplies were exhausted, and spot and early deliveries sold at handsome premiums. It also happened that while the London supplies were meager, stocks at New York had accumulated, causing prices here to go far below the import parity. While in these instances the two markets did not move together, they did so most of the time. The London market must be kept well in view in considering the New York tin market during 1899.

When the year opened tin was selling here at 19c., and in London at £86. The demand was excellent, stocks throughout the world were small. The European operators soon recognized the strong statistical position and bought heavily, forcing the large short interest which existed at that time to cover. In consequence, the market rose to 25c. in New York, and £110 in London. The American consumption was rapidly increasing, and the stocks in this country were much depleted early in the month with the result that spot tin ruled at 0·5c. premium until the end of the month, when larger arrivals reduced it somewhat. The world's visible supply decreased 1,200 tons during January.

At the beginning of February the London market went to £114 2s. 6d., but now a reaction set in, the metal selling down to £105. At New York the metal

sold at 23·25c. Wide and violent fluctuations followed and consumers held off, but toward the end of the month they again had to buy, and prices crept to 24c.

During March business here was of small volume. Consumers were rather skeptical as to the continuance of the high values which had been established, and it was only toward the end of the month that the market became active. Meanwhile prices fluctuated widely in both New York and London. The shipments from the Straits had not increased, and early in April the prices rose again to 25·375c. in New York and £113 7s. 6d. in London.

Arrivals at New York during April were rather heavy, and the premium which had existed on spot tin was totally wiped out. It was now perceived that in spite of the fact that the metal had risen 50% in value, consumption was not being affected thereby, and in fact the visible supply during April was actually decreased by about 1,147 tons.

During May our market was exceedingly quiet; stocks were rather large. The London market fluctuated in an uncertain way and at the close of the month the metal was selling here at 25·75c.

Early in June it appeared that visible supplies had increased slightly as compared with those of January 1st, and the trade doubted the maintenance of the high prices which had been established. It certainly seemed that with supplies more than adequate prices would suffer. In London fluctuations were wide. At the end of the month, however, a bull movement was started with the result that prices advanced constantly, reaching £128 5s.

Till July our market responded but sluggishly to London fluctuations owing to the plentiful supplies on hand; but about the middle of July it went to 28·25c., consumers having exhausted their stocks.

During the six months that had passed the increase in consumption had taken place principally in America and England. On the European continent deliveries from warehouse and imports had actually decreased over 5,000 tons. This apparent reduction, however, appears to have been in consequence of consumers using up all stocks they were in the habit of carrying in their mills. It developed later in the year that the actual amount of consumption there had not changed, for during the next few months it increased to the extent it had fallen short, and it then became apparent that everybody had been working without stock.

July witnessed tremendous fluctuations in London, due primarily to speculative, and then to consumptive purchases. The metal sold up to £146 in London and 32·5c. at New York.

During August the London market fluctuated continually, but the New York stocks, in consequence of the heavy arrivals, being again replenished, the market here did not follow the London changes closely. The market here fluctuated from 30c. to 33c.

In September the market moved in an uncertain way, but the undertone was decidedly firm. Spot tin ruled at a premium. At the same time the London stock was being reduced owing to the fact that heavy shipments from the Straits were being made direct to New York. Their arrival at the middle of the month gave the New York market a rather weak tone, but at the end of the month a

strong speculative movement both here and abroad advanced the prices to 33c., while London touched £150. The September statistics showed an increase in the visible supply of 1,100 tons, which made consumers rather cautious and bull manipulations conservative.

During October the market was dull and weakish. At the end of the month, amid great excitement and wide fluctuation, London quotations fell sharply. In one day over £5 were lost. In New York tin sold for 30·75c., and at the end of the month the metal sold for 30·5c. Meanwhile the usual Dutch sale had been made for 400 tons more than the ordinary quantity, and this disheartened the bull manipulators, as well as the consumers in general.

Prices in November started on a downward course by leaps and bounds, and 26c. for New York and £118 for London was reached in the first half of the month. The London market had actually declined about £29 in two weeks, and the New York market about 6c. All sorts of rumors were in circulation, one of them to the effect that a stock held back by the Chinese at the Straits was being forced upon the market. However, the movement was due to the decrease in consumption and slight increase in production.

The third week in November witnessed the most violent fluctuations in London, with the result that business here was practically at a standstill. At the close of the week tin in our market was selling at 28c. and in London at £128. At the end of the month the market was about 27·5c. for spot and 27c. for futures.

The unfortunate trouble in the Transvaal and the reverses of the English, coupled with a tight money market, played sad havoc with the market during December. The consumptive demand showed a considerable falling off, most of the tin plate mills being closed down for inventory and repairs. It was not until the last few days of the month that a somewhat better feeling prevailed, and the year closed with Straits in carload lots selling at 25c. for spot, with furnace at 24c.

AVERAGE MONTHLY PRICES OF TIN IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1895	13·25	13·35	13·20	14·00	14·65	14·15	14·40	14·35	14·45	14·65	14·40	13·91	14·05
1896	13·02	13·44	13·30	13·34	13·54	13·59	13·63	13·49	13·15	12·94	13·09	12·96	13·29
1897	13·44	13·59	13·43	13·34	13·44	13·77	13·89	13·80	13·98	13·88	13·79	13·71	13·67
1898	13·87	14·08	14·38	14·60	14·52	15·22	15·60	16·23	16·03	17·42	18·20	18·30	15·70
1899	22·48	24·20	23·82	24 98	25·76	25·85	29·63	31·53	32·74	31·99	28·51	25·88	25·12

London.—The tin market, so often a source of bewilderment to onlookers, must have astonished the world at large in a more than ordinary degree at the beginning of 1899, when prices rose by leaps and bounds, and large transactions took place amid the wildest excitement. The phenomenal advance of £23 per ton during the month of January was a circumstance which many found it difficult to explain, but not only was the article in an exceptionally strong position, in the relation of supply and demand, but other powerful forces were at work which could hardly have failed in their object.

As in copper, America led the way in a great speculative movement, and the continual rise of prices in the States tended to make operations in our market bold. A large business was done in "options," at full values, and the optimism of the bull party knew no limits. Probably the most important feature of the market was the continual and steady decrease of stocks, coupled with a growing consumption with a production which did not increase, and here the great strength of the market really lay, as this state of things had been going on for some considerable time, and the "boom" in America naturally brought matters to a crisis. The values of cash and forward tin at the opening were only £87 and £88 respectively, but at the end of a week nearly £6 had been gained. A further rise of £4 induced some profit taking and brought a temporary reaction, but the close of January was strong at £111 10s. cash and £111 three months, the backwardation being caused by some free offerings of forward tin. The turnover was about 23,000 tons for the month. The stocks in Europe and afloat thereto at the beginning of the year were 15,666 tons, being a decrease of 1,934 tons on the previous month's supplies.

February marked a keen contest between the bull and bear parties, which swayed the market like a pendulum. Strong buying continued, until £115 was paid, but a weaker tendency in America then became manifest and the bears seized the opportunity they had long been waiting for and sold large quantities of forward metal at declining prices. These warrants, however, were boldly taken up by the bulls, who made no effort to stay the decline. Their faith in the market was justified, for no sooner had the selling ceased when a smart recovery took place, prices rising from £104 10s. to £108 10s. for three months in one day. On further buying £111 15s. was reached, but here the bears renewed their efforts and hammered down the market most effectively to the extent of about £7. At one moment the backwardation on three months tin was as much as 45s. A better feeling in America then asserted itself and once more £110 was paid, but the tone was unsettled, in the expectation of heavy. shipments from the Straits, and the bulls having relaxed their efforts, the close was quieter at £108 cash and £107 15s. three months. The consumptive demand at this time was good.

The shipments from the Straits during February were 4,600 tons, which was less than had been expected, and with the aid of better reports from America the market was strengthened, in the beginning of March, against the influence of free selling, which caused but temporary drops. Later in the month American purchasers came in, while the bear selling practically ceased, and the market assumed a sounder appearance, with a contango on three months tin; £109 was paid for cash, but the speculative interest became less strong, and after further unimportant fluctuations the close was steady at £109 12s. 6d. cash and £110 three months. Consumption, both in Europe and America, was very good and supplies were steadily falling off.

In April there was more interest shown in the article, consumers and speculators alike buying in anticipation of higher values. This improved prices, which reached £113 7s. 6d. early in the month. A further advance, as well as some excitement, was caused by an inquiry in the market for 2,500 tons from

consumers who were stated to be very large buyers for a new invention, prices rising to £117. No actual business seems to have resulted from the inquiry, but the market remained strong owing to the steady rise in silver and to the general opinion then held that the article was extremely sound on its own merits. The bears were defeated in their efforts to weaken the market, and the high price of £120 was reached, but this offered too strong a temptation to some holders, who sold freely, and prices fell about £2.

May opened with good statistics, which showed a decrease in the supplies of 1,277 tons. This encouraged large speculative purchases, which brought prices to £120 10s., but a reaction in the silver market had a contrary effect, while free offerings of forward metal by prominent bear operators further tended to depress the market, which fell nearly £5. Other operators were, however, equally energetic, while American advices were of a most optimistic nature, but, being pulled both ways, the market was irregular, and the month closed with values at £117 18s. 9d. cash and £118 10s. three months.

June began with a quiet and somewhat depressed market, but with a strong undercurrent which made itself felt as soon as any support was forthcoming. Good prices were realized at a Banka sale in Amsterdam. This, and the strong statistical position, acted partly as a counter influence against some heavy realizations by outside holders. A new speculative element then asserted itself and gave the market a very strong tendency, values rising to £120 17s. 6d. for cash and £122 for three months. Prices in the East were also higher and the supplies continued to fall off.

In July both consumers and speculators bought liberally and prices advanced from day to day, till £128 cash and £129 3s. 9d. three months were quickly reached. Attention was then concentrated upon the article simultaneously from London, America and the East, and a very large business was done, at advancing prices, amid all the excitement of a boom. Each week's advance eclipsed the former one, and by the end of the month prices had reached the sensational level of £144 10s. cash and £146 three months. The efforts to increase the production at this time were handicapped by the extreme scarcity of coolie labor in the East, and this was not without its effect upon the market.

Early in August the boom reached its climax, with forward metal at £147, and realizations on a large scale were then made, causing a drop of nearly £3 in a few days. America, however, stimulated the market with large buying orders, and values rose again to £146 cash, but home operators again seized this opportunity to realize their holdings and violent fluctuations followed. After touching £139 12s. 6d. the market improved somewhat and closed at £142 16s. 3d. cash and £143 forward.

In September the strength of the statistical position became a factor which was apparent to all. The supplies were continuing steadily to fall off and there was little hope of an increased production for some time to come. Strong reports from America came forward and a steady upward movement seemed to be in progress when a temporary relapse took place, chiefly under the influence of politics, the outbreak of hostilities in the Transvaal being imminent. Cash metal became scarce and caused a backwardation against forward tin of 15s. to

20s. In the latter half of the month, however, the market improved greatly under new support, both at home and in America, and large purchases brought prices up to the highest level reached, *viz.*, £150 15s. for cash. The close was quieter at £149 12s. 6d. cash, with forward metal at 22s. 6d. less.

In October the contango between cash and three months tin was restored, the dearness of money inducing sales of the former position. The statistics for the latter half of November stocks were not so favorable as had been expected, but the American stocks were believed to be much overestimated. Prices were sensitive for a time and influenced downward by cheap sales of Dutch tin, as well as the weaker tendency of other markets. The bears chose this opportunity to make a determined raid on the market, and sold aggressively, prices falling as low as £138 15s. for three months. This level, however, attracted the attention of the opposite party, whose purchases caused a recovery of £5, but under further realizations at the end of the month, prices declined again to £141 10s. cash.

The bears resumed their vigorous tactics in November and the market could no longer stand against them. Large quantities were sold at declining prices, and in the middle of the month there was a complete collapse, prices touching as low as £118 10s., but this was caused chiefly by a scare among the bulls, some of whom realized heavily. The market was not entirely abandoned, and when more courage was shown by its supporters, prices improved to £132 cash. It will be seen from these figures that the fluctuations were extremely violent, but the market was so sensitive that almost every transaction changed the level one way or the other. The lowest figures brought out some consumers' demand, as well as some new speculative orders, but the free selling continued from all sides, the high prices in this country offering temptation to sellers in the East. At the end of the month, however, the tendency was steadier at about £125 to £126.

In the beginning of December it looked as if prices were going to be more steadily maintained, but cheap offers from the East had a bad effect, and when £120 for cash and three months had been reached the sudden failure of a broker, whose commitments were said to be about 2,000 tons, caused most violent fluctuations, prices falling in two days to £108 for spot. Most firms on the Metal Exchange were affected by this failure and were anxious to resell the quantities suddenly thrown on their hands. Strong efforts, however, were made to check this decline and resulted in an improvement to £114, but this was followed by a further reaction, owing to realizations, and £109 5s. cash and £110 2s. 6d. three months were accepted. The same influences which depressed the other markets, particularly the stringency of money, helped to weaken the position of tin, which might otherwise have borne this failure more firmly, and values fell to £103 for cash and £104 10s. for forward. Renewed buying from America, coupled with better news from the Transvaal, gave the market a firmer tendency at the end of the month, and the year closed with spot selling at £111 15s., three months at £112 15s.

Notes on the Metallurgy of Tin.

By Henry Louis.

In spite of the high price and keen demand for tin, there has been practically no change in the sources of supply during the past year. In Europe, Spain alone has shown any alteration; the Almaraz tin mines have been definitely closed down, no smelting having ever been done by this company. The San Finx mines have produced a fair amount of tin ore, and the company has paid a dividend; the ore produced has been sold in England. It is of fair quality, but contains considerable wolfram. Messrs. Lake and Holloway have patented a process for converting wolfram into sodium tungstate by a continuous acting electric furnace, but this does not yet seem to have been used on a working scale.

Tin has recently been coming into the London market from the Niger district; it is smelted by the natives, apparently from stream tin ores, and is sold by them in the form of cast rods about 12 in. long and 0·125 in. in diameter. The methods of smelting and casting are not known. The tin is of very high quality, containing 99·9% Sn. There is every probability that this source of tin will be thoroughly explored and developed in the near future.

The Smelting Works at Tostedt, Germany.—These works, noteworthy in many respects, are owned by Robertson & Bense, of Hamburg. The ores treated are practically all Bolivian ores, small parcels of other ores being rarely purchased by the firm. It is worthy of note that the situation of the works, at first sight apparently quite unsuited to their purpose, in reality presents many advantages. Although remote from the centers of production of either the ore or the fuel, the works nevertheless lie on the great trunk line that connects the important seaports of Hamburg and Bremen with the coal fields of Westphalia; coal can thus be brought readily and cheaply to the works under the differential rates granted by the German Government, so that, although inferior in quality to imported Welsh coal, it is found more economical to obtain fuel from the German coal fields. The ores treated are imported through Hamburg or Bremen, and the metal produced is easily forwarded to market at home or abroad by rail or sea. Robertson & Bense buy through their local agents from 3,000 to 4,000 tons of Bolivian ore a year, which constitutes about two-thirds of the entire annual ore output of that country. These Bolivian ores vary greatly in character, and contain many impurities, such as bismuth, lead, copper, antimony, arsenic, sulphur, etc.; they are usually of low grade, carrying 40 to 60% Sn, 10 to 30% SiO_2, and up to 10% Fe. The ores arrive in bags containing about 1 cwt. each, which when delivered to the works are opened and examined, sampled, assayed, and very carefully classed according to their qualities.

Each batch of ore, if not already in fine powder, is ground in a ball mill. The ground ore is then charged into the roasting furnaces and calcined until perfectly sweet. There are three reverberatory roasting furnaces, of the "Fortschaufelungsofen" type; one has two hearths, and the remaining two have four hearths each; the hearths of the latter, which are of newer construction, are

carried on iron rails so as to be easily accessible for repairs. The smelter furnaces calcine from 2·5 to 4 tons, the larger from 5 to 10 tons of ore per 24 hours, the amount put through varying with the percentage of sulphur present. Great stress is laid upon the thoroughness of the calcination, the object being to remove all sulphur entirely, together with as much arsenic as possible. The ore when thoroughly calcined is drawn out onto the floor in front of the furnace, where it is allowed to cool. It is then wheeled to the smelting furnace where the charges are mixed; these consist of the roasted ore with 15 to 20% of finely ground anthracite coal and lime, the amount of the latter being 50% more than that required to flux the silica in the ore, the composition of slag aimed at being a neutral silicate.

The smelting furnaces are of the usual reverberatory type with working doors at the front and side. In the most recent form, the entire furnace is carried upon iron girders supported upon cast-iron pillars set in rectangular brick-lined pits about 5 ft. deep, and is thus entirely free from the floor. The hearth proper is carried on iron rails. The workmen thus have free access through the ashpit to the space beneath the bed and can easily remove and collect any tin that may trickle through the hearth or through the masonry of the furnace. The air space beneath the furnace is kept well ventilated. The bottom is made of brick, upon which is burnt the working bottom proper, made of sand. This sand bottom is dished out and slopes in the usual way toward the tap hole and lasts, with occasional repairs, about one month. The maximum length of the bed is 23 ft., and its maximum width, 16 ft.; its area is 300 sq. ft.; the grate area is 28 sq. ft.

There are two large and two small smelting furnaces; each of the latter can treat 4 tons, 5 cwt. of ore (or 4 tons, 15 cwt. of slags) per 24 hours; the larger of the others can treat 6 tons, 15 cwt. of ore (or 8 tons of slags), and the smaller, the dimensions of which have just been given, 5 tons, 10 cwt. of ore (or 6 tons of slags) per 24 hours. It is intended to work three of the four furnaces continuously, keeping one in reserve. The average time of working a charge is from 8 to 10 hours with a coal consumption of about 50% of the weight of the charge. The temperature is kept fairly low when smelting ore, the slag smelting requiring a somewhat greater heat. When the charge is thoroughly melted the tin is tapped out into a cast-iron kettle and the slag is run into sand beds. The solidified slag is then broken up and recharged into the reverberatory furnace, together with about 6 to 8% of hardhead obtained from the liquating furnace; the slag charge is worked precisely like the ore charge, the products being tin of a poorer quality and slag which is sent to the blast furnace. The latter is a cupola furnace about 30 ft. high and 7 ft. diameter at the boshes, and has 3 tuyeres; it can smelt 20 to 24 tons of slag per 24 hours. The coke consumption is 15 to 18% of the weight of slag smelted, 6 to 10% of hardhead being also added. The resulting slags are resmelted in this furnace once or occasionally twice before they are clean enough to be thrown away. The clean slags contain less than 5% Sn. The work-tin produced in these various smelting operations passes to the refinery, in which are two liquating furnaces of rectangular bed about 18 ft. long by 8 ft. wide, sloping uniformly from one side to the other, and

having a fire-place at one end. The bed is divided into three compartments, and in refining the ingots of work-tin are charged at the upper edge of the central division, and the tin liquates out and runs through a tap hole at the lower edge into moulds placed to receive it, leaving a mass of refinery residues on the bed. These residues are then transferred to the first division of the bed nearest to the fire-bridge, which is naturally the hottest; an additional amount of tin is thus obtained which is run into moulds and the resulting ingots charged into the second division, the residual hardhead being drawn out. Meanwhile all the tin that has run out of division No. 2 of the bed is placed in the third or coolest compartment and then again liquated, yielding a tin which is ready for poling, and a small amount of residues which is transferred to the second division of the bed. Each liquating furnace can treat 3 tons of work-tin in 24 hours.

There are two hemispherical cast-iron poling kettles, each holding a charge of 6·5 tons of tin, which is melted down slowly and poled in the usual way with a piece of green wood for half an hour to one hour, after which the refined metal is ladled into moulds. A kettle can treat two charges in a 12-hour shift.

The works produce three grades of tin depending mainly upon the quality of the ores employed:

I. "Bell" tin, guaranteed 99·90% tin, and often containing up to 99·95%.

II. "Extra" tin, containing 98% Sn and about 1% each of antimony and lead, with traces of copper and iron.

III. "R. B." tin, containing 95% Sn, with varying amounts of impurities.

Owing to the large proportion of impurities present in the ore treated, notably of iron, the production of hardhead in the course of liquation is in excess of that consumed in smelting, and various processes have been devised for utilizing the considerable proportion of tin which it contains. One process consists in heating the hardhead with metallic lead, producing solders (alloys of lead and tin) of varying composition and leaving a residue, mainly of iron with a small amount of tin and lead, too small to warrant further treatment. This process is quite successful technically, but the demand for alloys of lead and tin of somewhat variable composition is by no means steady.

An electro-chemical method is now being tried which promises satisfactory results, although it has not as yet become perfectly practical. The slag is granulated and treated with dilute sulphuric acid heated by steam, yielding a solution of tin sulphate which is used as the electrolyte; slabs of hardhead form the anodes and sheets of tin the cathodes and all are contained in a long rectangular tank similar to those in use for electrolytic deposition. By regulating the current strength and the strength of the solution (the latter being kept in active circulation) a firm deposit of tin can be obtained.

The Tostedt works are able to treat tin ores which are probably the worst and the most impure in the world, and yet produce a tin equal to the very best in the market; moreover they know exactly the composition of the metal produced, and can therefore say at once which of their products is by its chemical composition the best suited for any required purpose, thus representing a stage of progress far in advance of that attained by the Cornish tin smelters.

TUNGSTEN.

THE production of tungsten ore in 1899 was 89 long tons of high grade ore carrying 60 to 70% of tungstic acid. In addition the mines in Lawrence County, S. D., produced 106 tons of ore containing 38 to 50%. About one-half of this ore was taken from the Durango mine, the balance coming from the Sula, Harrison, Golden Summit and Two Strike mines. The ore as mined contains 10 to 15% tungstic acid. A small amount was also produced in Connecticut. Poulot & Voilleque mined three tons of ore at their Colorado mines. About 7 long tons of ore containing 65 to 68% tungstic acid was shipped from the Arizona mines to Philadelphia.

Stein & Boericke, Ltd., of Philadelphia, manufacture metallic tungsten as well as ferrotungsten, used in the preparation of tungsten steel. They report a product of 102,000 lb. of metallic tungsten, ferrotungsten and tungstic acid. There was considerable variation in the prices of tungsten and ferrotungsten due to sharp competition and dealers were unwilling to make quotations. In 1898 tungsten sold from 60c.@$2 per lb., and 37% ferrotungsten from 27@ 56c. per lb. The prices received for ores also varied widely. The Arizona shipments were marketed in Philadelphia for $490 per ton for 68% ore. Shipments to the Chrome Steel Works, Brooklyn, N. Y., were valued at $500 a ton on the basis of 60% ore, but more recently purchases were made of 50% ore for $225 per short ton.

A deposit of wolframite was discovered recently at Northeast Margaree, Inverness Co., Nova Scotia. The vein is said to be 3 ft. wide, and there is said to be a considerable quantity of ore, which can be concentrated to a high percentage. It contains 67·47% tungstic acid. Near Thornboro, in Queensland, wolframite was found associated with native bismuth and molybdenite. Small quantities were exported. It was sold locally for £30 per ton. Scheelite was found in Otago, New Zealand, near Lake Wakatipu, where there is a 4-ft. lode, half of which is scheelite.

NOTES ON THE METALLURGY OF TUNGSTEN.

BY W. BORCHERS.

THE ores from which tungsten is extracted are tungstic ocher, WO_3, scheelite, $CaWO_4$, and wolframite, $FeWO_4$. Of these, the last named frequently accompanies cassiterite, upon whose reduction in the furnace it has an injurious effect, owing to the fact that it promotes the slagging of tin in large amounts. Tin slags are consequently a source of tungsten and are valuable from the fact that they are easily treated. The ores are suited for direct reduction in the furnace in case it is desired to make ferrotungsten; for preparation of the metal the pure oxide must first be secured.

Treatment of Ores for Ferrotungsten.—After crushing, the ore is mixed with the desired amount of iron in the form of oxide or scrap with the addition of 10 to 12% charcoal, 5% glass, and 10 to 12% pulverized quartz. If the ore is specially rich a small quantity (not more than 5%) of pitch or resin should be added. The mass is then placed in crucibles and smelted in the manner described under chromium.* The product, ferrotungsten, will be in the form of a metallic button.

Treatment of Crude Ore for Metallic Tungsten.—As with chromium the preparation of tungsten can be carried out to the best advantage by first converting it into a soluble sodium salt, separating this from the insoluble compound, iron, calcium, tin, etc., by lixiviation and filtration, and then changing the salt into oxide.

Oxidizing the Ore or Slag by Roasting with Soda Flux.—The ore or slag is mixed with soda in proportion to form sodium tungstate and roasted in a reverberatory furnace at a temperature below the melting point of the flux. The heat should not be any higher than is necessary to sinter the ore slightly.

Lixiviation of the Ore.—After roasting the ore is withdrawn hot from the furnace and placed in a kettle of water. The sudden shrinkage will break up the masses of molten material. The lye is then evaporated after it has obtained a strength of 10 to 12% tungsten. The insoluble residue of the ore is removed, washed and the wash water used for extracting freshly roasted ore. By further evaporation the mechanical impurities and some of the salts held in solution, as sodium sulphate, the excess of soda, etc., will separate, after which the liquid should be allowed to clear and then be withdrawn into new vessels. The evaporation may be continued until the sodium tungstate crystallizes, but this is not necessary unless the solution is still impure.

Precipitation of Tungstic Acid.—The concentrated lye, or solution of the crystals, is next treated with hot hydrochloric acid. The heating of the acid can be conveniently effected in a stoneware vessel into which hot steam is conveyed through a lead pipe. The hydrochloric acid precipitates the tungsten as oxide WO_3 in the form of a yellow powder, which can be recovered by filtering.

Reduction of Tungstic Oxide.—This is accomplished after mixing 11 to 12% charcoal and 5% pitch or resin, by heating in a regenerative gas furnace of the type described under chromium, or if a melt is desired, in an electric furnace. Tungsten, like chromium, has an affinity for carbon, and care should be taken to avoid an excess of the latter above what is required to effect reduction. According to the method employed the tungsten will be in the form of a gray, heavy powder (sp. gr., 19) or a hard, dense white metal. Although it has a high melting point, it is easily taken up by molten iron and other metals. With iron it forms a dense, hard alloy, which finds application in the manufacture of mechanics' tools and other instruments where great hardness is required.

Many self-hardening cutting tools are made from tungsten steel, which has the special property of preserving its hardness even when heated to temperatures that would rapidly draw the temper from ordinary high carbon tool steels.

* See "Notes on the Metallurgy of Chromium," under the caption, "Chromium," earlier in this volume.

ZINC.

THE production of spelter in the United States during 1899 was 129,675 short tons, an amount exceeding the production of any previous year. The increase over 1898 was 15,571 tons. All the smelting districts, excepting in Missouri, gave an increased yield. The Joplin region showed great enterprise, and the good effects as to the increased yield will be shown in 1900 as well. The smelters kept pace with the advance in ore production and yielded larger outputs. The production of spelter by districts is given in the subjoined table:

PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1894.	1895.	1896.	1897.	1898.	1899.
Illinois and Indiana	28,948	33,748	31,656	38,680	46,698	49,290
Kansas	26,060	25,916	36,566	33,395	38,543	55,872
Missouri	12,000	10,726		18,412	21,063	15,710
South and East	6,996	11,468	9,415	9,900	7,805	8,803
Total tons of 2,000 lb.	74,004	81,858	77,637	100,387	114,104	129,675
Total tons of 2,240 lb.	66,074	72,088	69,319	89,628	101,879	115,781
Total metric tons	67,135	74,245	70,432	91,071	103,514	117,644

This shows an increase of production over 1898 of 15,571 tons, which extended to all districts except Missouri. The gain in Illinois and Indiana amounted to 2,597 tons; while Kansas showed an increase of 17,329 tons, that more than offset the decrease in Missouri, which amounted to 5,353 tons. While the ore is produced in both Missouri and Kansas, it is smelted for the most part in Kansas, as nearly all the large smelters have moved their plants to points on the Iola natural gas belt. It is worthy of note that the production in Missouri and Kansas in the first half of the year was practically 10,000 tons greater than in the second half, a condition which was due to the higher price of the metal in the former period.

The consumption of spelter has been estimated as follows: 20% sheet, 15% brass and yellow, 50% galvanized, 15% sundries.

There have been no marked changes to report in the East and South. The New Jersey Zinc Co. remains the sole producer in the East while the Bertha Zinc and Mineral Co. is the principal producer in the South. The total increase in this division over the 1898 production amounted to 998 tons.

The principal producers in Illinois are the Matthiessen & Hegeler Zinc Co.

and the Illinois Zinc Co.; there are three other companies of small production. In Kansas, of the eight companies, the Lanyon Zinc Co. is the principal producer and S. C. Edgar in Missouri contributed the greatest individual output of the plants there located.

The strong demand for ores has stimulated to some extent the working in the old mines in northwest Illinois and southwest Wisconsin, but no new ore deposits of importance have been opened there.

The Joplin district was the scene of activity during the year, and the important changes in mining and selling the ores there carried out are fully set forth in the section on the progress of the metallurgy of zinc in the United States during 1899, given later in this chapter.

The production of zinc oxide in the United States during 1899 amounted to 39,663 short tons, an increase of 6,916 tons over the production in 1898. Zinc oxide has been in large demand, stimulated by the comparatively low prices at which contracts for the year were made. This branch of the industry is still on a sound basis. There was an increase in average value at the works from $68 per ton in 1898 to $84 per ton in 1899, due to the advanced cost of labor, ores, supplies and freights. The greater part of this output was made by the New Jersey Zinc Co., at Jersey City, Newark, N. J., Bethlehem, Pa., and Mineral Point, Wis. Page & Krause, at St. Louis, make annually from 400 to 500 tons and the American Zinc and Lead Co., of Cañon City, Colo., produce a pigment which is a mixture of oxidized compounds of zinc and lead.

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES.

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Tons.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1896.....	15,863	14,391	$1,189,725	$75·00	1898.....	32,747	29,708	$2,226,796	$68·00
1897.....	26,269	23,825	1,686,020	64·20	1899.....	39,663	35,982	3,331,692	84·00

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES. (IN POUNDS.)

Year.	Sheets, Blocks, Pigs, and Old.		Manufactures.	Total Value.	Oxide.	
					Dry.	In Oil.
1895.....	864,113	$29,352	$12,189	$41,545	4,546,033	129,343
1896.....	856,044	25,904	15,728	41,632	4,572,781	311,033
1897.....	2,557,341	95,883	19,481	115,314	5,564,753	502,357
1898.....	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899.....	2,985,463	151,956	14,804	166,750	3,012,709	41,699

EXPORTS OF ZINC AND ZINC ORE FROM THE UNITED STATES. (IN POUNDS.)

Year.	Ore and Oxide.		Plates, Sheets, Pigs, and Bars.		Manufactures.	Total Value.
1894.....	.....	$5	3,621,934	$144,278	$99,418	$243,701
1895.....	48,000	1,008	3,141,285	155,975	50,126	207,109
1896.....	4,648,000	47,406	20,300,169	1,013,620	51,001	1,112,029
1897.....	*a* 22,220,907	315,490	28,490,662	1,356,588	71,021	1,743,049
1898.....	*b* 31,418,859	552,064	20,998,418	1,083,959	138,165	1,724,188
1899.....	*c* 67,126,506	1,092,543	13,509,316	742,521	143,239	1,878,295

(*a*) Includes zinc oxide, 3,718,507 lb. ($104,140). (*b*) Zinc oxide, 7,849,059 lb. ($252,194). (*c*) Zinc oxide, 10,685,226 lb. ($366,598).

The use of zinc chloride for preserving railroad ties has been developed to a considerable extent. Of the nine concerns engaged in this industry, four of the larger reported a consumption of 1,888 short tons. It is estimated that the consumption by all companies in 1900 will amount to 3,000 tons.

The American Smelting & Refining Co. at its Argentine works, produced in 1899 as a by-product in smelting lead ores, 528 short tons of zinc sulphate. This zinc salt was also manufactured in the United States by several chemical companies. It is reported that the greater part of the output is consumed in the manufacture of glue and special paints.

Colorado.—Considerable ore was shipped from Creede, Leadville, and other points to the zinc smelters for experiments involving the use of large quantities in the furnaces.

While the zinc ores of Colorado are generally too impure from their iron and lead content to be considered desirable for zinc smelting, there have been some clean concentrates made that, considering their low price, were available for profitable treatment.

About 8,000 tons at a nominal price were shipped to foreign smelters via Galveston.

Iowa.—(By H. Foster Bain.)—The Dubuque region has long been famous for its lead mines. These began to produce in 1788, and were a very important source of lead from 1830 to 1860. With the opening of the lead-silver mines of the West, the importance of the Dubuque region declined until at the present time the annual production is only about 1,000,000 lb. of pig lead. The early lead miners knew nothing of zinc ores, and long after their discovery in that region the market conditions rendered them of no value. Since 1881 the Iowa mines have sold the carbonate of zinc or "drybone," and for years blende has been known to exist in that region. The principal buyer has used the carbonate ore for making zinc oxide, a purpose for which it is in great demand. The price of the carbonate ore, however, has not been sufficient to stimulate its production, and no considerable bodies of blende have been opened up in working for it. There has been no special demand for blende itself.

The drybone is the alteration product of blende, and is advantageously located above the water level so that is can be easily mined without heavy outlay. The price has been so low, however, that small profits only have been made. The blende generally occurs below the water level, and requires a considerable investment for a plant with pumps before the ore can be mined. Furthermore, milling is necessary, as much of the ore occurs mixed with rock, or as a lining to small cavities in the dolomite which have been formed by the solvent action of water. These facts have prevented the development of the ore deposits by the class of miners engaged in mining, and occasionally shipping drybone. No important mining enterprise has been carried on in the region, and the people have not been stimulated to take hold of the work either directly or by familiarity with methods and results obtained elsewhere. Up to this year no practical effort has been made to mine for zinc blende.

The Alpine mine is fortunately situated near a bluff where the water level is low. Recent geological changes in water level, due possibly to the agencies

of the glacial period, have exposed a body of mixed bone and jack, or partially altered blende, which is now being mined. A crude hand concentration cleans the ore sufficiently to make it salable as spelter ore. The price realized was so much better than the price of the bone ore alone that other mines were opened up and deeper workings made for jack. The Dubuque Ore Concentrating Co. has erected an 80-ton custom mill for coarse concentration and has contracts with the Alpine mine and others.

A few experienced Joplin men have come into the region, and a certain amount of local capital has been enlisted. A number of properties are now nearly ready to produce ore. The Pike's Peak Co. has shown that in ordinary situations pumping is no more difficult than elsewhere. It has demonstrated also that the lower ore horizons which yield fine results in Wisconsin are ore-bearing under cover at Dubuque. Deeper work on the Levens, Bush, Avenue Top, Langworthy, Rabbit Hollow and Jew ranges shows that ore of good character occurs below the old lead and drybone levels, an occurrence similar to the Alpine mine. At present pumps are being installed in four mines. The Alpine and the Langworthy mines are opening up deeper levels without the necessity for pumping, and the Bush and Avenue Top are arranging to drain their workings by a short level. The conditions for 1900 are very favorable.

The zinc ores at Dubuque are found in and under old lead mines, and in mining them a certain amount of lead ore is produced, although the bodies of the latter mineral seem more uncertain and of smaller average size. The zinc ores occur in what are known as crevices that have well defined courses, and have been traced in individual cases for five miles. The main crevices are nearly parallel and run approximately east and west. They cross at low angles and are crossed by north-south crevices, which act as feeders. The intersections are the most favorable places for "chimneys" or ore shoots. The ores are usually found in greatest abundance at certain horizons along which the crevices normally broaden out into "openings" that may be packed with ore or may be open and cave-like. The crevices extend to the bottom of the Galena limestone which is here from 225 to 250 ft. thick, and may prove valuable through the underlying Trenton limestone, although this adds 50 ft. to the depth of the workings. Below this they are doubtless cut off by the Saint Peter sandstone, an important water bearer. It is not practicable to mine here below the base of the Trenton. In Wisconsin the ores in the lower horizon assume the form of "flats" or blanket veins. It has not been proved that the deposits in Iowa are similar as the development has not been carried low enough to demonstrate it; certain significant phenomena, however, point in that direction. The region has not been violently disturbed, but certain low broad folds have traversed it, and given direction to the parallel crevices. The best ore bodies have been found on the sides of these folds and, in Wisconsin, in the synclines. In Iowa there is no important faulting beyond that found in the disturbed rocks. Analyses demonstrate the presence of the ores in small quantity disseminated throughout the rocks and the ore bodies are doubtless concentrations of this disseminated material. While not yet proved it may be considered doubtful that these disseminated bodies are themselves worth working at present. The ore bodies at

Dubuque are smaller than those in the Joplin region, but they possess an advantage in that they are richer, and have more guides to follow in locating them. The mined ores contain from 10 to 50% Zn, with the average between 20 and 25%. At present the lower grade ores are left in the mine, possibly in the future they will be taken out. Ores containing much less than 20% of Zn, have not been extensively handled. In 1899 nine mines working intermittently produced 2,174 tons of zinc ore, which sold for $20,542. Most of this was dry-bone, which averaged about $10 a ton. A little blende was sold at prices ranging from $18 to $34 a ton, with an average value of $25 a ton. Certain of the mines produced an ore carrying pyrites which cannot be jigged out. Ore of this character does not command a high price.

New Jersey.—The following notes on the recent improvements of the New Jersey Zinc Co. have been contributed by A. Heckscher. At Franklin Furnace exploration has shown up an ore body of unique composition and unusual extent. A separating plant has been devised under the Wetherill system, auxiliary to the one already in operation, and of double its capacity. The new plant will readily crush, size and separate into its constituent minerals 1,000 tons of ore daily, and will enter the field during the year 1900. After separation, a portion of the willemite ore is exported and the remainder utilized here for spelter manufacture. All of the franklinite ore is used primarily for zinc oxide and subsequently for spiegeleisen in blast furnaces.

At Palmerton, near Mauch Chunk, on the Lehigh & Susquehanna branch of the Central Railroad of New Jersey, the same company is erecting a plant for the manufacture of zinc oxide, spiegeleisen and spelter from its ores. The plant is of ample capacity to manufacture eventually all of the company's products and it has been planned with a view toward the installation of every approved method now known for the economical utilization of fuel and labor and the distribution of power.

A small though interesting industry, which owes its origin to zinc, has for two years past maintained a steady growth at Bound Brook, N. J. It is the manufacture of liquid anhydrous sulphurous acid gas from the sulphur gases obtained from zinc blende roasting or from brimstone burners, under Dr. Max Schroeder's patents. This process has been described in detail elsewhere in this volume, and reference should be made thereto.

Price fluctuations abroad have also been violent; in the latter part of the year there was a material decline, particularly in the prices of ore. In the ore field high prices and great scarcity of furnace supplies at the end of 1898 stimulated the reopening of old mines and explorations in new fields. From Australia considerable ore has been shipped and is in the way of being shipped. Much of it is made up of concentrates produced through the installation of a large plant under the Wetherill system. It must be said, however, that the Australian ore, so far as known, is not very desirable furnace material, as it contains many impurities.

Missouri and Kansas.—The year 1899 has been one of the most marked in the history of the Missouri-Kansas zinc mining district, and great progress has been made in the methods of operation, the amount of money invested and

the class of investors. The output of zinc ore in 1899 amounted to 512,912,390 lb., an increase of 41,665,770 lb. over the 1898 production of 470,246,620 lb.

Zinc ore reached the unprecedented price of $55 per ton in April of last year, and the average price was $10·10 per ton higher than in 1898 throughout the entire year. Market conditions have been disturbed from various causes and the demand for metal has been cut short by the war in South Africa, the inability of sheet-iron manufacturers to deliver sheets to galvanizers and from other causes.

The Missouri & Kansas Zinc Miners' Association has prevented a surplus from accumulating by ordering a shut down of operations on June 26th, continuing until July 10th, and again on October 1st, continuing for four weeks. In the latter fully 90% of the mills in the district participated, and the amount of ore affected by these two suspensions was about 20,000 tons. The price of metal fell steadily during the last six months of the year, but in spite of a falling metal market and a lessened demand for spelter, the prices were maintained for a while, probably due for the most part to the extreme competition between the older smelters and the newer ones in Kansas, using natural gas as a fuel. The close of the year saw top-grade zinc ore selling at $36 per ton when the association schedule price was only $33 per ton for 60% ore.

The ratio of increase in the production of zinc ore was not as great compared with 1898 as that year was compared with 1897, when the production increased over the previous year to the extent of 114,900,400 lb., while the comparatively small increase of 41,665,770 lb. over last year, with the greatly increased demand and far better prices, shows that there is room for expansion in the production of zinc without overloading the market. The great increase in valuation indicates the large profits that may be made in the industry.

The wages of miners have been advanced throughout the district from 20 to 25%, and the great increase in the number of mills has advanced the wages of unskilled labor to a marked degree. Ordinary labor is paid from $2·25 to $2·50 per day, ground men and ground bosses from $2·75 to $3·50 per day. There have been no labor disturbances of any kind during the year.

The hand-jig system of cleaning ore has been almost entirely superseded by the modern high-power steam mill, and it is estimated that over 400 steam plants are now in operation in the district, half of which have been built during 1899.

The year opened with top-grade zinc ore selling at $29·50 per ton, but it had advanced to $32 on January 14th, sold for $48 per ton during the week ending February 18th, and continued to advance until, during the week ending April 22d, the price for fancy grade zinc ore was $55 per ton. It then commenced to decline slowly, reaching $40 per ton November 4th, and touched the lowest point on November 20th, when it sold at $32·50, from which it advanced to $37 for the week ending December 24th, and closed the year at $36 per ton. The average price for the first six months of the year was $35·98, and for the last six months $41·925, the average for the year being $38·54.

The average price for the year 1898 was $28·44 per ton, against $38·54 for

this year. A comparison of these figures with those for 1898, until then the greatest year in the history of the zinc and lead mining industry, shows the remarkable stride made in 1899. The average monthly prices paid during the past four years are given in the following table:

Year.	Jan.	Feb.	March	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average
1896.....	$24·00	$23·50	$23·00	$23·00	$21·50	$21·00	$21·50	$21·00	$20·00	$20·50	$23·50	$25·50	$22·33
1897.....	22·125	21·50	21·00	21·125	21·60	21·875	22·50	22·50	22·625	22·75	23·50	24·25	22·28
1898.....	23·00	22·50	23·00	24·62	26·50	28·50	28·00	28·37	31·00	33·70	36·25	37·00	28·44
1899.....	32·25	43·37	43·40	51·50	50.50	45·50	44·20	45·00	43·75	43·50	35·00	36·00	38·54

The combined value of lead and zinc ore for the year was $10,883,628, against $7,145,262 for the year 1898, and the increase in the zinc output amounted to 41,665,770 lb.; the lead output, however, was less by 4,193,160 lb.

The prevailing high prices for zinc ore naturally stimulated prospecting in this section, and an item of interest and value in this connection is the Crossmann drill chart which is given in the article on the Galena-Joplin Lead and Zinc Mining District, elsewhere in this volume.

This chart was completed from the records of 30 years' drilling and showed that more than half of all drill holes of over 500 ft. depth struck ore. As a result of this systematic work, the system of drilling has been changed to a marked degree and drilling to 250 ft. and more has increased the number of ore discoveries fourfold.

Probably $12,000,000 has been invested in mines and mineral lands during the year. A large number of the sales are not published, and with many of the published sales no valuation is given out. The sales actually published amount to $9,682,230, and the Secretary of State for Missouri has granted during the year certificates of incorporation to 151 domestic corporations, with a total capital of $12,256,700 and 36 foreign corporations, with a capital of $10,350,000; a total of $22,606,700. Among the companies which have acquired large holdings during the year are the American Zinc, Lead and Smelting Co., whose combined properties represent an investment of $1,200,000; the United Zinc Companies, which have invested $642,000; the Colleys, who have placed properties to the extent of $2,506,000; and others ranging from $50,000 to $300,000.

Tennessee.—In the Eastern district, while considerable prospecting work was done during the year, there was but little commercial activity in zinc ore mining. The old Mossy Creek workings were reopened and several hundred tons of good grade concentrates were turned out by the mill and shipped to a smelter at Marion, Ind. The ores at Mossy Creek, New Market and near Knoxville are all of the same general character, carbonates being found at the surface, usually in small quantities, these deposits soon giving place to blende of light yellow color disseminated in magnesian limestone.

Virginia.—Pulaski County is being exploited by the Bertha Zinc Co., and while no zinc ore is being mined at present the company is shipping about 1,000 tons of limonite per month. The property presents a heavy outcrop of limonite in clay, under which is found massive calamine very intimately mixed

with limonite. Efforts are now being directed to the matter of magnetic separation and until that question is settled the property cannot become a zinc producer, as the calamine carries far too much iron to be commercially available.

A radical change in the system of mining has been made at the Bertha mines, in Wythe County during the past year. The pure calamine ores found occur in pockets contained in stiff clay or "buckfat" and occupy a limestone floor, the whole being overlaid with a heavy cover, from 25 to 125 ft. thick, of clay carrying limonite. For years the zinc ores have been won by underground methods, the shafts being sunk through the overburden and every foot of the soft ground being timbered until the zinc ores could be withdrawn. During the past year it was found that the heavy overburden on most of the mineral area was available as wash iron ore, and it is now being stripped extensively by open cuts and shipped to the Pulaski Iron Co.'s furnace. As this cover of iron clay is removed the deeper bodies of calamine are made quite available by open cut work. The iron ore obtained here is free of zinc, and the zinc ore practically free of iron, a stiff clay parting usually separating the minerals.

The Wythe lead and zinc mine in Wythe County is in active operation. The former underground methods are being abandoned and open cut work substituted with good results, the wash iron ore in the overburden being saved. This company manufactures and sells its lead output, sells its iron ores on the open market and delivers its dressed zinc ore, the chief product, exclusively to the Bertha Co. The property is equipped with an excellent plant, the water power of the New River being utilized as electric power for the varied purposes of mining, hauling and washing the ores. The ores in this mine being mixed necessitate a separation of the iron from the zinc, and this is accomplished at present by a Payne separator working on roasted ore.

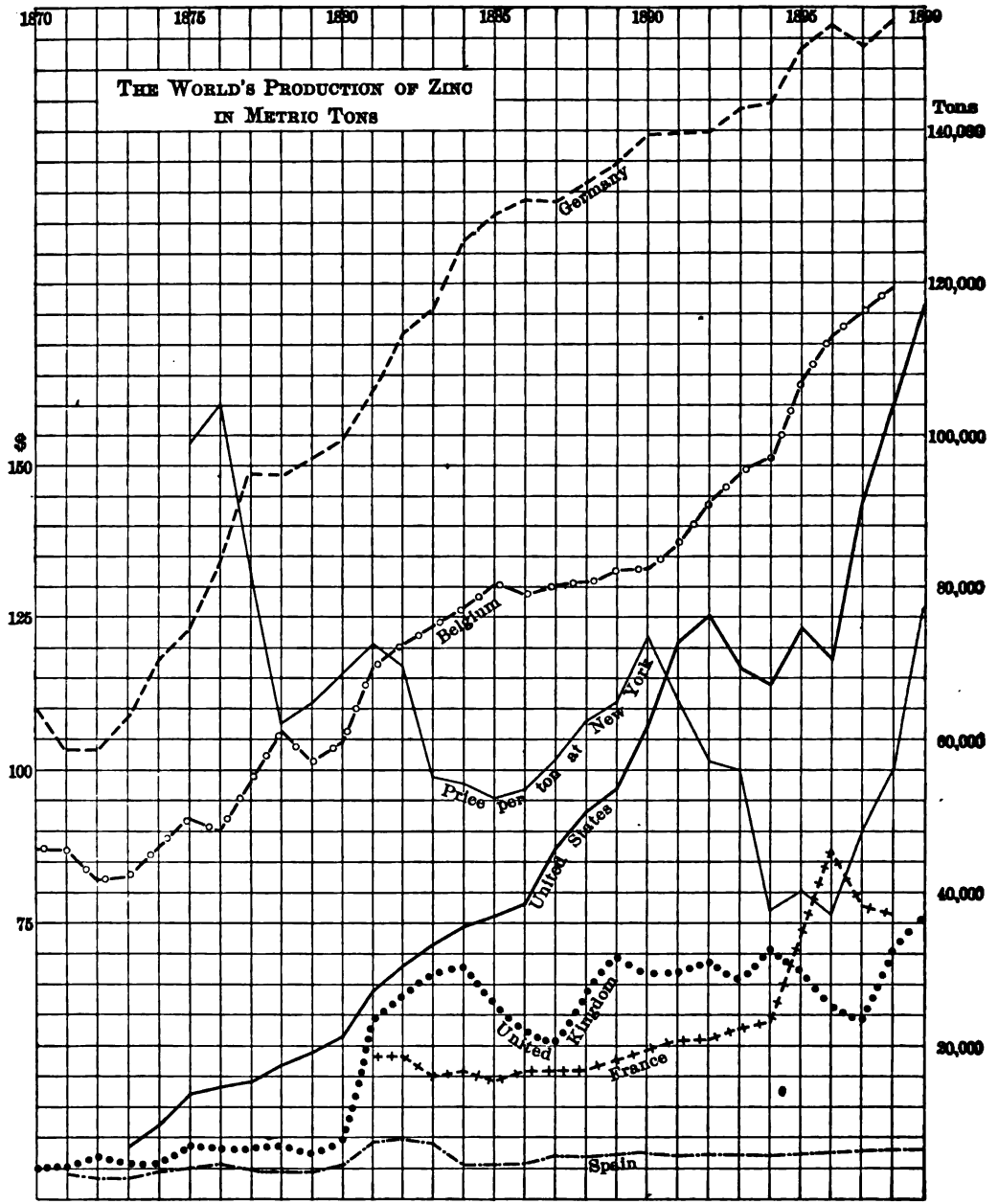
Small quantities of disseminated zinc blende in limestone are of frequent occurrence in Virginia, being found from Roanoke on the east, to Knoxville, Tenn., on the west. At no point along this belt in Virginia, however, are there any shipping mines of these ores, notwithstanding quite active prospecting operations from time to time. During the year more or less development work has been carried on at a number of places, among which are noted the following:

Martin Mine, Roanoke County.—A large open pit has been excavated and a shaft sunk in limestone. About 250 tons of rock carrying rather lean blende has been hand sorted and awaits concentration. It is reported that a company has been organized in Philadelphia to build a mill and work the mine. This mine was unsuccessfully worked for silver many years ago.

Keesling's Mills, Wythe County.—Several pits have been made into a bold outcrop of magnesian limestone carrying nodules of resin blende. Several diamond drill holes have also been bored, but without results of commercial importance.

New River Mineral Co., Ivanhoe.—In the progress of its large iron ore operations this company has found some zinc blende of fair grade in limestone, and is now sinking a shaft in hard rock to determine the size and continuity of the zinc-bearing strata.

Fabers Mills, Nelson County.—This mine was worked for lead during the



The above diagram represents the production by smelters and includes zinc made from both foreign and domestic ores. In Belgium a large part of the output is derived from imported ores, and this is true also of the United Kingdom and France. The United States, Germany and Spain are exporters of ore.

Civil War. Recently the old workings have been cleaned out and extended, developing a vein of quartz-carrying blende and galena. Jigs are now being installed preparatory to cleaning up a stock of pay rock.

PRODUCTION OF ZINC IN THE WORLD. (IN METRIC TONS.)

Year.	Austria. (a)	Belgium. (b)	France. (b)	Germany (b)	Russia. (d)	Spain. (b)	United Kingdom. (c)		United States. (e)	Totals.
							Native Ores.	Foreign Ores.		
1894.....	6,810	97,041	23,387	143,577	5,014	5,100	8,260	24,318	67,135	380,642
1895.....	6,456	107,664	24,200	150,286	5,029	5,845	6,760	23,207	74,245	403,692
1896.....	6,888	113,361	45,585	153,062	6,257	6,133	7,224	18,054	70,432	417,016
1897.....	6,236	116,067	38,067	150,739	5,868	6,244	7,162	16,648	91,070	438,096
1898.....	7,302	119,067	37,155	154,867	5,664	6,031	7,162	21,225	103,514	461,987
1899.....	7,305	(f)	26,982	153,155	6,325	6,500	32,222		117,644	490,139

(a) The statistics for Austria are taken from the official reports of the Mines Department, except for 1899, for which the figure reported by Henry R. Merton & Co. has been used. It is to be noted that Merton's figures for Austria are generally higher than the official figures notwithstanding that the Austrian statisticians include the make of zinc gray with the spelter. Thus Merton reported 9,382 metric tons as the Austrian production in 1897, while the official statisticians gave only 6,236. The discrepancy appears to be due to the Galician works, which give higher figures to Messrs. Merton than to the government, those for the three remaining works in Austria agreeing closely.

(b) Official statistics, except for 1899.

(c) The statistics for the United Kingdom are arrived at by deducting the zinc produced from domestic ores, as reported in the official Blue Books, from the total output of the smelting works as stated in the reports of Messrs. Henry R. Merton & Co.

(d) From official reports, except the figures for 1898 and 1899, which are reported by Henry R. Merton & Co.

(e) Statistics compiled from direct returns by the producers to THE MINERAL INDUSTRY.

(f) According to Henry R. Merton & Co., Belgium, Holland and the Rhine district of Germany in 1899 produced 192,994 metric tons of spelter, against 191,886 in 1898; and Upper Silesia produced 100,167 (99,382).

PRODUCTION OF ZINC ORE IN EUROPE. (IN METRIC TONS.)

Year.	Algeria	Austria	Belgium.	France.	Germany.	Greece. (b)	Italy.	Norway. (a)	Russia.	Spain.	Sweden	United Kingdom
1893.....	24,400	30,531	11,310	74,400	787,911	22,589	132,767	(c)	.....	62,616	46,623	23,880
1894.....	29,708	28,491	11,585	80,065	728,616	20,830	132,777	200	62,420	58,964	47,029	22,170
1895.....	14,300	25,862	12,230	72,969	706,423	24,031	121,197	(c)	57,213	54,109	31,349	17,758
1896.....	17,587	26,887	11,630	81,346	729,942	22,700	118,171	450	59,680	54,828	44,041	19,629
1897.....	32,269	27,468	10,954	83,044	663,850	30,906	122,214	908	54,524	73,848	56,636	19,587
1898.....	29,80 0	27,395	11,475	85,550	641,706	32,045	132,099	(d)	(d)	99,836	61,627	23,929

(a) Zinc-lead ore. (b) Including blende and calcined calamine. (c) Not reported in the government statistics. (d) Statistics not yet published.

Canada.—In the early part of 1900 about 300 tons of zinc ore, averaging 50% Zn, were shipped from the Zenith mine near Rossport, to the new electrolytic refining works, at Hamilton, Ontario. The new refining works were erected to treat and refine nickel ores and matte and zinc ores.

THE SPELTER MARKET IN 1899.

New York.—The market conditions during the entire year were markedly variable, and the constant and rapid fluctuations rendered business difficult and diminished profits. The brass trade throughout the year has been good and consumed a larger amount of spelter than perhaps in any previous year. The consumption of sheet zinc was rather irregular and suffered from the wide fluctuations in spelter, still it has done very well. The capacity of the plants in operation appears sufficient to meet present trade requirements.

The year commenced with a firm tendency, with prices ruling at 4·85c. at St. Louis and 5c. at New York, but then a very strong demand, especially for galvanizing purposes, started up, which rapidly drained the light stocks on hand, and prices during the month quickly advanced to 5·35c. at St. Louis and

5·65c. at New York. By the end of February prices advanced to about 6·375c. at St. Louis and 6·625c. at New York, at which high figures orders fell off, and a sharp decline set in, which lasted throughout the month of March, during which period prices at one time declined to 5·875c. at St. Louis and 6·125c. at New York. This reaction brought about a better business and the market improved slowly during April, by the end of the month again advancing to 6·50@6·625c. at St. Louis and 6·75@6·875c. at New York. These proved to be about the highest prices of the year and they were firmly upheld until the end of May, when again a marked flatness became noticeable. In June the decline became more pronounced culminating early in July, when 5·35c. at St. Louis and 5·50c. at New York was reached.

At these figures the smelters had to sell at tremendous losses, considering the high values for ores, which kept up remarkably well. The pressure to sell grew less, and this for some little time gave a stimulus to the market, but it was soon observed that the galvanizing trade was no longer as good as early in the year. In the beginning of September, sales were made as low as 5·125c. at St. Louis and 5·375c. at New York. At these figures consumers again bought quite freely. Fluctuations were the order of the day, and it was then that the Miners' Association decided to close down the majority of the mines and hold out for ore prices, which, with the decline of spelter, had necessarily to come down. They agreed not to sell ore below $43 for 60% concentrates. This shutting out of a large proportion of the product necessitated in turn smelters drawing their furnaces, particularly as prices for ore proved much too high to produce metal and sell it at the ruling market values. This fight continued until the middle of November, and it was hoped that during this time the spelter market would advance, but this did not turn out to be so, and we find that early in November the ruling values were 4·625@4·75c. at St. Louis and 4·875@5c. at New York. The main reason for this very large drop was the enormous falling off in the demand for galvanizing purposes, which culminated in December when prices dropped to 4·15@4·25c. at St. Louis and 4·375@4·5c. at New York; these being the low water mark of the year. In spite of the heavy reduction in the output, stocks accumulated, but the low figures admitted of some heavy sales to Europe, which materially relieved the situation, as the surplus on the market suddenly disappeared. The lower prices also attracted the attention of consumers and a good business took place, with prices rapidly advancing to 4·60c. at St Louis and 4·85c. at New York. The unsettled condition of the stock markets and the scarcity of money in the principal centers of the United States caused a drop in the prices, closing at the end of the year at 4·375c. at St. Louis and 4·625c. at New York.

AVERAGE MONTHLY PRICES OF SPELTER IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1895	3·28	3·20	3·23	3·30	3·50	3·65	3·75	4·15	4·30	4·10	3·55	3·49	3·63
1896	3·75	4·03	4·20	4·09	3·98	4·10	3·97	3·76	3·60	3·72	3·99	4·14	3·94
1897	3·91	4·02	4·12	4·13	4·21	4·21	4·32	4·26	4·18	4·17	4·03	3·89	4·12
1898	3·96	4·04	4·25	4·26	4·27	4·77	4·66	4·58	4·67	4·98	5·29	5·10	4·57
1899	5·34	6·28	6·31	6·67	6·88	5·98	5·82	5·65	5·50	5·32	4·64	4·66	5·75

London.—This was a rapidly rising market in the beginning of the year, when consumers bought actively in anticipation of higher values. On the Continent especially there was a large demand for spelter and sheet zinc, and as European supplies were insufficient the American market was strong and advancing and 50s. per ton was gained during January. Ordinary brands rose from £24 17s. 6d. to £26 17s. 6d. and specials from £24 10s. to £27. Consumption in America was increasing to such an extent that users there made large purchases, some for very far forward delivery and as a similar tendency prevailed on this side, prices rose steadily, and in February £28 2s. 6d. was paid for ordinaries. In Silesia many of the producers were sold out till the end of the year and Belgian supplies were falling off owing to labor difficulties.

At the higher values, in March, dealers pressed sales, causing a decline of about 20s., but the second state of trade proved stronger than their operations, and with a renewed demand at the end of the month prices rose to £27 15s. A good business in hard spelter was also done at fair values. The market continued to strengthen in April, partly in sympathy with the rising American market, and partly influenced by a good demand for galvanized iron. Supplies continued scarce and prices rose to £28 12s. 6d., and was maintained in spite of large offerings in May. The American market at this time was quite above our parity, precluding export business. In June prices were driven down to £27 5s., but this level attracted more consumers' orders and the tendency improved. The demand then fell off entirely, and on holders becoming anxious sellers, prices suffered considerably, £25 being accepted for ordinaries. At this level further realizations took place in July by disturbed holders, but on the other hand the lower figures again attracted consumers, who were badly provided, and prices rose to £25 18s. 9d. on strong buying.

The tone was very dull in August. Second-hand lots were constantly pressed for sale and several large Continental firms being also anxious sellers, drove the values down to as low as £22. With dull trade on the one hand, and strong American reports and some speculative buying on the other, prices fluctuated back and forward in September and closed at £22 10s. Specials were quoted at £23 to £23 5s. In spite of the healthy condition of the galvanized sheet trade, the market was again depressed by free sellers, who accepted down to £21 10s., and in November prices continued to drop, although the activity in the galvanized-iron trade during this month was almost unprecedented and consumers were buying more freely. This decline was owing, no doubt, to the weaker position of the American market and the probability of export trade being resumed with that country, as well as a general lack of confidence which made buyers very cautious. Ordinaries fell as low as £19 17s. 6d. to £20, while £20 5s. to £20 7s. 6d. were quoted for specials.

In December, however, there was at last a turn for the better, both here and in America, where the supplies available for export were considerably reduced. Large purchases were made on the Continent at advancing prices and American sellers withdrew from the market. The London market lagged somewhat behind the others, but the tendency was distinctly firmer at £20 5s. to £20 7s. 6d. for good ordinaries and £20 10s. to £20 12s. 6d. for specials.

Breslau.—There was a marked advance in the production of zinc during 1899, and the price was increased to such an extent as to approach the highest figures of 1873. Good orders for consumption supported by active speculation carried the January price of 24 marks per 50 kg. f. o. b. Breslau up to 29 marks in May. A reaction naturally folowed this excessive strain and a diminished consumption resulted, which was noticed particularly in sheet zinc—consumers employing cheaper substitutes for that metal. With the consequent hesitancy in the demand, prices then took a downward tendency and reached the lowest point, 19·50 marks in December; as soon as this question was reached, a considerable demand for consumption and speculation arose, the price rallied to some extent and closed the year with 20·5 marks.

The following prices were quoted per 50 kg. f. o. b. Breslau: January, 24, 24·50, 25·50, 26 marks; February, 26·50, 27·50, 28 marks; March, 27·25, 27, 27·50, 28 marks; April, 28·25, 28·50 marks; May, 28·25, 28·50, 29, 28·25, marks; June, 26·50 to 25·75, 26 marks; July, 26, 25·75 marks; August, 24, 23·25, 23·75 marks; September, 20·50, 23, 22·75 marks; October, 22·25, 22·75 marks; November, 21·50, 20·50 marks; December, 20·50, 19·50, 20·50 marks. The official average prices as reported quarterly by the Mining Bureau at Breslau are given in the subjoined table:

	1894.	1895.	1896.	1897.	1898.	1899.
First quarter	14·50	12·50	13·50	16·00	16·50	25·00
Second quarter	14·00	13·00	15·50	15·50	17·50	26·00
Third quarter	14·00	13·50	15·50	15·50	19·00	22·50
Fourth quarter	13·00	13·50	11·00	16·50	22·00	19·50
Average	13·87	13·12	15·12	15·87	18·75	23·25

The production in Silesia during 1899 was on an equality with that of 1898 and, while there was a great increase in the production in the United States, the home consumption and price there were also increased to such an extent that any large shipments to Europe were prevented. Attempts were made during the year to form a consolidation that would eventually include all the European producers. The director general of the Vieille Montagne undertook during the summer, to collect written statements from the various works setting forth their inclinations; this method of correspondence was selected so that ultimately the arrangement could be consummated without delay. While the general tendency is toward consolidation, the matter is still in abeyance.

The business in sheet zinc in Germany was generally satisfactory, although the high prices diminished the consumption and the use of zinc for galvanizing iron sheets was abandoned to a considerable extent. An equality of supply and demand resulting in profit to the manufacturers was only possible by limiting the production. The total production in 1899 of sheet zinc in Germany amounted to 35,000 metric tons against 39,863 tons in 1898. The price at the beginning of the year was 54 marks per 100 kg. This increased to its highest point of 60·50 marks on April 25th, after which it declined, and closed the year at 48 marks, basis Oberhausen. The sales of sheet zinc were managed by the sales bureau of the Schlesischen Actien-Gesellschaft für Bergbau und

Zinkhütten-Betrieb in Berlin. In Austria-Hungary, the sales of sheet zinc from all the zinc rolling mills were conducted by the Actien-Gesellschaft der Oesterreichisch-ungarischen Zink Walzwerke through the main office at Privoz, Moravia.

The amount of zinc ore imported into Germany in the 11 months (January-November) 1899 exceeded the amount exported by 61,536 metric tons. These ores were derived from Austria-Hungary, Sweden, Spain, United States and Australia. Ores were exported to Belgium, Holland and Austria-Hungary.

In the latter months of the year American ores were of great profit. Ores from the Joplin district containing 63% Zn were sold at $33 per ton of 2,000 lb. and Colorado ores of 48% Zn, 0·3% Pb, 10% Fe, 0·4 oz. Ag, were placed on the market at $22·50 per ton of 2,000 lb.

The business in zinc white in 1899 was not as satisfactory as in other branches of the zinc industry, the disturbing factor of increased spelter cost which entailed less profit from filling contracts that had been given for future delivery, were strongly intensified in 1899; furthermore the rise in price, not keeping in ratio with the increased cost of spelter, caused a losing business for part of the year; later, however, this adverse condition was ameliorated. The construction of a new zinc white factory in the Rhine province promises to help the industry.

Zinc Dust.—From the beginning of the year until May, the market preserved a weak tendency and it was not possible to maintain an equal ratio with the increasing price of spelter. This condition was particularly noticeable in June. Increased consumption had been satisfied for a long time, but several industries required larger quantities and caused a shortage in manufactured goods; especially in the last quarter were inquirers active at home and abroad and prices were obtained that were considerably higher than the spelter. Normally the course of exchange is about 95% the official price of spelter. The situation was intensified in October when 99 cases were destroyed at Oppeln. It is scarcely to be supposed that the present condition will remain permanent in 1900. The increased production from Silesia and Rhenish-Westphalia will be utilized and the supply and demand will gradually assume proper proportions.

Sales extended from Germany and Austria-Hungary to Switzerland, Italy, Norway, Great Britain, Japan, India, Africa and the United States.

The use of zinc dust is quite general and it is now replacing zinc shavings in the precipitation of gold from solutions obtained in the cyanide process; about 420 g. are required for each ounce of gold to be precipitated.

Cadmium.—The condition of this industry remained passive throughout the year without particular disturbance. The price depended on quality and quantity and ranged from 500 to 700 marks per 100 kg.

Progress in the Metallurgy of Zinc in the United States in 1899.

By Walter Renton Ingalls.

The zinc smelting industry in the United States in 1899 suffered from disturbed industrial conditions. Early in the year the price of both zinc ore and

spelter rose to abnormally high marks, and coincidently the mine operators of Missouri and Kansas organized an association to regulate the price of ore and maintain it at a high level. The results were at first equal to their expectations. To what extent, however, the creation of the higher prices for ore were due to their efforts, and to what extent they were due to the natural conditions of supply and demand is uncertain and a matter of dispute. It is certain, however, that when the natural conditions became unfavorable later in the year the association of miners was quite unable to maintain the price of ore against the natural law of supply and demand, and values fell to normal figures. In its contest with the smelters, who are themselves governed by the conditions of the metal market, the Missouri & Kansas Zinc Miners' Association failed largely in its object; nevertheless, it succeeded in effecting certain important improvements, especially the principle of an assay basis for the sale of ore, and by unifying the interests of the ore producers has given them a much stronger position in dealing with the ore buyers. The association started out with the idea of regulating the price of ore by the market price of the metal, and if it had adhered to that policy, it might have been entirely successful in its object. When, however, the price of the metal began to trend downward, which tendency appeared first in the early summer, the association practically abandoned its original contention and refused to reduce the price of ore accordingly. Efforts were made to control prices artificially, both of ore and spelter, by restricting the production of ore, but it was not taken into account that a high price for metal limits consumption and the price of spelter receded steadily notwithstanding the efforts of the miners to maintain it, in which they were aided by the smelters who were loaded up with spelter that had cost them dearly and were naturally anxious to protect themselves as far as possible against the further losses to be suffered on a falling market. In the autumn the smelters found themselves in the position of being called upon by the miners to pay more for the zinc in ore than they could realize for it as spelter, allowing for freights and the cost of smelting. Of course the least favorably situated works were the most seriously affected, but there was a period when probably not even the best works in the West could be operated profitably. For a time the smelters continued to run at a loss, adopting that policy because the cost of closing the campaign of a distillation furnace is itself expensive and the cost of closing a works completely, with the consequent disorganization of its operating force, and the loss to be incurred perhaps in suspending existing contracts, is proportionately more expensive; wherefore works were kept running in the expectation of a speedy improvement in conditions. As a matter of fact there were not very many which were closed completely, but there were few of those in the West which during the autumn were using as much as 50% of their smelting capacity. When for these reasons the smelters ceased to be active buyers of ore and the miners began to have difficulty in disposing of their product, the position of the latter soon became untenable, and since the end of October the price of ore has been made for the most part in the old way, that is to say, the demand of the smelters to meet their requirements, and the ability of the miners to supply it. Prices have ruled, however, at a higher level than formerly and

in so far the Missouri & Kansas Zinc Miners' Association has served a purpose. It should be said, however, that the higher level of prices is probably due quite as much to the increased competition among the smelters, the newer concerns at Iola and Cherryvale, which enjoy costless, or almost costless, natural gas for fuel, being able to smelt more cheaply than the older works which use coal for heating the furnaces, wherefore, the latter have difficulty in meeting the competition of the newer works, although the recovery of metal is in general lower at the gas smelteries than at the others.

The present capacity for smelting zinc ore in the Western works is a good deal in excess of the productive capacity of the mines or the demand for spelter for consumption. It should be considered, however, that a large part of the smelting capacity is of an antiquated type (some of the works which have been lately in operation, and possibly are in operation at the present time, have been in use since zinc smelting was first undertaken in the West, that is to say about 25 years ago, with practically no improvement of plant), on which account it should be looked upon as out-of-date capacity, which should be discarded in favor of more economical modern appliances. The producers of the ore have a natural right to share in the greater profit that can be obtained by better metallurgical practice, and it would be unfair to expect them to be contented with the results of 10 years ago; much less with those of 20 years ago. This is not to be interpreted as saying that there has been no improvement in the American practice in zinc smelting in 10 or 20 years. As a matter of fact, certain of the larger works in the West have been very enterprising in the introduction of improvements and their practice is highly developed in certain directions, while in the smaller works, although there have been few radical innovations, the smelting is carried on with greater care and a better understanding than formerly, the business is more closely looked to, and better results are obtained both in cost of smelting and percentage of metal recovered than used to be the case. Notwithstanding those improved features, however, it is a conservative statement to say that in general the practice of zinc smelting in the West is a good deal more backward than in Belgium and Germany, and for these reasons the building of new smelteries ought not to be regarded as merely adding to the already existing surplus of smelting capacity with the probable result of a competition ruinous to all; but on the contrary should be looked upon as the replacement of old, worn-out plants by newer ones of a better type.

I have already spoken of the change in the ore market at Joplin, whereby ore is now bought and sold upon an assay basis. The value of the ore has of course been governed always by its tenor in zinc and undesirable impurities, but formerly those factors were determined only by the judgment of the ore buyers, whose perceptions, it must be granted, were very keenly developed. However, no one will pretend that a proper sample and assay of a lot of ore can be made by a mere visual inspection, and it was natural that the smelters buying the ore should take care to have the chances of error in their favor. At present all of the large smelters have chemical laboratories, and several independent laboratories have been established in Joplin, and assays are generally made to determine the value of ore bought and sold. The methods are still very crude, inas-

much as the sampling of the ore is done imperfectly, but it is a step in the right direction. No change has yet been made in the manner of determining weights, and transactions are still effected on the basis of the wet weight of the concentrated ore with possibly an arbitrary deduction on the part of the smelter by special stipulation in the case of ores which are thought likely to contain more than the 2 or 3% water which the average coarse concentrates hold. This method is quite unscientific and should be replaced by a dry basis of settlement, which is the only logical one. In the assay of zinc ores in the West the ferrocyanide method of determining zinc is commonly employed, being conducted in practically the same manner as in the laboratories of the silver-lead smelting works of Colorado and the far West. This is a very satisfactory method of assay, and is far more expeditious than the sodium sulphide method which is commonly used abroad.

The high price for zinc ore that prevailed in the Joplin district during the first three quarters of 1899 led the smelters to look for supplies of cheaper ore elsewhere. As a result of such inquiries a considerable amount of ore was obtained from other States, especially from Colorado. Zinc ore was shipped from at least three points in Colorado, namely, Creede, Leadville and Montezuma. These shipments went to works in Kansas, in Illinois and in Indiana, besides a considerable quantity which was exported to Belgium via Galveston. In general the Colorado zinc ore is both ferruginous and lead bearing, and cannot be freed cleanly from either of the undesirable elements, on which account it is an inferior ore at the best. However, some of the Colorado ore has been dressed so cleanly as to be of very good grade and a desirable product to buy, in view of the lower cost at which it can be had. There was a considerable export shipment of ore from New Jersey. The willemite from that region is greatly liked for its freedom from lead and the ductility of the resulting metal. It is now partly used abroad to improve the quality of common spelter. A small amount of zinc ore is said to have been shipped to Waukegan, Ill., from New Mexico in 1899. There was a good deal of talk concerning the zinc deposits of Arkansas, as to which there are conflicting reports. They are unfavorably situated with respect to railways, but as the deposits are developed the railways will undoubtedly furnish better shipping facilities, if the outlook is promising. The old mines at Mossy Creek, Straight Creek, and Lead Mine Bend in Tennessee, were reopened in 1899, and a small amount of ore was shipped from them to smelteries in Indiana.

Under the conditions existing in 1899, the building of new smelteries or the extension of the capacity of works already in operation was not to be expected. Several new works which were begun in the previous year were put in operation. These comprise a plant of 1,200 retorts built by the Prime Western Spelter Co. in the Iola district and plants of the same size built in the same district by George E. Nicholson and the Cherokee-Lanyon Spelter Co.; also the large plant of the Edgar Zinc Co. at Cherryvale. The smelteries of the Robert Lanyon's Sons Spelter Co. and W. & J. Lanyon at Iola, LaHarpe and Pittsburg, Kan., were taken over by the Lanyon Zinc Co., an Eastern corporation. The Fairmount Zinc Co. built a small works at Fairmount, Ind., but they were destroyed by fire during the summer and will probably not be rebuilt. Owing to the approaching

exhaustion of the natural gas supply in Indiana the conditions for zinc smelting in that district, where the works are small ones, have become rather unfavorable.

With respect to the American practice in zinc smelting, the developments in recent years which have been of the greatest interest and importance are the adoption of mechanical furnaces for roasting the blende, and the establishment of smelteries at Iola and Cherryvale, Kan., where natural gas is available as fuel. Great economies in the cost of smelting have been effected by those innovations. There is no particular type of mechanical roasting furnace which has found general favor, the various smelters which have adopted them having taken the course of securing territorial rights to some type of furnace, which precludes a rival smelter from the use of that furnace, except through payment of a royalty to his competitor, over and above what must be paid to the manufacturer and inventor. The Lanyon Zinc Co., at Iola and LaHarpe, Kan., uses the Ropp furnace; the Cherokee-Lanyon Spelter Co., at its Iola works, uses the Zellweger furnace; the Empire Zinc Co., of Joplin, has both the Pearce and Wethey furnaces, while the Brown horseshoe furnace is used by the Collinsville Zinc Co. at Collinsville, Ill., and by the Edgar Zinc Co. at Carondelet, Mo., and Cherryvale, Kan. Owing to the circumstances mentioned there are no reliable data as to the efficiency of these various furnaces in blende roasting. However, they all do satisfactory work, apparently, and have so well established the success of the principle of mechanical blende roasting that no up-to-date smelter would revert to the old hand-raked furnaces. By the use of natural gas for heating the furnaces, for roasting and distillation and all other purposes, the smelter is relieved from the necessity of buying any coal except for reduction material, while there is also a large saving in labor on account of the smaller quantity of coal to be handled, the disappearance of stokers, and to a large extent of ashes. On the other hand the natural gas smelter is at the expense of drilling wells for his supply, piping it to his works and its maintenance, besides certain other disadvantages which are not yet quite clear. Indeed, the use of natural gas in zinc smelting in Kansas is still so new that no fixed practice can yet be considered to have been established, which accounts for the divergence in opinion among various smelters as to the actual economy which is realized in smelting at Iola as compared with Pittsburg, Kan., where coal is the fuel. It is true that natural gas has been used a longer time in Indiana, but the results in the latter district do not appear to have been well analyzed.

The distillation furnace employed at Iola is generally a long double furnace with 300 retorts per side, arranged in 5 rows of 60 each. Originally the gas was admitted at one end and the products of combustion were allowed to escape at the opposite end, air being introduced at intervals along the front of the furnace. More recently these furnaces have been designed for the introduction of gas at intervals in their length, as well as at the end, and the air for combustion is forced in by a blower. At Cherryvale and generally in Indiana the natural gas fired furnaces have 100 retorts per side, arranged in 5 rows of 20 each. Among the works in Kansas and Missouri, which employ coal as fuel, the Belgian furnace, with 112 retorts per side, arranged in 7 rows of 16 each, is by all means the most common.

The technical literature of the year contained few references of interest concerning the metallurgy of zinc; even the number of new processes proposed for the treatment of mixed sulphide ores was fewer than ordinarily, and there was no one of more than passing interest. The mixed sulphide ore problem has now been advanced by long strides toward a successful outcome, through the satisfactory results in the treatment of such ores that have been obtained by the Fry process and the Wetherill magnetic separating machines. By the aid of both processes New South Wales has already become a large producer of zinc ore, which although not of the best quality, nevertheless adds to the resources of European smelters, who also have a comparatively new and highly important supply in the concentrated willemite which is produced by the Wetherill machine from the mixed ore of New Jersey, once in itself a knotty metallurgical problem.

In summing up the literature of the year it is necessary to refer to the results of an extensive examination by E. Jensch concerning the cadmium contents of zinc ores.* He came to the conclusion that the percentage of cadmium in blende is much less than the statements in most text books imply. Ores from 20 different mines showed a range between 0·004 and 0·360%, giving a general average of 0·102%. Blende from the upper Harz, reputed rich in cadmium, contained a maximum of 0·1%. Blende from Styria, Carinthia and Carniola showed from 0·003 to 0·140%. The highest assay was obtained from a Finnish ore, which contained 0·46%. Owing to its great volatility a large part of the cadmium existing in zinc ore is lost in roasting; for example, a Silesian ore which assayed 0·11% before roasting, assayed only 0·042% after roasting. A part of the loss is recovered in the flue dust, which contains from 0·22 to 2·02% Cd, 40% of which exists as a sulphate. The occurrence of cadmium in coal has also been noted.

Robert C. Schuepphaus described in the *Journal of the Society of Chemical Industry*, November 30, 1899, p. 987, a series of experiments as to the behavior of zinc oxide at high temperature under pressure. An electric furnace with a cast-iron melting pot, lined with fire clay, of sufficient strength to withstand a pressure of 10 atmospheres, was used. The experiments, which were conducted on a small scale, were directed toward the smelting of zinc ore by ordinary methods in place of distillation in small retorts. The difficulty of applying ordinary smelting processes to zinc ore, or at least one of the difficulties, is chiefly caused by the close proximity of the reduction temperature of zinc oxide and the boiling point of metallic zinc. Dr. Emil E. Lungwitz, in his English Patent No. 9,072, of May 7, 1895, showed that the temperature of ebullition is a function of the pressure, and pointed out a method of utilizing that principle. If the pressure be raised sufficiently within a closed apparatus and zinc oxide be reduced therein, metallic zinc must be obtained in liquid form. At the temperature of 1,034°C., which corresponds to the temperature of the slag tapped from a lead smelting furnace, the tension of zinc vapor (according to Barus) is less than two atmospheres; consequently the technical difficulties of carrying out such a process in the opinion of Dr. Schuepphaus, are not formidable. According to his experiments the reduction of zinc oxide by carbon begins at a

* *Berg- und Hüttenmännische Zeitung*, 1899, p. 14.

temperature of 910°C. In heating charges of zinc oxide and carbon up to 1,150°C. and maintaining a pressure in excess of the calculated tension of zinc vapor, ingots of zinc were obtained at the bottom of the crucible, although its temperature was far beyond 930°C., the boiling point of zinc under atmospheric pressure. This method of smelting is suggested, among other things, for the production of spelter from zinc dust or blue powder, which usually contains 8 to 10% zinc oxide. The commercial practicability of the process is not discussed. It is referred to here especially in view of the scientific interest of the experiments.

Dr. Schuepphaus also found that upon heating zinc oxide and carbon under pressure, on many occasions large quantities of a powder of canary-yellow color were obtained. Analysis showed the presence of more zinc than is contained in the oxide. Pursuing the idea that this yellow compound might be a low oxide of zinc the following experiments were made, each of which yielded a body permanently yellow: (1) Zinc oxide was heated in an atmosphere of pure nitrogen; (2) zinc oxide was heated in vacuo; (3) zinc oxide, mixed with zinc dust, was heated under pressure. These experiments proved it to be highly probable that zinc oxide on heating loses part of its oxygen and that the new compound is stable if cold in the absence of oxygen; indeed it has been kept in the open air for a period of six weeks, and remained unchanged.

There was nothing of great importance published with respect to analytical methods for the determination of zinc. A. C. Langmuir, in the *Journal of the American Chemical Society* for February, 1899, described a new gravimetric method with which he obtained good results. For the determination of zinc in spelter he considers that in impure samples it is often more convenient to determine the zinc directly, than by difference, and equally accurate. One-half gram of the sample is dissolved in nitric acid and lead is separated electrolytically in a solution containing 10% nitric acid. A platinum cylinder is used to collect the lead peroxide. The solution is then evaporated to dryness in a weighed dish, ignited and weighed. The precipitate is dissolved with a little hydrochloric acid, reduced with two or three grams of granulated zinc, and after the addition of sulphuric acid to complete the reaction is diluted with ice water to one liter and titrated with permanganate to show the amount of iron present, for which correction is made in the previous weight. If cadmium, arsenic or other impurities are present in more than traces, this method would be inapplicable unless modified to include treatment by hydrogen sulphide. E. Jensch* finds that in the determination of zinc in aluminous ores the retention of zinc by the precipitate produced by ammonia is due chiefly to the presence of alumina in the ore. Repeated re-solution and re-precipitation overcome the difficulty. Four re-precipitations sufficed to give a precipitate of iron and alumina free from zinc in the worst instances observed.

A matter of interest bearing upon the consumption of zinc is the considerable amount of zinc chloride which is now being used extensively by certain railways in the West for the preservation of wood, especially sleepers. Although there is nothing new in the use of the chloride for this purpose, its application has not

* *Zeitschrift für Angewandte Chemie*, 1899 (20), 465-467.

been general. The treatment of the wood consists in forcing by hydraulic pressure a solution of the salt of about 2·5°B. in strength, into the pores, and then drying. This may be followed by a second injection of tannin or of glue, though the subsequent treatment is of doubtful value, its purpose being to protect the chloride from weathering influences. According to W. W. Curtis, in the *Engineering News* of January 29, 1899, the Atchison, Topeka & Santa Fé Railway Co. has been conducting experiments in this direction on a large scale during the last twelve years with very satisfactory results; other companies in the West have since taken it up. The life of wooden sleepers is said to be prolonged by this treatment to two or three times its usual length. The amount of zinc chloride consumed for this purpose in the United States is already rather large.

The zinc smelting works of the United States were visited in 1899 by M. Gaston de Sinçay, director-general of the Société Anonyme de la Vieille Montagne of Belgium. M. de Sinçay appreciated greatly the frankness with which American smelters disclosed and discussed their metallurgical methods, and volunteered information concerning the practice of his own company. This he has done in a communication addressed to the editor of THE MINERAL INDUSTRY, replying to a series of leading questions which I have had the honor to translate and present as follows:

NOTES ON ZINC SMELTING BY THE VIEILLE MONTAGNE CO.

The metallurgy of zinc in Europe dates from the beginning of the nineteenth century, and originated almost simultaneously in Belgium and Silesia. The first furnaces erected at Liége, Belgium, were designed by Daniel Dony. The minerals treated in them were rich and easily reducible calamines from the mine of Vieille Montagne (Altenberg). The furnaces were short and high, comprising 46 small cylindrical retorts arranged in eight horizontal rows.* They were heated by long flame coal from the Liége Basin. The distillation required 12 hours in the lower rows of retorts, and 24 hours in the upper. Metal was drawn generally four times during the period of reduction.

In Silesia, where the calamines were of very low grade, and more difficultly reducible than those of the Vieille Montagne, the furnaces were designed very large, but of little height. They contained only one row of retorts, in which the charge remained 48 hours, but after the first 24 hours, when the reduction was partially completed, an additional charge of mineral was introduced.

In the Silesian furnaces the heating was effected by the down-draught system, while in the Belgium furnace the retorts were directly over the fire and the draught was upward. The two systems of furnaces were maintained for a long time with their original differences, but later when the development of the industry rendered necessary the treatment of ore of different source and grade, the smelters were led to make great modifications in the design of their furnaces. One result of such adaptations is the Belgian-Silesian, or Rhenish, furnace, which

*In the old furnaces the skewback of the roof arch was supported by an arch from end to end of the furnace, so the uppermost rows of retorts could not receive the same number as the lower. It is for that reason that the total, 46, mentioned above, is not a multiple of 8.—W. R. I.

resembles the Silesian by its great width and down-draught method of heating, and the Belgian by the superposition of several rows of muffles. The first Rhenish furnaces had only two rows of muffles, but in Belgium, where the smelters had the tradition of the *liegeois* furnace a third row was introduced. Then, to approach still more the Liége furnace the muffles of the upper two rows were replaced by retorts at one of the Vieille Montagne works, and finally all the muffles were replaced by retorts of various capacities.* At the present time the zinc furnaces are generally of the combination type, more or less profoundly modified, and the reduction is effected in 24 hours.

In the oldest works of the Vieille Montagne Co., where the old type of Liége furnaces is best conserved, the zinc distillation furnaces have 200 retorts arranged in five horizontal rows,† but as the system of direct heating tends to disappear, one side is heated directly from the grate and the other by down-draught (*flamme renversée*). In other works of that company the furnaces partake more of the Silesian system, and are characterized by down-draught heating and elliptical retorts of various size. The furnaces at Viviez, in the south of France, have 132 retorts in the two sides, while those of three other works have only 108. These different dispositions result not only from the type of furnace from which they have been developed, but also from the local conditions and the habits of the workmen, which it has been sought to modify as little as possible.

The mode of burning the coal to heat the furnaces does not differ necessarily with the type of furnace, but is governed rather by the quality and composition of the coal itself. Originally all kinds of furnaces were heated from flat grates. Later gas firing was introduced, and producers of various systems were employed, among which may be mentioned the Ponsart and the Siemens, with recuperation of heat. In France, Westphalia and Rhenish Prussia, where a very fat coal is available, gas firing, with or without heat recuperation, is now generally adopted. In Belgium, on the other hand, where the coal is only semi-fat, direct grate-firing is still generally maintained. Such Belgium works as have adopted gas firing are obliged to import German coal. The Vieille Montagne Co. has always sought to use the coal most easily obtained, and consequently in order to avoid recourse to foreign coal retains direct firing for the more part of its furnaces. Nevertheless it is admitted that gas firing is generally more advantageous than direct firing, when suitable coal is available, and in France where the coal burned contains as much as 38% volatile matter, the Vieille Montagne Co. has obtained an economy of about 10 to 15% as compared with direct firing.

The old form of cylindrical retorts are no longer used in any of the Vieille Montagne furnaces, having been replaced by those of an elliptical cross section, which permits the charging of a larger quantity of material, while giving the heat the same facility of penetration. It is sought to diminish the thickness of the walls of the retort as much as possible, since the thinner they are the more

* The vessels employed in the distillation in both the Silesian and Belgian furnace are retorts, properly speaking, but the Silesian are called muffles from their shape, and in the absence of a more specific word the Belgian "creusets" have been translated simply "retorts."—W. R. I.

† This refers to a double furnace with 100 retorts per side, each side having 5 rows of 20 each; two such furnaces are built in a block, which thus has 400 retorts.—W. R. I.

easily they are traversed by the heat. The thickness that must be given them, however, depends upon the form of the retorts and their dimensions, especially their length, and varies also with the character of the clay from which they are made, and the nature of the ore treated. No precise rule to determine the proper thickness of retort walls can be given therefore, and it must be determined to meet the conditions of each individual works. The internal capacity of the retorts in the Vieille Montagne works varies from 50 to 70 cubic decimeters. The dimensions are kept uniform, as much as possible, in each works, but the quality and richness of the ore charged is varied according to the place the retorts occupy in the furnace, and the amount of heat they receive. Originally large muffles were employed for the treatment of the low grade calamines of Upper Silesia, and small retorts for the rich calamines of Belgium, which was entirely rational, but at the present time, because of the diversity of the ores available, it is possible to compose mixtures which differ but little in grade.

The ores treated by the Vieille Montagne Co. are of extremely diverse composition. They are received not only from most of the countries of Europe, but also from Africa, Asia, America, and even from Oceanica. The smelting works of the company being situated in Belgium, France and Germany the refractory material employed is also of very diverse composition. The nature of the ores to be smelted, and that of the clay for preparing the retorts bear an important relation, which depends upon numerous factors, and requires the constant attention of the smelting superintendents. It is obvious, however, that the prolonged drying needed by the retorts renders difficult any experimental modification in their composition and shape, and involves a long time in obtaining results from any such change. It is sought therefore to maintain a regular mixture of refractory material in each works, in order not to develop unduly its stock of retorts, and the mixtures of ore are compounded to conform thereto so far as possible. About all that can be reduced to a dictum concerning the adjustment of the relation between the ore and refractory material is the rule that the retorts destined for the treatment of ores with a basic gangue should contain a larger proportion of alumina than those intended for ores with an acid gangue, a higher percentage of silica being desired for the latter. If a higher silica content be required in the clay batch it is introduced in the form of quartz, sand or silicious rock, according to the resources of the district in which the works are situated.

In preparing the mixture of ore the idea is to incorporate a large number of different products, of which the injurious constituents of each will neutralize themselves. If in exceptional cases there are ores with a particularly bad gangue they are used in the mixture only in very small proportion, according to the indications of experience. When the industrial conditions require that certain ores be treated separately, as, for example, those of the Vieille Montagne mine at Moresnet, the influence of the gangue is offset by the composition of the batch employed for making the retorts, as indicated above.

In general, ores with a high percentage of iron are not greatly desired by zinc smelters. Nevertheless certain blendes containing after roasting 20% ferric oxide, about 1% manganese oxide, and 6% silica are treated by the Vieille

Montagne Co. without difficulty, and under normal conditions. Those particular blendes can be well desulphurized, however, and after roasting are easily reducible. Low grade calamines assaying 40% ferric oxide have been smelted, exceptionally, with good results in so far as the consumption of retorts is concerned. On the other hand some calamines with a high percentage of ferric oxide, but of a very refractory character, have given very bad results. When the ore is very ferruginous, and at the same time contains 2 to 3% sulphur, it may be expected that the smelting will give bad results, and a large consumption of retorts. In brief, this question is dependent especially upon the character of the ore and the temperature which is required for its reduction.

With respect to lead in the ore mixture, the Vieille Montagne Co. prepares its charges so that they never contain more than 3 to 5% of that metal, since a greater percentage rapidly destroys the retorts. The charges for furnaces, which are to be driven hard (*a chaude allure*) should not contain more than 3% Pb; but in those which are to be smelted at a lower temperature, the tenor in lead may rise to 4%, and even to 5%. There is only one works in Belgium which makes a practice of treating ores high in lead, namely, that at Sclaigneaux. That works has furnaces of the modified Liége type, and probably runs them at the lowest possible temperature, sacrificing something in the recovery of zinc, but gaining in a larger proportion of lead, which remains non-scorified in the residues in the retorts. The residues are subsequently enriched in lead by washing. With ore assaying 6% Pb it may be estimated that 12% of the quantity present will be distilled over with the zinc, and the residues after washing, may be raised to a tenor of about 40%.

The loss of zinc in distillation occurs chiefly in the residue remaining in the retort, by volatilization from the condensers, *i.e.*, the vapor which they fail to condense, and in the re-treatment of blue powder. Besides these there may be a loss by volatilization through the walls of the retorts. In general the volatilization losses form only the smaller part of the total, but under certain circumstances they may become much more important. Such increased losses may happen, especially when the retorts because of their bad composition or bad fabrication, are so porous as to permit zinc vapor to penetrate their walls, or when they remain in the furnace too long a time, or when for any reason they suffer a great deterioration.

The employment of iron prolongs on the condensers is general in all the works of the Vieille Montagne Co. In these are collected powders, which are partly metal, in a very finely divided form, and partly zinc oxide, which are designated "gray oxide," (English, "blue powder," "zinc dust"). This product is either resmelted or sold after sifting for use in the manufacture of pigment. The quality of powder that is collected varies greatly, according to the running of the furnace, and the nature of the ore smelted. However, an additional saving of about 2% may be estimated in general by the use of prolongs.

The Galena-Joplin Lead and Zinc District.

Historical and Geological.—(By Erasmus Haworth.)—It is difficult to determine the exact date of the discovery of lead ore or zinc ore in the Galena-Joplin district. According to Winslow* lead ore was discovered two miles east of Joplin in 1848 by William Tingle, and within the limits of Joplin in 1849 by Daniel Campbell. About two years later mining operations began in Center Creek, at the village of Minersville, now Oronogo. Mining, however, was prosecuted with little vigor, so that by 1854 only 200 tons of galena had been produced in this locality. But during the next few years the activity was greater, and the yield of lead ore was correspondingly increased.

According to Broadhead,† mining operations began in Newton County in 1850, and developed much more rapidly than in Jasper County, with such mines opened during the next few years as the Prairie Diggings' Brook Hollow, Nobleton, and Cedar Creek, not far from the present site of Granby, although as late as 1850 Swallow‡ wrote that there was not a cabin at Granby, and that only one shaft had been sunk. Up to 1854 the total production of Jasper and Newton counties had reached, according to Swallow's estimate, about 862 tons of lead, the zinc ore being neglected. By 1857 he estimated that 4,000 tons of lead had been produced, while from February to September of that year Granby smelting furnace of Blow & Kennett had smelted about 800 tons. By the close of 1860 Swallow estimated that 300 shafts had been put down in and about Granby. Previous to the establishment of the lead smelter at Granby the ore was principally taken by wagon to Fort Smith and carried by boat down the Arkansas River for shipment to New Orleans, New York and Boston. Some of the mines about Granby were unusually productive. Broadhead§ cites the following facts: Holman, Hersey and Gatzel and others obtain from two shafts 3,500,000 lb. of galena in three months, which at the current market price was worth about $73,500. The old Fraser shaft yielded an average of 100,000 lb. of ore per month.

With the beginning of the war in 1861 mining operations were almost entirely suspended over the entire district, but began anew with greatly increased vigor in 1865. At this date Mr. Henry T. Blow obtained a lease upon the Granby mines, and caused rapid developments in both mining and smelting operations. A new impetus was likewise given to mining enterprises over the whole of southwestern Missouri and northern Arkansas, and a lead smelter was established in the early seventies as far away as Boone County.

The first discovery of ore at Joplin, as above stated, was in 1848 or 1849. But little attention was given to mining in this region, however, until 1870. In August of that year two prospectors, J. B. Sargant and E. R. Moffett, discovered large quantities of lead ore within the present city limits. Soon after, in 1871, a miner by the name of Moon found a very rich deposit on East Joplin Hill, a locality which for years thereafter was known as Moon Range or Moon Diggings. Early in 1870 not a single house existed in the present town of Joplin, but within

* *Geological Survey of Missouri*, 1894, Vol. VI., p. 281. † *Ibid.* 1873-74, p. 488.

‡ *Geological Report along the line of the Southwest branch of the Pacific Railroad in the State of Missouri*, St. Louis, 1857. § *Ibid*, p. 489.

two years a town of 5,000 inhabitants was there. These rich discoveries resulted in the establishment of a lead smelter at Joplin in 1871 by Moffett and Sargant.

Few mining towns of America have grown more rapidly, or attracted more attention than Joplin during the first few years of its existence.

The Joplin discoveries stimulated prospecting in all the surrounding country, more particularly to the N.W., west and S.W. Frequently discoveries were made at many points along Center Creek and Turkey Creek, some resulting in extensive developments, while others were soon forgotten.

About four miles north of Joplin, on the west side of a little tributary flowing northward into Center Creek, ore was discovered in 1875 on the farm of Mr. Webb, which resulted in the development of the mining town of Webb City. Shortly after ore was discovered along the east side of the same stream on the farm of a Mr. Carter, and resulted in the establishment in 1876 of the mining town of Carterville.

In April, 1876, lead ore was discovered in Short Creek Valley on the farm of Jesse Harper about 1·5 miles east of the Missouri-Kansas line. Intense excitement followed this discovery, and the Bonanza mining camp was established. In April, 1877, the same prospector discovered a large body of lead ore about a mile further east, which resulted in the development of the mines at Galena.

The excitement and rapid development at Joplin in 1871 was now perhaps more than doubled at Galena. It was estimated that more than 5,000 people were on the grounds within three months after the discovery. Mining lots were laid out, leases obtained, and the prospectors were unusually fortunate in opening up large bodies of lead ore, not only in the valley land immediately adjacent to the discovery, but also to the S.E. along the uplands for more than a mile. Year after year the productive area expanded as good ore bodies were found in every direction from the original point of discovery. At the present time the whole Short Creek Valley, from Central City, four or more miles east in Missouri, down to and including the whole Bonanza area is almost one continuous series of successful mines. Also both north and south from Short Creek, to a distance of four or more miles, are here and there, rich and productive mines in operation.

In 1886 the mines at Aurora, in Lawrence County, Mo., were opened, followed by others along the line of the "Frisco" railroad to the West, until at the present time numerous mines are in operation between Aurora and Webb City.

The Galena-Joplin district now comprises an area extending from the vicinity of Carthage, on the N.W. to about 25 miles S.W., by way of Webb City and Carterville, Joplin, Belleville, and Galena, and some distance into the Indian Territory on the east side of Spring River, south of the Kansas State line. In the opposite direction the territory reaches from the line mentioned across into Newton County, and as far east as Pierce City and Aurora, in southern Lawrence County. Throughout this whole area of nearly 1,000 square miles, both lead and zinc ores have been found in large quantities, and mining camps have sprung up and flourished at many different places.

The earliest mining operations in the Galena-Joplin district were confined to a search for lead ore, as no market for zinc ore existed at that time. The first western zinc smelter was established in 1860 at La Salle, Ill., by Matthiesson

& Hegeler. In 1867, a zinc smelter was established at Potosi, in Washington County, Mo., by G. Hesselmeyer, to smelt the zinc ores produced in that part of the State. In 1869 a zinc smelter was established at Carondelet, South St. Louis, and obtained its ores very largely from St. Francis County, and adjacent territory. In a short time other smelters were established at the same place. The first systematic search for zinc ore was made at Granby in 1871, by an agent of the Missouri Zinc Co., and the first shipment of zinc ore from Joplin was made to La Salle in 1871, at a price of about $3 a ton. In a short time, however, the price advanced to $15 a ton. The first zinc smelter on Kansas soil was established in 1873 at Weir City, and in 1878 the Lanyons erected the first zinc smelter at Pittsburg, Kan. In 1881 smelters were built at Rich Hill, Mo., and Joplin, since when their number has increased with great rapidity until at present there are more than 12,000 retorts in operation in southeastern Kansas, which consume the greater part of the Joplin district zinc ores. As early as 1875 it was estimated that Missouri alone produced 23,500 tons of zinc ore, while in 1880 the total value of the annual production of zinc ore was considerably ahead of that of the lead ore for the entire district.

General Geology of the District.—The ores of the Galena-Joplin district are all contained in the Sub-Carboniferous rocks unless, perchance, it should be found that the deeper seated ones are in the underlying formations. The strata are inclined westward from 15 to 20 ft. to the mile, and therefore outcrop to the east along the western slope of the Ozark uplift.

The Sub-Carboniferous rocks constitute essentially a limestone formation, and are a part of the great Mississippi valley Sub-Carboniferous area, one of the most remarkable in character and extent in the world. The individual layers are frequently very heavy, reaching from 5 to 10 ft. in thickness, and the bedding plane spaces carry a minimum amount of earthy and shaly matter, so that the whole mass of limestone is unusually free from shale partings or other similar material. Sandstone is almost entirely unknown within their limits.

Here and there throughout the area an unusually heavy deposit of flint rock seems to be irregularly interbedded with the limestone. In some places it is from 200 to 300 ft. thick, and has a lateral extent of from 2 to 5 miles with scarcely any intervening limestone. This is particularly the case about Galena, where the largest flint bodies occur. In the opposite extreme thin layers of flint and limestone, no more than half an inch in thickness alternate with each other. Between these two extremes are all possible variations, the flint rock has frequently assumed nodular and rounded concretionary shapes interbedded with the limestone, which by the removal of the limestone by weathering agents has left the rounded concretionary flint masses as residual products and producing a formation known to the miner as "bouldery" or "open" ground of the greatest irregularity in character.

Frequently the limestone and flint are interbedded, and sometimes in shafting, the surface is covered with limestone, underneath which are heavy bodies of flint with ore. The thickness of the overlying limestone varies with the locality. About Carterville, and in many places in the vicinity of Joplin, particularly west and N.W., the limestone is from 50 to 100 ft. thick, and even more in extreme

cases, while beneath the heavy lime cap, flint bodies with rich ore are found. Farther east toward Pierce City and Aurora, the bedding plains of the flint rock are more pronounced, and the interbedding of limestone and flint is more noticeable.

In addition to these two primary rocks there is throughout the whole mining district a large amount of secondary material deposited from water solution, and varying in color from a light brown almost to black, dependent upon the nature and amount of impurities. Usually a considerable amount of zinc blende is present in this matrix, in some places scattered so evenly through it, and in such finely divided particles that it is impossible to separate it by any milling processes thus far employed; in other places the ore forms well shaped crystals, surrounded by the secondary rock in such a way as to show that the crystals were formed before the enclosing rock mass. An extreme form of this matrix is found in the bouldery ground where the openings between the boulders are filled with a variety of minute particles of quartz-sand and clay not yet hardened into a solid rock, which often carries large quantities of ore. In the opposite extreme the secondary rock assumes a hardness almost equal to that of the primary flint itself. The exceedingly large amount of it is a great surprise to geologists who have studied the district, and represents a degree of mineralization or deposition from water solution, perhaps almost unsurpassed elsewhere in America. It constitutes the Cherokite of Jenny.

The openings in the rock masses constitute the cavities into which the ore has been deposited, and are of great variety. In the bouldery ground it appears that weathering agents have dissolved out the greater part of the limestone, and have left the flint, clay, and an occasional lime boulder which has escaped solution, as an irregular residual mass. The flint masses vary greatly in size and shape; some are of angular fracture fragments, and others of rounded concretionary masses, and as there has been no sorting by water or other agents of translation, these residual masses are exceedingly heterogeneous. The openings between the rock fragments are partially filled with clay left as a residue from the solution of the more or less impure limestone. Percolating waters also have deposited a variety of material from solution.

Sometimes ore is totally absent from places where good opportunities have apparently been afforded for its deposition. Elsewhere nearly the whole of the spaces between the rock boulders is filled with ore, and constitutes an exceedingly rich deposit. The masses of ore vary through all degrees from extreme fineness to large boulders weighing hundreds of pounds, and in rare instances tons. During the summer of 1898 near Central City an opening of this kind was found which was entirely filled with a soft mud-like mass containing zinc blende in large quantities in the form of minute crystals about the thirty-second part of an inch in diameter. It produced over 200 tons of mud which yielded when milled more than 100 tons of zinc ore.

In locations where the ore is found in a comparatively solid flint rock the principal openings are fissures which, in a vertical position, produce the so-called "ribbed ground." This type of opening is most abundant at Galena, and is also well known in the vicinity of Joplin, Carterville and Webb City. It

seems that earthquake movements have fractured the flint rock to the greatest possible extent, and that the fractures generally assume a vertical direction with the greatest variety of azimuths, with a predominance of those N.E. and S.W. They may be a few inches apart, although traceable in an up-and-down direction as far as the height of the mine opening permits examination. The fissures vary in prominence; some are hardly visible, others are from 2 to 12 in. wide, and in extreme cases they are from 5 to 20 ft. wide. The fissures have been filled with vein stuffs similar to the openings in the bouldery ground. The larger ones occasionally are filled with the black rock which carries a variable amount of ore, or they may be entirely filled with a shale-like mass producing the so-called "soapstone bass."

It frequently happens that the fractures in the flint rock are irregular in position and direction. Sometimes large masses are found with fractures crossing them in all directions so close together that it is difficult to find a piece of flint an inch in diameter entirely free from fracture. The miners ordinarily express this ultra-fractured condition by the somewhat inelegant, and yet expressive term of "hog-chawed flint."

Farther to the east, vertical fissures are much less abundant, and are hardly noticeable around Aurora and Wentworth. The ore bodies there are confined principally to the bouldery ground, similar to that of Joplin and Galena, or to the openings between the different layers of limestone and flint. Limestone is generally found at the surface, and well bedded layers of flint are encountered at a depth of from 50 to 100 ft., with considerable spaces between individual layers filled with mixtures of clay and ore and some of the black rock in a manner similar to those obtained at Joplin and Galena. This condition is so marked that the mining methods are somewhat varied. Instead of the imperative use of the drill for the reception of the charge of blasting powder, it is often possible to push an iron bar into the spaces between the flint layers for a distance of 5 or 6 ft., and make an opening into which powder can be placed. This process is described by the miners as "snaking," and can be accomplished at a very small cost.

An extended examination of the Galena-Joplin mining district impresses one with the idea that the numerous fissures in the rocks are in some way connected with the earthquake movements which accompanied the Ozark uplift. The Ozark area is elliptical in outline with the major axis some 250 miles long, with a N.E. and S.W. trend, and reaches from a point in the Indian Territory, a little north of the Arkansas River to a point north of Iron Mountain, Mo. The transverse diameter of the ellipse is one-half as great. It is now believed that this uplift was produced by forces which acted parallel to the radii of the earth, and formed a true monoclinal type similar to the monoclinals so well known in the great Cordillary area of the West. Here, however, the uplifted area is so small that the term monoclinal is hardly applicable, although in miniature it corresponds so well in structure.

This uplifting process caused unusual fracturing around the borders, accompanied by a stretch or elongation of the strata, as is shown abundantly by the numerous fractures in the coal beds of southeast Kansas and southern Missouri,

which is N.W. of the area considered. A study of the Kansas coal fields shows that these fissures, or "horse-backs," as they are called by the miners, exist in great abundance in Cherokee and Crawford counties, Kan., just at the N.W. limit of the Ozark area, while further to the north and west such fractures are entirely unknown. As all these fissures are opened, some to an extent of 20 or 30 ft., they furnish positive proof that a great stretch or elongation of the strata was produced. If this explanation be accepted one can readily understand: first, how the earthquake movements would fracture all the rock strata around the borders of the uplifted area; second, how the general results would be an opening of the fractures by stretch rather than a closing by lateral pressure; and third, how the fractures would be much more numerous in areas of exceedingly brittle rock like flint, even though the same Sub-Carboniferous rocks extend far beyond the flint areas.

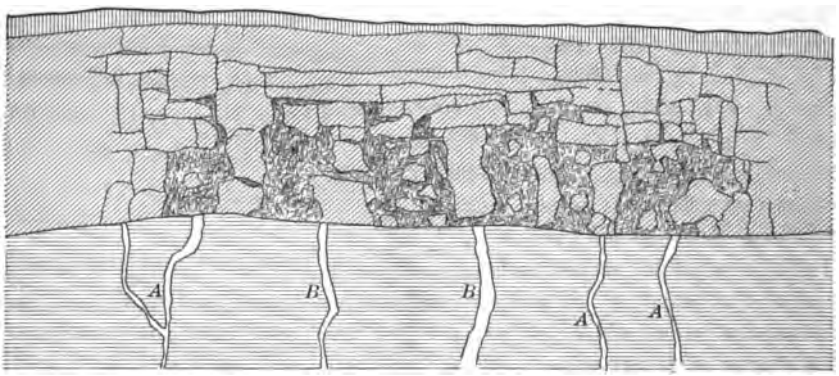
The Aurora-Pierce City region is too far removed from the borders of the uplifted area to have suffered such intense fracturing as occurred at Joplin and Galena, while the Sub-Carboniferous rocks which extend northward to the vicinity of Sedalia are almost barren of ore because they contain but little flint, and hence could not have been easily fractured; and because they are too far removed from the Ozark area to have been seriously affected by the earthquake movements which accompanied the Ozark uplift.

One cannot refrain from the mention of the fact that the whole mining area of southern Missouri and northern Arkansas, so far as mining developments have yet been made, corresponds almost exactly to the mining area and the borders of the Ozark uplifted area. We have the mines of Poplin and Galena on the western border which reach to the N.E. through southern Dade County, and cut the corner of Cedar County, Polk, Hickory and the southern half of Benton County, Morgan, Camden and Miller counties of Missouri. On the east and S.E. the line passes through Crawford, Washington, St. Francis and Madison counties, and includes the well-known mining areas of Potosi, Bonne Terre, and Mine la Motte, and the copper mining area of Ste. Genevieve County. Then turning backward to the S.W. borders of the uplift area we have the unusually wide expanse of "shines" throughout Renyolds, Shannon, Carter, and Howell counties, and into northern Arkansas through Baxter, Marion and Boone counties, with a relatively wide distribution of both lead and zinc ores throughout almost all of the southern tier of the counties in Missouri. It is probable that future developments will be confined with tolerable closeness to this border area, although possibly here and there farther up toward the dome of the uplift, valuable mines may be found. The larger the amount of flint rock, which is imbedded with the limestone, the greater the probability of such occurrences as the extreme brittleness of the flint permits its extensive fracturing by earth tremors which would hardly affect the more elastic beds of limestone and sandstone. The existence of the ore beds of the Aurora-Pierce City district can be accounted for in this manner, as they are near the summit of the Ozark dome.

The following opinion is held by Frank Nicholson in regard to the mode of occurrence of the Joplin ores:

"It is a matter of considerable interest to the geologist and also to the miner

that all the ore bodies so far found in the Joplin district occur along determinate lines of fracture. At the time of the Ozark uplift, there was a bending and buckling of the strata toward the west, reaching well into Kansas and the Indian Territory. These seismic movements resulted in certain lines of fracture that became afterward channels for subterranean waters. The formation consists of magnesian limestone of the Lower Carboniferous period, interstratified with chert, and the circulating waters carrying carbon dioxide dissolved out the lime and carried it away in solution. The chert being insoluble was left behind, and the fractured masses being without support, fell into the cavities resulting from the dissolution of the lime. Later, thermal waters carrying in solution the sulphates of lead and zinc traversed these same cavities, and meeting with sulphur waters, the sulphates were precipitated as sulphides, and formed a deposit, cementing together the angular fragments of chert. These deposits are all



Soil.
Sub-carboniferous.
Archaen.

A—Fissure formed prior to the sub-carboniferous period.
B—Fissure formed subsequent to the sub-carboniferous period.

FIG. 1

TYPICAL CROSS-SECTION SHOWING FORMATION AFTER ERUPTIVE ACTION REDUCED PART OF SEGMENTS TO A BRECCIATED STATE, THEREBY FORMING CHAMBERS FOR THE RECEPTION OF ORE DEPOSITIONS.

essentially cavern deposits, and while in their shape they are more or less irregular, yet they all approximate more or less the lens form, and the outline of the ore bodies resembles very closely limestone caverns in other districts. This condition is clearly show in the illustrations given in Fig. 1.

"A careful study of the walls of each cavern will, therefore, enable the engineer to estimate with reasonable probability the remaining contents of that cavern, after a certain portion of the ore has been removed. In fact, this is the only method of getting at the available ore in this district. "Ore in sight," in the technical acceptation of that term, does not exist in southwest Missouri. In other words, it rarely or never happens that a man owning a mine in this district blocks out the ore so that the examining engineer is enabled to get at and sample three sides of the block of ore."

The different ores of the Galena-Joplin district are, in the order of their abundance, zinc blende or sphalerite, galena, pyrite, marcasite, and chalcopyrite,

associated almost exclusively with the minerals calcite, dolomite, and small quantities of baryte or heavy spar. The greater part of the vein space is occupied by the back rock above described, which may be looked upon as a true gangue, as it originated from water solution in a similar way to the usual deposition of other gangue materials.

Calcite, the "tiff" of the miner, is most abundant in the limestone areas or adjacent to them, and consequently is found in much larger quantities about Carterville, Webb City, and to the north and west of Joplin, than elsewhere. It is least abundant in the heavy flint areas of Galena, where even a trace of it is rarely found. Dolomite occurs intimately associated with calcite, and is more abundant even than the latter, although never in such large crystals. The extremely curved rhombohedra are generally from a quarter to half an inch, and are found plastered against the walls of the fissures and openings, particularly abundant in contact with limestone. Pyrite and marcasite occur irregularly throughout the entire area, with marcasite in much the greater amount, and constitute the "mundic" of the miner. Their presence is very objectionable, as their gravity is so near to that of blende; they cannot be completely separated by any of the milling processes now in operation. The ore-buyers dock the ore heavily if either pyrite of marcasite is present.

Chalcopyrite is found in small quantities throughout almost the entire area. It occurs as small three-cornered crystals which rest upon the surface of calcite, dolomite, or zinc blende, and not infrequently upon fragments of the flint rock itself, showing that it is of more recent formation than the material upon which it rests. Heavy spar is found in small quantities with no regularity of occurrence.

In general the zinc ore occurs at greater depths than the lead ore, but exceptions exist. It is not at all uncommon to hear the miners or mine operators speak of a shaft too deep for lead ore, and not deep enough for zinc ore. The frequency of such a statement points to the evidence of such a relation. Yet throughout the whole district there are many exceptions to this rule. Sometimes zinc ore is found at the very surface as on the south bank of Short Creek, near the State line, between Missouri and Kansas, where the zinc ore associated with marcasite occurs in comparatively great qualities immediately beneath a thin soil covering, and, in reality, it is found "from the grass roots down." Elsewhere the drill or shaft has passed through more than 150 ft. of rock before zinc ore was reached. In many instances the two ores are intimately associated so that hand specimens may be found with equal quantities of each.

Lead ore frequently occurs at comparatively great depths and below zinc ore, as is abundantly shown in many mines located over the entire district. For example, at the Sunflower mine, Galena, during the summer of 1899 a shaft, from which large quantities of zinc ore had been obtained, was deepened; at a depth of 130 ft. a large body of lead ore was reached which carried but little zinc ore. Near by, on the Geisinger land, a shaft, after it had passed through considerable zinc ore, developed a heavy body of lead ore at a depth of more than 100 ft., which produced as much as 50,000 lb. a week. At the present date it has maintained its productiveness, although this body of lead ore has been mined for nearly six months.

During recent years there has been considerable discussion throughout the mining district in regard to the probable depths at which ore can be found, and a strong belief is held that ore exists in as great or greater quantities below the 200 ft. level as above it. It is difficult to decide the truth of this view on account of the limited knowledge of the conditions below the bottom of the mines now in operation. The number of drill holes of more than 200 ft. depth is so small that it is somewhat hazardous to draw conclusions from the slight evidence afforded by them. In the Galena district, the great majority of the shafts are less than 100 ft. deep, the deepest one is 165 ft. on the south side land. In the vicinity of Joplin mining in general is deeper, many shafts are from 150 to 250 ft. deep, while at Webb City it is reported that one has reached 280 ft. All, or nearly all of these deep shafts reveal ore at the bottom as good as that found at shallower depths. This, so far as it goes, favors the idea that future developments may be based upon the well grounded hope of a continuance of the ore to depths from 300 to 500 ft.

COST OF PROSPECTING BY DRILLING.*—(By Harold Titcomb.)—The ore bodies of the Missouri-Kansas district are exceedingly variable in size, shape, richness and depth below the surface, and the indications of their existence are vague.

Many ranges of ore have been traced in northeasterly or northwesterly directions. and old miners often speak of the "Two o'clock and Ten o'clock ranges." A large number of fractures in the country rock have N.E. or N.W. strikes, and these fractures may have been influential in determining the flow of ore bearing solutions. In the early days of mining nearly all prospecting was confined to the valleys or "draws," but many rich and productive mines have been located on hills.

Prospecting is carried on by means of shafts or drill holes. On account of the presence of irregular and shattered layers of chert and limestone, it is impracticable to use a diamond drill in the district, and the cable-tool system, called churn-drilling, is employed.

A portable, self-contained style of prospecting drill, either traction or non-traction, is generally used. A few oil-well derricks are seen here and there, but as the depths of holes drilled is usually but 150 to 250 ft., the portable drill is more convenient and popular. The "Star" and the "Keystone" drills are most favorably considered. The cost of a drill with one set of tools is about $1,100 cash, or $1,300 if bought on the installment plan. In the latter case the drill manufacturer usually holds a chattel mortgage on the machine until paid for. These prices include only necessary tools.

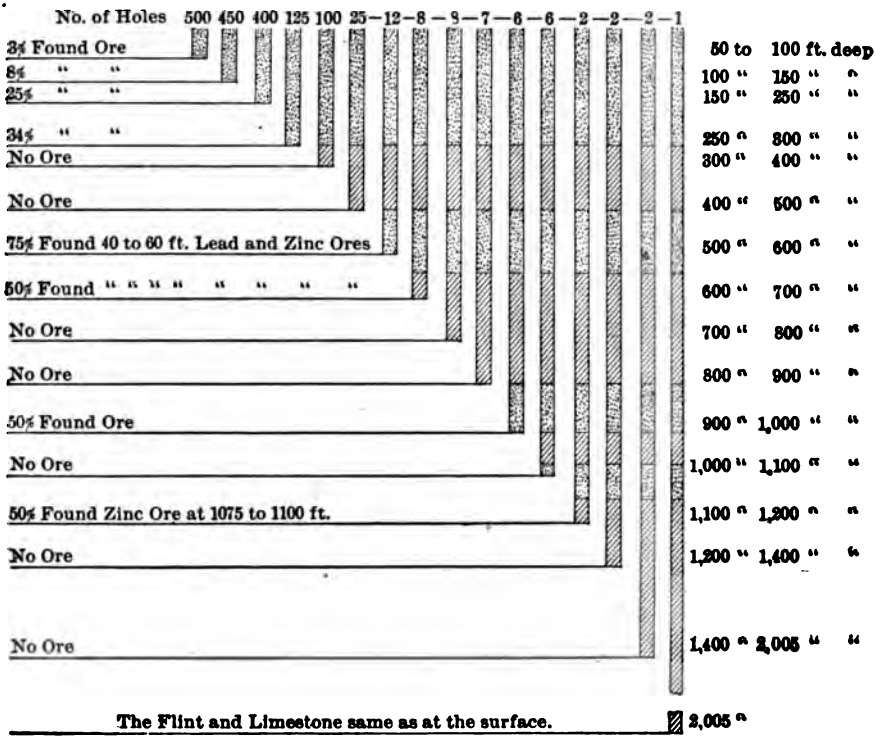
The size of hole drilled is usually 5·625 in. in diameter. An 8-in. bit is often used in starting a hole, especially where the surface is soft, and casing is put in where the ground will not stand well. The non-traction drills require four to six horses to move them, and the time occupied in moving a drill one-quarter mile and in setting it up ready to start the new hole, is from 2·5 to 5 hours. Two men are required to operate a prospecting drill.

The drills are usually driven by steam engines, fed by bituminous coal. The

* *Engineering and Mining Journal*, April 7, 1900.

cost of coal is variable, but usually as low as $1·50 to $2·50 per ton delivered. A drill engine requires one-third to one-half ton of coal per ten-hour shift. Water, for the boiler and the drill hole, is brought to the drill in barrels or tank-wagons, for there is seldom a natural supply of water at the locality where the hole is to be put down. The consumption of water may be stated as about two bbl. per hour, and the cost as 5 to 8c. per bbl. In drilling, water is usually struck in the hole within 30 or 40 ft. of the surface, except in localities contiguous to mines, where the ground is dry to a greater depth.

The somewhat hackneyed phrase, "depending on conditions," is seldom more



P. L. CROSSMAN & BROTHERS' CHART SHOWING DRILL RESULTS IN THE JOPLIN REGION.

appropriate than when applied to the speed of drilling. The following figures on speed were given by a drilling contractor: In ordinary soft limestone, 2 to 3 ft. per hour; in hard, tough, blue limestone, as low as 1 ft. per 10 hours; in ordinary chert ("flint"), 1 to 2 ft. per hour. Soft, white limestone is the most favorable ground to drill. The hole requires no casing, and a bit often lasts until 10 ft. has been drilled in the rock before it needs resharpening. As a rough average, the speed of drilling may be stated at from 100 to 200 ft. per week of six shifts of 10 hours each.

The cost of drilling a hole is largely dependent on the speed with which it is put down, and hence is very variable. The contract price charged for drilling

to ordinary depths, as 200 to 250 ft., is $1·25 per foot of hole. The actual cost is made as follows, speed being assumed at 100 ft. per week of six shifts of 10 hours each:

	Per week.		Per week.
One drill man at $3·50 per day	$21·00	Moving drill and sundries	$7·50
One assistant, at $2 per day	12·00		
Water, coal and oil	12·00	Total cost	$62·50
Wear and tear on ropes and machinery at 10c. per ft. of hole	10·00	Cost per ft. of hole	0·62½

Another estimate per foot leading to the same results, is: Labor, 25c.; water, coal, oil and repairs, 25c.; incidentals, 12·5c.; total cost, 62·5c.

As there is no fixed limit to the depth at which the ore bodies occur, it can be said that the deeper the hole the better the chances for striking ore.

The drilling firm, P. L. Crossman & Bros., has compiled a chart covering the results of about 1,650 holes drilled during a period of 30 years.* According to this chart (given on the preceding page) out of 450 holes drilled to a depth of 100 ft., 8% found ore; eight out of 12 holes, put down 500 ft., struck ore. One hole found zinc ore at 1,075 to 1,100 ft. "At a depth of 2,000 ft. the cuttings show no change in the mineral formation of the country."

Sometimes a certain locality will seem to be underlaid by an almost continuous ore body or series of ore bodies, and nearly every drill hole strikes ore. Other tracts of land are barren for wide areas, or contain ore in scattered and infrequent deposits.

MINING AND MILLING.—(By Frank Nicholson.)—Perhaps in no other mining district in the world are appearances so deceptive as in southwest Missouri. It invariably happens that when an engineer from some other section visits this district, he is filled with surprise at the crude appliances and unminer-like methods of extracting and treating ore in this district; in fact, it has become an old story with the natives to have one engineer after another denounce their primitive methods, and speak of revolutionizing the industry, but it may be safely stated also that in nine cases out of ten these attempted revolutions result in the impoverishment and total discouragement of the revolutionizers.

There is some reason for all this. The southwest Missouri zinc district is not a new camp; lead ore has been extracted there for nearly a half century, and for the last thirty years zinc ore has been mined in steadily increasing quantities until the present monthly output approaches $1,000,000 in value. The mining methods in vogue in the zinc district are essentially the outcome of a natural evolution—the result of long experience. Almost every imaginable device has been tried to benefit the handling of the ores, and those appliances of value have been adopted and retained; the result being that to-day the standard methods in use are specially adapted to the existing conditions. Large shafts, equipped with cages, expensive head gear and costly hoisting appliances so common in many mining regions, are out of place in a district where it is cheaper to sink a new shaft to the ore than to tram 400 ft. underground.

The mode of occurrence of the ore in southwest Missouri differs so radically from that with which the average mining engineer is familiar, that any attempt

* *Engineering and Mining Journal*, March 18, 1899.

to apply fissure vein methods of working to these superficial deposits results in an enormous increase in working costs per ton of rock handled.

Method of Mining.—A shaft is sunk from the surface until it reaches the ore body, which is usually at a depth of from 60 to 150 ft. The shaft is then sunk say from 10 to 30 ft. into the ore body, leaving 5 to 10 ft. as a sump, and a heading is started out from the shaft, taking the ore from the floor of the drift to the top of the ore body. This heading at the start is, of course, only the width of the shaft, but is gradually widened out to 20 ft. or more, until the walls of the cavern are reached, when the entire ore body is extracted to the level of the floor of the original drift above. After this extraction is completed the shaft is again sunk a distance of from 10 to 30 ft., and the process of extraction is repeated until the entire productive area is exhausted. In case of very soft ground, requiring careful timbering, the method of procedure is as follows: The shaft is sunk to the ore body, and about 18 ft. into the ore; a sump of 6 ft. or more is left, and an opening is cut in one side of the shaft through the cribbing. A heavy sill is laid down, two posts are set up on this sill and a cap placed on them; pointed poles, known as "spiles," are then driven close together over the top of this cap into the soft ore body a distance of 6 or 8 ft., as in ordinary forepoling, so as to form a lagging for the next set of timbers. After these poles are driven into place the material is excavated below them, and the drift in this way is started. A second set of timbers is put in at a distance of from 3 to 6 ft. from the shaft; a second set of spiles is driven below the first cap and above the second cap, the material excavated beyond the second set of timbers, below this set of spiles, and the process repeated. Ordinarily, after the drift has gone a short distance from the shaft, it is carried about 12 ft. in height and from 14 to 20 ft. in width, depending upon the nature of the ground. After one drift is extended entirely across the productive area, a second drift is taken up by the side of the first, and so on until the entire cavity is excavated down to the level of the floor of the drift. This being completed, the shaft is sunk another 12 ft., the entire productive area already worked over is floored with lagging poles, varying in diameter from 4 to 6 in.; holes are bored in each post about midway of its height, and one-half a stick of dynamite is placed in each augur hole, and all are fired simultaneously. This breaks the posts and the upper ground comes down solid upon the floor of poles, and makes the taking up of the next stope a much simpler matter than the preceding one. In extracting the ore from the next and every subsequent level, the roof is prepared, and the work consists of setting up the ordinary sets of timbers, sill, two posts and a cap. Of course, either in firm or in soft ground, water may be troublesome, and may largely increase the cost of mining, but as far as my own investigations have proceeded, the heaviest amount of water encountered by any one pump shaft in the district is about 1,500,000 gal. a day; it varies from this amount to nothing. Comparatively few mines are entirely dry after the depth reaches 150 ft., unless they are drained by some neighboring pump shaft.

The appliances for hoisting in this district are of the simplest; usually small, single cylinder, paper friction hoists are used, carrying 0·5 to 0·75 in. wire rope, or 1·5 in. Manila ropes, and an ore bucket capable of holding about 500

lb. of mine rock, though in some cases the ore buckets have a capacity of 1,000 lb., and in others, only 200 to 300 lb.

A large amount of work is done by these crude appliances, as all shafts are shallow, averaging about 150 ft. A 15 h. p. engine, with a 500 lb. bucket, can raise without difficulty 100 tons of ore from the 150-ft. level in 9 hours. In exceptional cases twice this amount has been raised; under these conditions this would represent 400 trips in 8 hours, or almost at the rate of a trip every 30 seconds. I have rarely seen a case where the mill was delayed through the slowness of the hoister. It is usually the case that the hoisting engine is not used more than one-half the time, the other half consisting of delays due to slowness of the underground trammers, who, in turn, are delayed by lack of material.

Different varieties of ground are encountered in the Joplin district, very soft, medium and extremely hard, and in some cases careful timbering is required. Mining timbers in this district are plentiful and cheap; a round stick 12 ft. long and 12 in. in diameter at the small end, costs 6c. per in. of diameter, or 72c. per stick, or approximately $10 per 1,000 ft., board measure, delivered.

The cost of pumping varies with the plant used, which in the Joplin district are the Cornish lift, the Cornish force pump, the ordinary type of steam pump and pneumatic pumps.

The Cornish force pump is the most economical and satisfactory of these, where the quantity of water is large and regular. The pneumatic pump has the special recommendation that it can be placed on an ordinary drill hole, and does not require a shaft compartment for its installation. I have gathered figures on the fuel cost for each type of pump, and find that the expense of operating the Cornish force type is less than one-half that required either by the ordinary type of steam pump or the pneumatic pump.

In ordinary ground, ordinary hand drilling is employed, and no timbering is required, except in special cases. The ground can be handled, as a rule, at less cost than either the very soft or very hard ground, and the mines so located have generally proven more remunerative. In very hard ground air drills are generally used, and recently a number of compressor plants have been installed. While no timbering is required in this ground, the expense is increased by the hardness of the rock, and the larger amount of explosives required.

Milling.—The first milling done in the Joplin district was by means of hand jigs. The hand jig is still in evidence, and is the usual transition stage between the period of discovery and development; in fact, until the mine is ready for the erection of a mill. It has been estimated that during the summer months, when the weather is favorable to the operation of hand jig plants, one-third of the total production of the district is derived from them, and while this may have been true up to within the last year, the enormous multiplication of mills during the past twelve months—each mill replacing some hand jig plant—makes it highly probable that the combined output of the hand jig plants is very much less than one-third, or even one-fourth of the entire output of the district.

The standard 100-ton concentrating plant in use in this district is the evolution of experience. A rough comparison of the mills built five or ten years

ago, with a capacity of 25 to 35 tons in 10 hours, with the modern mill, having a capacity of 10 tons an hour, shows the remarkable advance that has been made. The cost of this mill, varying from $6,500 to $8,500, is surprisingly small when its capacity is considered. In the southeast Missouri lead district a concentrating plant capable of handling 10 tons an hour, costs from $25,000 to $35,000, and the saving is certainly no better than that of the modern 100-ton mill in the Joplin district. It is usual in mills of large capacity to have a number of jigs of three compartments each, the number being limited only by the intended capacity of the mill. In the Joplin district, whatever the capacity of the mill, there are usually only two lines of jigs, of five and six compartments respectively, and occasionally a third line, known as "sand jigs."

The ore first passes to a crusher, generally of the Blake type, which is located on top of a storage bin. The bin is provided with an ore bin gate and a chute leading to a pulley feeder, with sheet iron hopper, placed above a pair of 36×12 in. belt driven rolls. This feeder sends the ore in a continuous stream to the rolls which crush and deliver it to the boot of a belt elevator, provided with 16-in. buckets for elevating it to the first cylindrical screen in a set of two described later.

The oversize from the first cylindrical revolving screen, or that which does not pass through the 0·5-in. holes, is delivered to the second pair of 36×12 in. rolls, also driven by belts, and set to crush finer. The product from these rolls also is delivered to the same elevator, joining the product from the first set of rolls, and the entire lot is elevated to the set of two screens. Each of the two screens is 36 in. in diameter by 84 in. long, and provided with hood and sheet steel housing and spout. The first screen is covered with sheet steel, punched with 0·5-in. round holes, and the second screen is covered with sheet steel, punched with 0·125-in. round holes. In the first screen the ore is separated into two sizes; the oversize, or that larger than 0·5 in., is conducted back to the second set of rolls, as already mentioned. The undersize from the first cylindrical screen, or that which passes through the 0·5-in. perforations, is delivered to the second cylindrical screens, which, as mentioned, has 0·125-in. holes. Here the ore is again separated into two sizes. All above 0·125 in., or the oversize, is conducted to a five compartment roughing jig. The undersize, or that which has passed through the 0·125-in. perforations, is delivered to a sand jig, described later on.

Each compartment in the roughing jig is provided with a steel wire cloth screen about four mesh. The speed of the counter shaft of this jig is about 175 revolutions per minute. The tailings from this jig are conducted to the second set of rolls above mentioned to be recrushed. The product is then elevated again to the first screen and sized as before described. The product from the same jig which has passed through the screens is elevated to a cleaning jig with six compartments, provided with screen of 4- and 6-mesh brass wire cloth. The speed of the counter shaft for this jig is about 225 revolutions per minute, and it will make clean concentrates both above and below screens, at least in four compartments. The middlings and tailings from this jig are conveyed to a third set of 36×12 in. rolls, set to crush fine. The product from these rolls

joins the product of the first and second set of rolls, and is elevated up to the set of two screens, where it is sized as above described.

The undersized from the second cylindrical screen, or that which has passed through 0·125-in. perforations, is delivered to a sand jig, which also has six compartments, covered with 12- and 14-mesh brass wire screen cloth, and will make clean concentrates both above and below screens. The speed of the counter shaft for this jig is about 260 revolutions per minute.

The slimes, or the very finely crushed ore, is conveyed to a system of pointed settling boxes. The pulp from the bottom of each compartment in this settling box is conveyed to a separate Frue vanner, with a 6 ft. wide, plain, endless rubber belt. All tailings from this mill are conveyed to an elevator with 50-ft. centers, and thereby discharged to the dump.

Frequently one of the sizing screens is omitted, and in some cases, three screens instead of two are used. It is quite unusual to find a mill with the Spitzkasten and Frue vanners.

Ordinarily, the tailings are not reground, and after the tailings and sand pass the second line of jigs, known locally as "cleaner jigs," they are transferred to the tailings pile by means of ordinary belt elevators and launders; in some cases it is the custom to run the sands into settling boxes, and then treat them by the Emmons sludge mill, which consists of a series of round tanks 24 ft. in diameter, and about 14 in. in depth, together with clean-up tanks, described later on. The operation of the Emmons mill is as follows:

A central cone 8 ft. in diameter, with a slope of 6 in. in 4 ft. serves as a distributor for the pulp. The sand, with about three times its weight of water, is fed into a central revolving distributor, flows to the cone, and thence into the annular space between the outer rim of the cone and the outer shell of the tank. Small blocks of wood are nailed along the outer rim of the cone to distribute the pulp more evenly as it drops from the cone to the annular surrounding space. Long gas pipe arms attached to the revolving distributor, carry pieces of canvas, that drag along the bottom of the annular space outside the cone with the effect that the pulp is leveled as it settles in the annular space, and no channels are formed, and the surface of the pulp in this part of the tank is kept perfectly smooth, forming a very much flattened cone.

As a result of this construction all the particles leaving the central cone are deposited in the annular space, at distances from the periphery of the cone exactly in proportion to their specific gravities; the lightest material being carried the greatest distance, and the heaviest being deposited next to the rim of the cone. When the annular space between the central cone and the outer shell of the round tank is filled with pulp, that portion next the cone is rich in jack. Then comes a portion of medium richness, which must be re-treated in this same tank, and finally a third portion extremely poor in jack, which is rejected. These portions are termed respectively, headings, middlings and tailings. The headings are fed into a hopper at the end of a clean-up tank, consisting of a rectangular box 4 ft. wide and 20 ft. long, with a slope of 6 in. in 20 ft. supplied with a feed board, which consists of an inclined apron, with small, diamond shaped pieces of wood nailed thereon for the uniform distribution of the pulp. A stream of

water is turned on the hopper, and the headings from the round tank are fed in as rapidly as they can be properly handled. A man with a piece of canvas keeps the pulp within 2 or 3 ft. of the head of the clean-up box perfectly smooth, in order to prevent the formation of channels, and to insure the uniform deposition of the pulp in accordance with the specific gravity of the various particles.

The space for a distance of 4 ft. from the head of the clean-up box is extremely rich, while lower down the product is medium, and at the tail of the box almost pure sand is found. The process is repeated until the 4 ft. next to the head of the box is sufficiently clean to go to the ore bins. This sand treated, averages from 4 to 7% of jack, and the Emmons mill saves about 90% of the jack contained in the sand. Of course, there is always a certain loss.

Figs. 3 and 4 and the subjoined specifications show clearly the construction of a standard 100-ton concentrating mill.

SPECIFICATIONS FOR A MILL TO CONCENTRATE 100 TONS PER TEN HOUR SHIFT.—*Ore Bin* (*Crusher Platform*).—Shall be 24×30 ft., and 12 ft. high at dumping place. Floor joist for bin shall be of 6×6 in., floor of 2-in. oak, covered with sheet steel 0·1875 in. in thickness, put on with 0·25-in. bolts; foundation of bin shall be built of 8×8 in. posts, 8×8 in. sills, 8×8 in. caps, and 6×8 in. slip braces, securely set and bolted with 0·75-in. bolts; foundation to be thoroughly braced with 6×8 in. timbers, reinforced, by longitudinal iron bats, both ways, bolts to be 0·75 in.

Foundation for Building.—Foundation shall be of cement and grouting on 8-ft. centers, and of sufficient size and set deep enough in the ground to guarantee the firmness and stability of the building, and to prevent all settling of the buildings.

Jig Room.—Shall be 24×64 ft., with walls 16 ft. high, with 6×8 in. sills, and 6×8 in. posts, with 8×8 in. caps, set on 8 ft. centers, with 2×8 double plates, 2×6 in. rafters; each cap to have a 6×6 in. truss, with bolt drawn through to caps; floor of jig room to be of No. 2, 1×12 in. doubled, posts 2×8 in. braces in jig-room, bents shall be 6×6 in., mortised and tenoned 51 in. run, all nail girths in building to be 2×6 in.

Engine and Boiler Room.—Shall be 30×28 ft. with 12-in. walls and partition between.

Coal Room.—To be 22 ft. long, 14 ft. wide, and 10 ft. high, 2-in. flooring. Coal room to have board roof, batted with 4-in. batting.

Crusher.—There shall be erected on heavy bent work of 12×12 in. timbers, one 16-in. Carterville crusher, same to be securely braced and rodded, and to be properly anchored to dead men bedded in cement; crusher frame to have rod and turnbuckle.

Rolls.—There shall be erected on 10×12 in. timbers, three sets of 24×14 in. rolls; all of the above rolls to be of the Carterville make.

Jigs.—There shall be 1 five-cell roughing steam jig, known as "Cooley Patent," with 30×42 in. cells, and 1 six-cell cleaning jig of same pattern, with 30×36 in. cells; 1 cell, 30×30 in., and 1 four-cell sand jig.

Jigs to be built of No. 1, 2×4 in. timbers, entirely free from black knots, these jigs to have Faust sieves; line shaft to be 2·9375 in., and shall be complete with

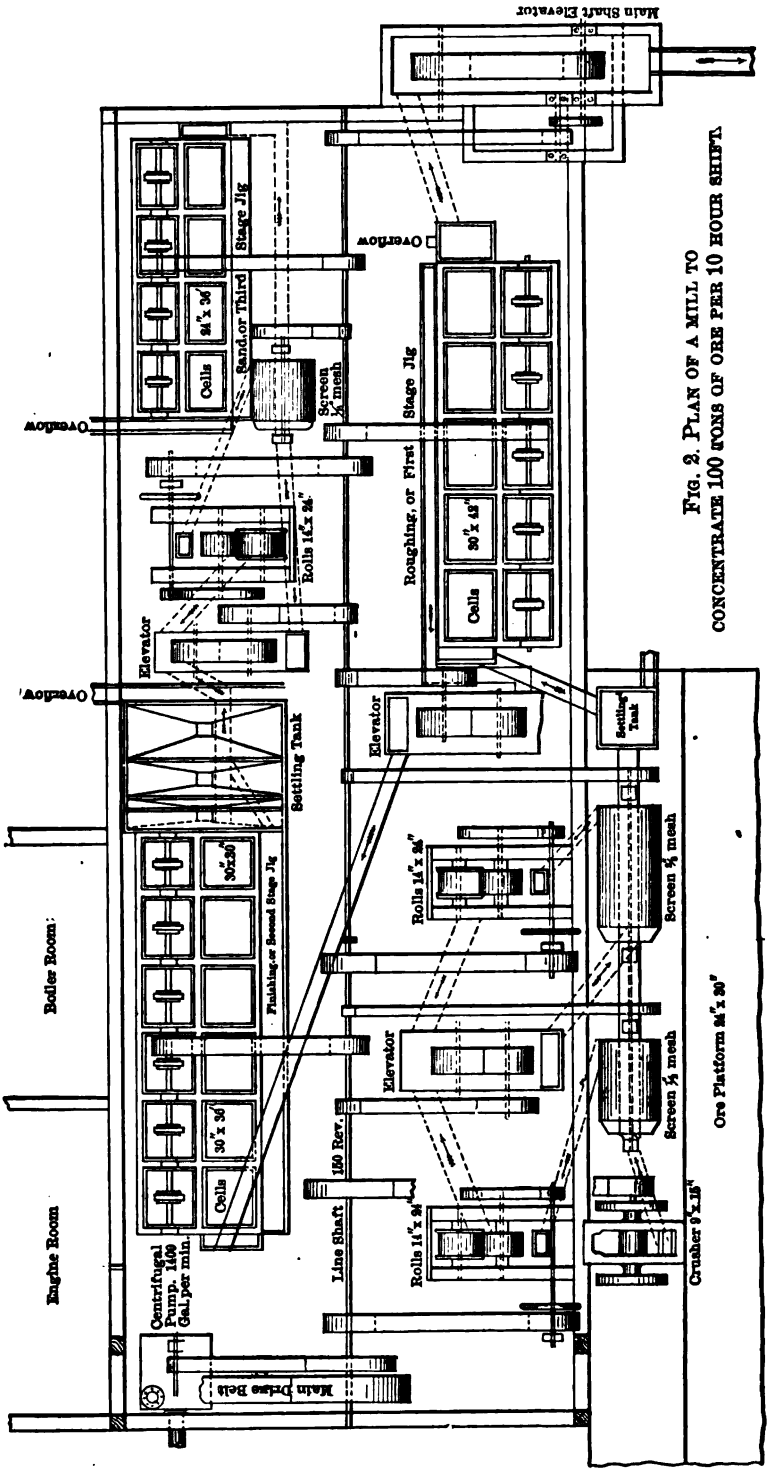


FIG. 2. PLAN OF A MILL TO CONCENTRATE 100 TONS OF ORE PER 10 HOUR SHIFT.

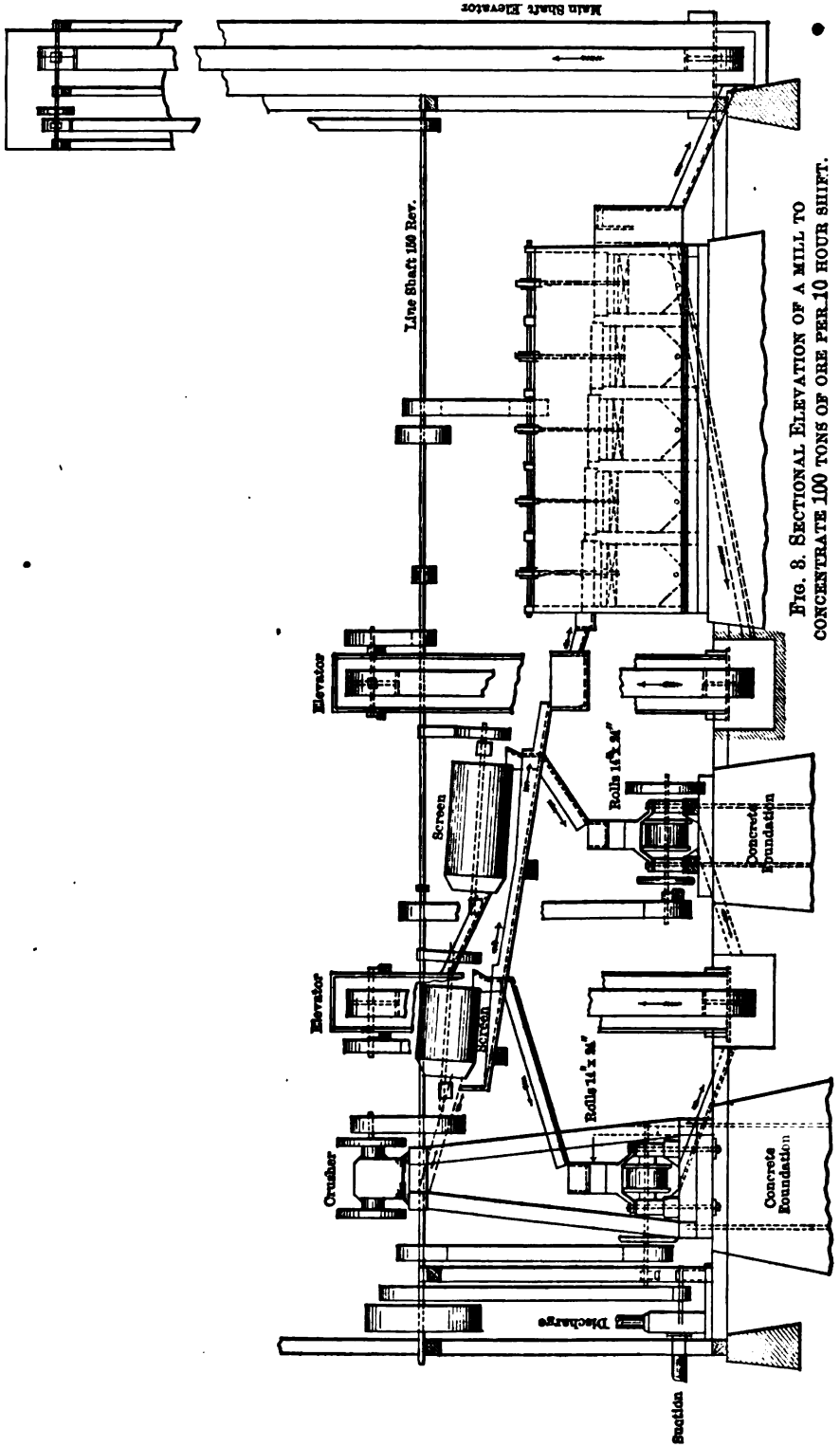


FIG. 8. SECTIONAL ELEVATION OF A MILL TO CONCENTRATE 100 TONS OF ORE PER 10 HOUR SHIFT.

grates, wires, cups, faucets, eccentrics and all necessary pulleys. The first three cells of rougher jig shall be equipped with grates with steel wire on last cells.

Elevators.—One tailing elevator to be 14 ft. long, 4 ft. wide at bottom, and 50 ft. high between pulleys, and to have 36-in. pit pulley keyed to a 2·9375-in. shaft and a 28-in. lap pulley, keyed to a shaft of same size; 16-in. belting and 16-in. cups, to be made of No. 12 steel, placed 20 in. apart, belting to be 6-ply equipped with belt tighteners, with levers; drive belt to be 8-in. 5-ply; legs to trough to be made of 2×6 in. timbers and trough to have six bents.

Two Smittem elevators to be 8 ft. long and 3 ft. wide at bottom, 16 ft. high between pulleys.

One sand elevator 6×3 ft. at bottom, and high enough to carry dirt properly to sand jig. All elevators to be equipped with 28-in. pit pulleys and 24-in. top pulleys to have 14-in. belting and 14-in. cups, of No. 12 steel set 16 in. apart: belting to be 6-ply; elevator shafts to be 2·9375 in.; pulley to be properly keyed on shaft; outside pulleys to be of the Sprinkle make.

Line Shaft.—Line shaft to be 2·9375 in.×64 ft.; this shaft to extend the entire length of jig room, and will have seven heavy babbitted bearings, serving to carry the following wood pulleys:

One 72×14 in., to drive from engine; one 54×8 in., to drive from crusher; one 24×8 in., to drive from roughing jig; one 30×10 in., to drive from cleaning jig; one 30×10 in., to drive from sand jig; three 30×10 in., to drive from rolls; one 16×8 in., to drive from tailing elevator; three 12×8 in., to drive from elevators; one 54×10 in., to drive from centrifugal pump; three 8×8 in., to drive from screens.

All line shaft pulleys shall be wood of the Sprinkle make; all other pulleys except elevator pulleys shall be iron, and shall be key-seated and properly bored and faced; all pulleys shall be of sizes in accordance with above specifications.

Screens.—There shall be one 3×4 ft. screen, 2·9375 in. shafting, and boxes No. 10 (0·5 in. mesh) perforated steel jacket; also one 3×6 ft. screen, shafting same as above No. 10 (0·375 in. mesh) perforated steel jacket; one 3×3 ft. screen shafting same as above No. 10 (0·25 in. mesh) perforated steel jacket. All spouting shall be lined with chilled cast spouting.

Belting.—All belting to be of the Boston make and guaranteed. (Special belting and to excel.) 14-in. six-ply for line shaft; 16-in. six-ply belting for tailing elevator; 10-in. six-ply for elevators; 10-in. five-ply for jig shafts drive belt; 6-in. five-ply for screens drive belt; 10-in. five-ply for breaker drive belt; 8-in. five-ply for rolls drive belt.

Pump.—One 4-in. discharge centrifugal pump with 50 ft. pipe, 5-in. pipe for suction and foot valve.

Tanks.—One supply tank for jigs 8×6 ft. and 5 ft. high, set on 6×8 in. posts, and thoroughly braced; one supply tank for boiler, 10×6 ft. and 6 ft. high, to be set on 6 posts, 8×8 in. timbers, 12 ft. high and thoroughly braced with 2×6 in. timbers.

Engine.—One Erie or Atlas engine, 70 h. p., with Gardiner stop governors of the latest and best make, with all trimmings and fittings, and to be set on 7 ft.

foundation of cement and grouting; the lower 4 ft. of foundation to be of black Diamond cement and gravel, the upper 3 ft. to be of Portland cement and sand.

Boiler.—One Erie or Atlas boiler to be set on a concrete foundation, and laid in hydraulic cement, cased in an 18-in. brick wall, and is to be covered over top by a 1-ft. brick wall and shall have four 6×6 in. buck stays on each side. Boiler shall be provided with flue blower, spuds, rake, feed pumps and whistle, and shall have 1·5-in. hole in dome to supply steam to steam hoist, and shall also be provided with safety valves and blow-off valve, complete steam gauge, water gauge, check valve, and shall also be equipped with smoke stack 36 in. diameter, 50 ft. high, with 0·3125-in. guy wire, and shall be of 100 h. p. capacity

Boxing.—All boxing and batting to be No. 2 grade, boxing, to be 1×12 in.; batting 1×4 in. Shingles to be cedar or white pine, good grade; all rafters to be 2×6 in.; roof to be sheeted 4 in. apart.

Spouting.—All spouting to be 2×6 in. lined with sheet steel.

Costs.—During the past year I have had direct charge as consulting engineer of eight companies at Joplin, controlling in all about 50 mills and 150 shafts, and I am in a position to speak with some authority. The cost of extraction of the ore, as compared with working costs elsewhere, is surprisingly low, a fact due, not especially to the efficiency of the native miner, but to the mode of occurrence of the ore deposit, where it not infrequently happens that bodies of ore several hundred feet in length, 100 ft. or more in width, and from 40 to 60 ft. in thickness, so occur that the entire extraction may be done without any sorting whatever.

It would be an extremely difficult matter to get at the average cost of extraction of one ton of mine rock in this district, owing to the fact that most of the properties are operated by lessees and small owners, who keep no systematic record of their expenses, and who are in total ignorance of the exact amount chargeable to any special item. An examination of the books of sixteen of the largest producing properties, and the records of our own work during the past year show that it is not unusual to extract the mine rock at a cost of 30c., or even less, per ton, and the milling charge has been lowered in some cases to 26·25c. This makes the total minimum cost of mining and milling amount to 56·25c. per ton, and, with an average saving of 6% of zinc blende would represent a cost of finished product amounting to $9 per ton. In other cases, the mining has been as high as a dollar per ton of mine rock, and the milling 34c. a ton, and with 6% ore, the total cost of one ton of ore ready for the smelter amounted to $21·44. In each case, at least $3 per ton must be added for incidentals of various sorts, making the minimum cost $12, and the maximum cost $24·44 per ton. These prices indicate the actual cost of production, and lessees paying a royalty of from 15 to 25%, would add from $7 to $12 to this cost of production, dependent upon the price of ore at the time.

I think it may be stated with certainty that the average cost of producing one ton of concentrates ready for the smelter in the Joplin district, including the royalty and other expenses attendant upon this production, is not less than $30 a ton, although it is possible for the better equipped properties owning both fee and leases to produce ore at costs ranging from $12 to $20 per ton.

PROGRESS IN ELECTRO-CHEMISTRY IN 1899.

By William Borchers.

ELECTRIC FURNACES.

The only progress worthy of mention has been made in the construction of calcium carbide furnaces of which notice will be taken under the section devoted to Inorganic Chemicals later on in this caption. It may be stated, however, that the introduction into furnace construction of the system of incandescent heating seems to have been successful. There is no doubt but that this system is the ideal one for heating; the heat is produced exactly where required without the aid of intermediate agents and without the necessity of producing a higher temperature than is actually needed.

NON-METALLIC ELEMENTS.

Bromine.—Kossuth has found that electrolysis offers a most economical way for the recovery of bromine from the waste liquors of potassium chloride works. The lyes contain 40% magnesium chloride, small quantities of alkaline chlorides, from 2 to 4% of magnesium sulphate and from 0·2 to 0·3% bromine. These solutions were subjected to the following treatment. They form the electrolyte contained in wooden tanks each about 6 ft. long, 2·5 ft. wide and 1·5 ft. deep. Carbon blocks were arranged as electrodes in series 0·5 in. apart and an electromotive force between the two electrodes of from 3 to 3·5 volts was maintained with a current density of about 35 ampères per sq. ft. of surface. The bromine was subsequently recovered from the electrolyzed lye by distillation.

Carbon—Graphite.—With reference to priority in the theory of the artificial production of graphite, my ideas on this subject were published in 1897* as follows: "I wish to point out briefly, that all substances which are able to enter a chemical combination more or less easily dissociable with carbon, will favor the crystallization of carbon. I name especially: borium, silicium, titanium, zirconium, vanadium, aluminum, the metals of the cerite mineral, chromium, molybdenium, wolframium (tungsten), uranium, manganese, iron, nickel, etc."

In the same publication, chiefly devoted to the conditions favorable to the formation of diamonds, I pointed out clearly, that by the aid of the substances

* *Zeitschrift für Elektrochemie*, March 20, 1897.

just named or their compounds at high temperatures, under high pressure will diamonds be formed, while without pressure graphite will result.

Fluorine.—Moissan found that in his apparatus constructed in 1891 the condensing tube for the fluorine may be made of copper. This metal becomes coated with an insoluble layer of copper fluoride, which protects the copper against the action of the concentrated hydrofluoric acid. He obtained 5 liters of fluorine gas with a current of 50 volts and 15 ampères operating from 6 to 10 minutes.

Oxygen-Ozone.—Makers of ozone apparatus have endeavored during recent

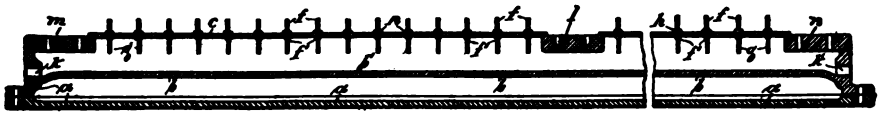


Fig. 1.—Longitudinal Section.

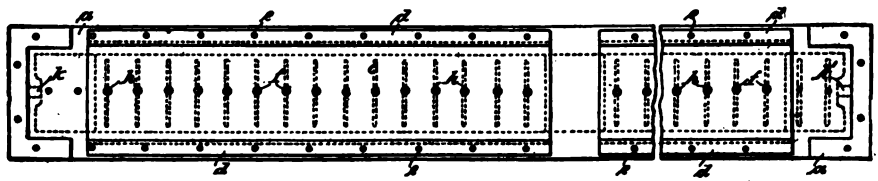
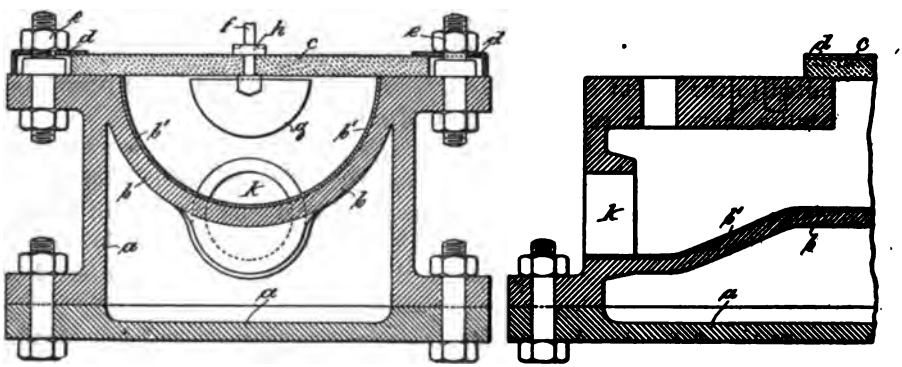


Fig. 2.—Plan.



Figs. 3 and 4.—Enlarged Sections, showing details of construction.

ELECTRICAL OZONE APPARATUS.

years to increase the output of rich gas by alternately cooling and exposing the current of gas to electrical discharges. Tyndall does not recognize this mode of working as economical and demonstrates by his experiments that much better results may be obtained without cooling the gas, if care be taken to pass it through an extended field of uniformly dense discharges. An apparatus on this principle contains in the casing *a* (see Figs. 1, 2, 3 and 4), the semicircular channel *b* with the enamel layer *b'*. The channel is covered by the glass plate *c*, and is held by angle-shaped spring-metal strips *d*, which are pressed down by screws *e*. The glass plate is provided with a number of holes for the insertion of the screws *f*. Semicircular metal disks *g*, preferably of steel are

attached to the slotted heads of these screws which with the disks *g* are held in position by nuts *h*. The high-tension circuit is by preference introduced directly into the apparatus through the screws *f* and, if desired, liquid resistances may also be placed in the circuit. The other pole of the high-tension service is connected directly to the casing *a*.

The gas to be ozonized is introduced through the opening *k* into the conduit formed by the channel *b* and the glass plates *c* and is removed through the opening *k'*. During the time the gas is passing through the conduit, the electric current passes uniformly and evenly in dark discharges from the disks *g* to the enamel layer or surface *b'* of the channel *b*. The metal disks *g* may receive suitable dimensions and inclinations, so as to produce a whirling motion to the gas, due consideration being given to insure that the outer circumference of each of the metal disks *g* is equally distant at all points from the surface of the channel *b*.

Phosphorus.—A full description of the Billaudot electrolytic phosphorus plant will be found under the caption "Phosphates and Phosphorus" elsewhere in this volume.

METALS.

Calcium.—Férée has succeeded in obtaining a calcium amalgam by electrolyzing a 50% calcium chloride solution with a quicksilver cathode and Moissan has obtained the pure calcium by electrolyzing molten calcium iodide.

Chromium.—J. F. L. Moller, of London, and E. A. G. Street, of Paris, have been investigating for some time the rules to be observed for the deposition of chromium from aqueous solutions. So far, however, their publications have been limited to the patent specifications, which do not convince the reader that the difficulties have been sufficiently cleared away.

Copper—Gold—Silver.—No important improvements in the recovery and refining of these metals have been made recently. The Electrical Copper Co., of London, is experimenting with the Perreur-Lloyd and Dumoulin patents. Recent investigations of Förster and Günther in Dresden have established the fact, that organic substances as applied in these patents are anything but desirable in the copper refining baths.

Cowper-Coles of London has made successful experiments and has obtained dense and smooth copper deposits by rotating the cathode fast enough to secure a high degree of friction between the depositing copper and the electrolytic solution.

Glucinum (Beryllium).—The qualities of this metal have drawn the attention of electrochemists to the discovery of new methods of manufacture, yet the results so far obtained from their researches are not apt to cheapen the metal. The chief difficulty does not lie in the electrolysis of beryllium compounds, but in the separation of these compounds from the accompanying alumina and silica of the beryllium mineral. The Liebmann process in which these difficulties were apparently overcome by a simple electrolysis of the mineral with an addition of fluorspar was examined by me. I found that the results claimed in the patents could not be realized.

Iron.—The results of the Stassano experimental plant in the Camonico

Valley of the Bergamosk Alps, Italy, will be looked forward to with great interest, as all previous endeavors to replace the blast furnace process by an electric smelting process have failed. A report of the *Iron and Coal Trades Review*, giving comparative calculations on the outlook of the Stassano process *vs.* the old method, takes a rather antiquated position for the present practice. The period in the development of the iron industry when from 1,600 to 1,700 kg. coal was required to make 1,000 kg. of iron has long since passed, and hopes founded upon this basis will not be realized.

Platinum.—An invention by Zuern proposes to do away with the lead smelting process for platinum ores. Granulated platinum ore in bags of porous material is placed into an electrolytic bath, consisting of an aqueous solution of ammonium carbonate with a carbon rod anode sunk into the granulated ore, which secures the contact with a source of electricity. The platinum dissolves and may be precipitated in the usual manner.

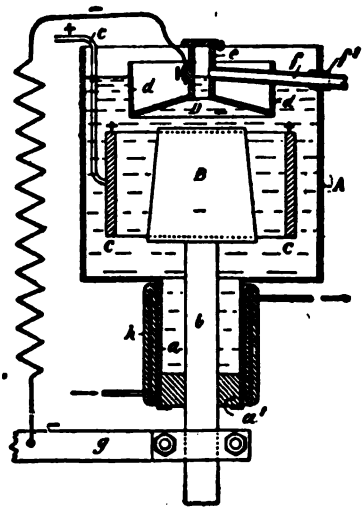


Fig. 5.—Cross-section of Assembled Apparatus.

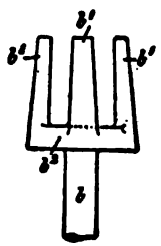


Fig. 6.—Details of Cathode Support.

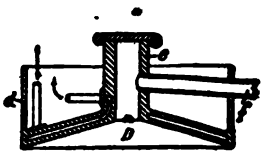


Fig. 7.—Details of Cap for collecting the Sodium.

MODIFIED CASTNER APPARATUS FOR SODIUM.

Sodium.—The manufacture of sodium under the Castner patents by electrolyzing molten caustic soda has constantly increased. Besides the Aluminium Co., at whose works at Oldbury, England, Castner first introduced his electrolytic process, the Electrochemical Co. at Niagara Falls, N. Y., and the Elektrochemische Fabrik Natrium at Rheinfelden, Germany, have started new works. The Farbwerke Meister, Lucius & Brüning at Höchst, who have a sodium plant of about 150 h. p. running, seem to supply their own demand only. The Elektrochemische Werke at Bitterfeld and the Aluminum-Industrie Actien-Gesellschaft at Neuhausen have been working under their own patents.

The original Castner apparatus has been modified to some extent by Becker, of Paris. The principal feature of the new apparatus (Figs. 5, 6 and 7), consists in a collecting cap for the sodium illustrated by *D*. These caps have an upright rim *d*, which allows the application of cooling agents. The other

parts of the apparatus have remained much the same as those of the original Castner cell. *A* is the melting pot with a smaller cylinder *a* projecting from the bottom, to insulate the terminal *b* of the cathode *B*. The insulation is effected by an annular block *a'* of non-conducting material and by keeping the heat concentrated in the upper part of the apparatus, so that the caustic soda in *a* will chill. For this purpose the tube *a* has received a cooling jacket *h* by Becker. The anode is a hollow iron cylinder as in the old apparatus. The light metal ascends during the operation into the vertical tube *e* of the cap and flows through the pipe *f* into a suitable receptacle.

Tin.—At the Töstedt works of Robertson & Bense* a part of the slags and of the hard metal is worked up by the Bohne process, which consists in grinding the slag, mixing it with sulphuric acid, boiling the resultant slime, diluting and filtering the solution and then subjecting it to electrolysis, using a part of the hard metal (tin-iron) as anodes.

Vanadium.—Cowper-Coles† reports that he has found a simple process for depositing the metal from aqueous solutions. The process does not seem to be difficult, but as the raw material is the oxide (known as vanadic acid) care must be taken to obtain the metal free from oxide during electrolysis. The process of Borchers for the reduction of all metallic oxides by electrically heated carbon seems to be much simpler as the vanadium oxides are most easily reduced in the furnace.

Zinc.—For years Cowper-Coles has endeavored to overcome the practical difficulties in the deposition of zinc, and as far as the art of coating other metals is concerned he has met with very good success. His propositions, however, for recovering zinc from ores and other zinc bearing substances are by no means inviting for investigators.

Moore's process is now being tried in a Belgian factory. The operation consists of smelting the zinc bearing ores with sodium bisulphate (niter cake). According to investigations of Neuendorf the temperature for a successful transformation of the sulphides of the ore into sulphates by means of the sodium bisulphate is too high. Only a part of the sulphur oxidizes to form sulphurous acid, another part evaporates unoxidized and the rest remains in the molten mass. The great drawback of this process is the large quantity of sodium sulphate that accumulates in the lyes. Neuendorf therefore recommends the application of more acid sulphates, especially sodium pentasulphate obtained by heating sodium bisulphate with three molecular weights of sulphuric acid:

$$Na_2S_2O_7.3H_2SO_4.$$

The decomposition of the sulphides of the ore into sulphates with the simultaneous disengagement of sulphurous acid takes place at a temperature of 90 to 100°C.

$$MS+Na_2S_2O_7.3H_2SO_4=MSO_4+Na_2SO_4+3H_2O+4SO_2.$$

Both investigators, however, seem to have overlooked the fact that silicates always present in ores, will also be decomposed by this treatment and that the gelatinous silicic acid will be a great impediment in the subsequent leaching and electrolyzing operations.

* See the special article under the caption, "Tin."

† *The Engineering and Mining Journal*, June 24, 1899, p. 744.

The failure of the Ashcroft process has been shown by the results of the Sulphide Corporation. The whole idea of the invention is a contradiction to our theoretical knowledge and practical experience. The German electrochemical zinc works at Duisburg and at Fürfuth stopped running in 1898. At the Duisburg works the leaching of the roasted blende and pyrites was not complete enough to make the remaining iron oxides of profit, although the neighboring iron works would have gladly used the leached oxide had it been sufficiently free from zinc. Another cause of failure is due to the cost of the electric energy, when produced from coal, it is too expensive for the extraction of zinc alone. The Fürfurth works made chloride of lime besides zinc, and for a time they did well, although they are now shut down.

Investigators are working on processes for recovering zinc from molten electrolytes, but they experience difficulty in dehydrating the zinc chloride obtained by the evaporation of the aqueous solution. This chloride retains some water above its melting point and suffers decomposition if attempts be made to expel it by further heating. Lorenz has succeeded in getting rid of the water during the first stage of the electrolysis, by decomposing the water first. Lyte proposes to expel the water by adding zinc dust to the molten chloride or by electrolyzing the molten mass with zinc anodes until all the water has been driven out. These processes certainly deserve more attention than a proposition of Steinhart, Vogel and Fry, who propose to dry the molten chloride in a partial vacuum by a current of dry air.

INORGANIC CHEMICALS.

Alkali and Chlorine Industry.—Among the new apparatus lately patented the only one of value is that of Lord Kelvin (Sir William Thomson), who adopted the Kellner system of working and of arranging the electrodes. The difference between his apparatus and that of Kellner's is in the position of the quicksilver gutters. In the Kelvin apparatus they are arranged horizontally in the bottom; while in the Kellner they are on the side walls.

Calcium Carbide.—The question of the proper method of preparing the material for the furnace is one that has brought forth considerable discussion. The original patents called for a fine grinding and a thorough mixing of the raw materials before they were added to the furnace. When carried out on a large scale, however, the large amount of gas caused much of the fine material to be carried away and deposited in the flues and chimneys, disturbing the process and causing considerable trouble. Borchers' patent* is the first to avoid this objection and specifies that the material shall be coarse and that the lime and carbon shall be charged in separate layers. This arrangement has been proven of advantage by Liebetantz at Düsseldorf and Carlson at the German Refinery, Frankfurt-am-Main.

The discussion of the preparation of the raw materials has developed some rather queer views. One party gives a rule for the size of the raw material, another condemns all reducing agents except coke and a third favors briquetting the furnace charge. The size of the furnace, the mode of heating adopted

* German Patent No. 104,954, Dec. 16, 1898.

and last but not least, the quality of the raw materials, will soon determine the size of the grain, the percentage of finer or coarser product to be allowed in the charge and the way of charging the furnace. I have found wood charcoal a most desirable reducing agent, when it can be had as cheap as coke.

Furnace designers seem to favor the incandescent heating system, and some excellent furnaces of this type have been brought out by the Société des Carbures Metalliques, by Maxim and others.

The statement that calcium carbide can be made without the aid of electricity has hastened the production of patent specifications irrespective of their applicability. According to a patent of Wolff, lime is reduced by aluminum in the presence of sufficient coal to form calcium carbide resulting in the direct combination of part of the aluminum with carbon; a rather expensive way of making aluminum carbide. Two other inventors, Woods and Byrom, at first produce hydrogen and then hydrogen carburets, the latter components in large excess. They seem to expect a great heating effect from the combustion of the hydrogen, a result which will not be ultimately successful.

At meetings of carbide and acetylene manufacturers, calculations of the theoretical and practical manufacturing expenses have been discussed. Notwithstanding that the difference in the bids for 24 electrical h. p. hours is pretty large—the lowest being 3·44 and the highest 9·38 per kg. of calcium carbide produced—the figure determined by me in 1895* at 4·1 to 4·4 kg. for 24 electrical h. p. hours is still nearest to the actual results of large furnaces.

White Lead.—The Elektrochemische Industrie Gesellschaft at Cologne-on the-Rhine, Germany, working the Luckow patents have produced electrolytically 2,500 tons of white lead during 1899.

* *Jahrbuch für Elektrochemie*, 1895, Vol. 1., p. 289.

LIQUID AIR.

By T. O'Conor Sloane.

The production of liquid air on a large scale and its subsequent manipulation are among the most interesting of the modern developments of natural science. For many years the scientific world has been greatly interested in the liquefaction of gases and if we examine the records of science from the beginning of the nineteenth century we will find that the greatest interest has always been aroused by discoveries in this direction. Among the early triumphs of Faraday is the liquefaction of chlorine gas, which although easy of accomplishment, was esteemed so great an achievement that the question of priority in it occasioned considerable feeling between him and Sir Humphrey Davy. Eventually Faraday found that the gas in question had been liquefied years before (in 1805-1806) by Northmore. Faraday's date was 1823. Again when Natterer, in 1845, produced liquefied carbon dioxide gas in large amount, the exhibition of the liquid was most enthusiastically appreciated by popular audiences. The audience of 1845 greeted liquid carbon dioxide and the snow-like solid produced by its evaporation, with the same degree of interest that the public of the present day has shown in the liquefaction of air.

In order to prove the unities of nature, it was necessary to liquefy all gases, and it seemed discordant with the general system of things that some elements should persist in retaining the gaseous state under all conditions. Gradually the gases, one after another, succumbed to pressure and cold until six only were unliquefiable by any processes applied. These were oxygen, nitrogen, hydrogen, carbon monoxide, nitrogen dioxide and marsh gas. Air is essentially a mixture of oxygen and nitrogen with small amounts of other gases, and its liquefaction is practically the liquefaction of a mixture of oxygen and nitrogen; for this a temperature of about —220°F. is required.

L. P. Cailletet of Châtillon-sur-Seine, France, and Raoul Pictet, of Geneva, Switzerland, claim the honor of priority in the liquefaction of air or its constituent gases. Cailletet used a very sudden release from pressure to produce the degree of cold required for liquefaction, while Pictet applied great pressure combined with external cooling circuits. Cailletet's method to a definite extent suggests the processes which have been so successfully applied for the production of liquid air on a large scale. He produced the cold by the expansion of the compressed gas itself, which is the principle employed by Tripler in the United States, Hanson in England, and Linde in Germany.

Cailletet's liquefactions, while of sufficient value from the scientific standpoint to win the enthusiastic approbation of the French Academy, were defective in scale. A very small quantity of gas was compressed to an enormous degree in a small glass tube. A powerful pump was required for this work, and the contrast in size between the large pump and the small glass tube was most striking. The gas to be liquefied was compressed in the tube and a few minutes' standing lowered its temperature; a valve was then opened to reduce the pressure suddenly to that of the atmosphere; as a result of the sudden expansion the temperature of the gas was lowered sufficiently to form a cloud of liquid which, however, disappeared immediately after its formation.

The first observation of this liquefaction was accidental. In an experiment with acetylene a cloud was produced which Cailletet attributed to moisture, but on investigation it proved to be liquefied acetylene gas. This furnished the clue that led to the production of clouds and dew from the so-called permanent gases.

Pictet compressed the gas to be experimented with in a long tube surrounded by a larger concentric one, the two forming a Liebig's condenser. A very cold liquefied gas as carbon dioxide, was admitted to the outer tube and a partial vacuum was maintained therein, so that the liquid in the outer tube instantly evaporated and produced thereby an intense cold. The carbon dioxide was cooled before admission by passing it through a preceding Liebig's condenser, which was cooled by the evaporation in its outer tube of another liquefied gas as sulphur dioxide. This system of cumulative cooling in successive cycles marked a new and definite method and has been very extensively adopted by experimenters. Pictet was not satisfied with merely demonstrating that one or the other gas could be liquefied, but desired to produce them in quantity.

There was a close approximation between the dates of the two investigators. It is safe to say that in 1876 neither Pictet nor Cailletet ever contemplated the possibility of carrying liquid gases about in pails as if they were water. This achievement is of comparatively recent date.

For every substance capable of existing in the liquid and gaseous states there are two important elements, termed critical points: One is critical temperature; the other is critical pressure, a secondary point dependent in a sense on the first. For every gas the general process for its liquefaction includes the application of pressure and cold. For each gas there is a temperature above which no pressure, however intense, will reduce it to a liquid, and below this temperature pressure will liquefy it. This temperature is its critical temperature and the pressure required to liquefy it at this temperature is termed the critical pressure. The difference between the two is clear; a sufficient cooling will liquefy any gas at any pressure, while no pressure is sufficient to liquefy a gas until its temperature falls below its critical point.

The great success which has attended the liquefaction of gases in recent years is due to the full utilization of these principles. The modern apparatus produces intense cold, and the air submitted to its action liquefies at ordinary pressures. The cooling is effected by the expansion of the air or other gas under treatment, which reduces the temperature; this cooled gas is utilized in cooling

the unexpanded gas by its circulation around the pipes containing it, and reduces its temperature constantly until it becomes sufficiently cold for liquefaction. This principle is termed self-intensive action and is precisely analogous to the action of a Siemens regenerative furnace. The earliest authentic suggestion of this action is found in an English "patent," issued to Carl W. Siemens, July 29, 1857. The patent is one of the preliminary statements termed "provisional specifications," which are not supposed to be as complete as the full letters patent.

Siemens first compresses air and then cools it; it is then supposed to expand in a cylinder or engine of any suitable construction with a consequent fall of its temperature; the cooled air then goes into an interchanger, where it cools the compressed air still more and then escapes into the air. The engine is kept cool by immersing it in brine, also cooled by the expanded air. Thus a constant reduction of temperature is provided for and in the words of the specification "the principle of the invention is adapted to produce an accumulated effect or an indefinite reduction of temperature."

The apparatus seems very crude to-day, yet it contains the idea of self-intensive refrigeration, and probably led one of the recent inventors to a very remarkable enunciation, in appealing to the Joule-Thomson effect for the working of his apparatus. This Siemens' specification is one of the most important references in the case as it describes self-intensive refrigeration and contemplates great reduction of temperature produced by the expansion of a gas released from compression.

Later, under date of November 6, 1885, we find also in the British Patent Records a similar specification by E. Solvay which contemplated self-intensive action for the production of low temperature. Like the Siemens' patent an engine was contemplated for the operation.

The modern self-intensive processes are simplifications of Siemens' and Solvay's apparatus, but the simplification is so vital that it constitutes a true invention. Both Siemens and Solvay seem to have been tied down to the idea that the expanding air must be made to do visible mechanical work, while the modern successful processes abandon this idea. A perfect gas expanding so as to produce no energy will not suffer any change of temperature; but a gas under terrestrial conditions never expands without exerting energy with a corresponding fall in temperature. Engines to take up the energy of the expanding gas are useless. They inevitably increase the conductivity of the system, producing impaired efficiency. When dealing with liquids approximating the absolute zero in temperature, a high degree of heat insulation is the first requirement.

Imagine a tube terminating in a jet or nozzle and of sufficient length to have the portion most remote from the jet immersed in a cooling tank or other refrigerating appliance. Air is pumped into the tube by an air compressor capable of producing a pressure of 1,000 lb. per sq. in. or more. The size of the jet opening must be in proper proportion to the other parts of the apparatus, primarily the pump, in order to maintain this pressure within the tube. As the air is pumped into the tube it rises in pressure and consequently in temperature. The cooling apparatus should be powerful enough to allow the air to reach the

jet section at a constant temperature. The air rushing out of the jet expands from one volume to fifty volumes or even to three hundred volumes, and in doing this against the pressure of the atmosphere energy is exerted. This energy must be taken from that of the expanding air, which causes a fall of many degrees in temperature.

This method gives a purely mechanical means of reducing temperature. A portion of the tube, termed the jet section, extends beyond the cooling apparatus, and if it is treated so as to cause the air to reach the jet at a constantly decreasing temperature—down to some limit determined by practical conditions—a new element is introduced. The tube is of heat-conducting material and is inclosed within a vessel of poorly conducting material or sheathed with felt or otherwise protected; the air from the nozzle escapes into this vessel, fills it, cools the pipe and then escapes. The air from the cooler enters the last described portion or jet section (which we may term the liquefier) at a constant tempera-

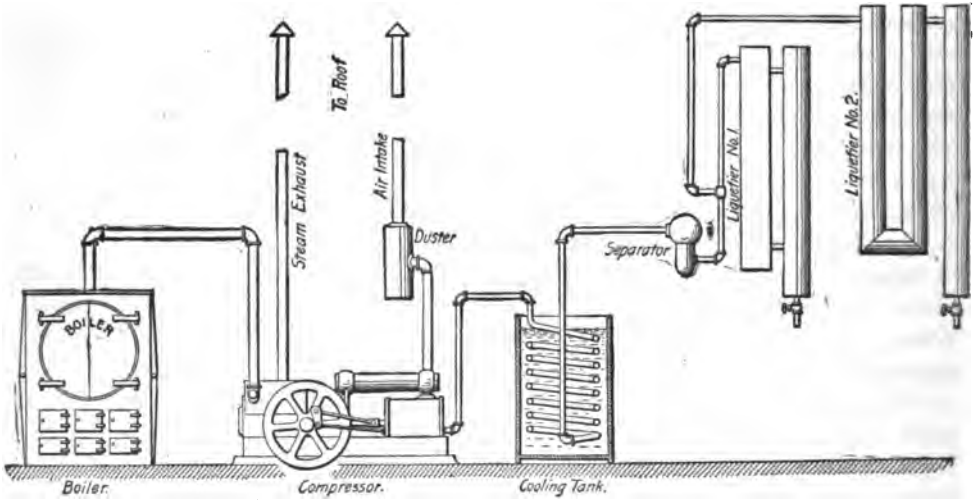


FIG. 1.—GENERAL ARRANGEMENT OF TRIPLER'S APPARATUS.

ture, but is constantly cooled therein to a lower temperature; and this intensive action continues until the reduction of temperature is so great that the air assumes the liquid state as it escapes from the jet.

It will be observed that liquefaction is brought about by reduction of temperature, not by increase of pressure, or to a relatively small extent by such increase. It is the temperature that is the controlling factor.

The pioneers in the economic production of liquid air by the purely mechanical process summarized above are Charles E. Tripler, of New York; Carl Linde, of Munich; William Hampson, of England, and Ostergren & Burger, now of New York. Charles E. Tripler for many years worked on the class of problems which includes the liquefaction of gases. The records of his early work are rather meager, as he did not keep a closely dated record of his investigations. In the year 1893 he secured patents in the British Patent Office, one of which embodies to some extent the self-intensive principle.

The Tripler patents are not yet issued, so that a full description of his apparatus cannot be given. Fig. 1 illustrates the general disposition of his apparatus. A powerful air compressing engine is used to compress the air to a pressure of from 2,000 to 2,500 lb. per sq. in. The air heated by the compression passes through a cooling apparatus, essentially a tubular condenser, which reduces it to a fixed temperature. Some water, oil or other impurity may be present and a trap or separator is provided to collect them. The compressed and cooled air freed from mechanical impurities, enters the liquefier. All that is visible of this is a long tube sheathed with non-conducting material with a stop-cock at its lower end. The general construction of the liquefier is as follows:

The condensed air is conducted in a sinuous course to a point near the bottom of the liquefier through long tubes or channels, which are surrounded by a cylindrical jacketed case, and are therefore quite concealed from view. At the lower end of the tubular system a jet is placed through which the air issues into the space between the tubes and the casing. The sudden expansion greatly reduces the temperature of the air. An arrangement of partitions, or divisions, directs the course of the expanded and cooled air in close proximity to the tubes, and in a general sense in the opposite direction to that followed by the air within the tubes. The air escapes from the liquefier at the top close to the place where the air originally enters. This arrangement brings about self-intensive cooling. The air enters the liquefier at a constant temperature, but escapes from the jet at a temperature progressively reduced within limits fixed by the practical elements of the design and construction of the apparatus. After a sufficient interval the temperature of the air as it escapes from the jet, falls below the critical point, and liquid air collects in the bottom of the case of the liquefier, from which it will flow like water, on opening the cock.

William Hampson's apparatus is illustrated in Figs. 2 and 3. His British patent is dated May 23, 1895, and his specification is as follows: "Never before has a refrigerating apparatus been designed with a view to the transfer (unavoidable leakage excepted) of all the cold of the expanded gas to the compressed gas which is going to be expanded." This statement expresses most clearly the idea of self-intensive or regenerative action, and in view of the substantial identity of the apparatus of the three inventors now under consideration, this sentence may be taken as an exponent of the action aimed at by the mechanism of each, even if not contemplated in the mind of each inventor.

The date of the Hampson patent is about one month earlier than that of the Linde patent. It is claimed that Linde's apparatus was shown in operation in May, 1895.

The Hampson apparatus contemplates the supply of compressed air or gas at a constant or even diminishing temperature by any convenient system—the apparatus proper includes only the liquefier and its construction is clearly shown in the two illustrations.

A cylindrical case is divided into annular divisions, within which is coiled a long metallic pipe winding up and down within them and terminating in a jet at the bottom of a downwardly projecting central extension of the apparatus.

Parallel with the coiled tube is a set of partitions which follows its course through the annular divisions. This disposition, clearly shown in the illustrations, has this effect—it virtually incloses the original tube within a second tube of square section. The air which escapes from the jet, cooled by its expansion, passes through the second square tube surrounding the inner tube and cooling it. The entering compressed gas in the inner tube and the escaping expanded gas in the outer tube follow opposite courses, so that theoretically a perfect arrangement for

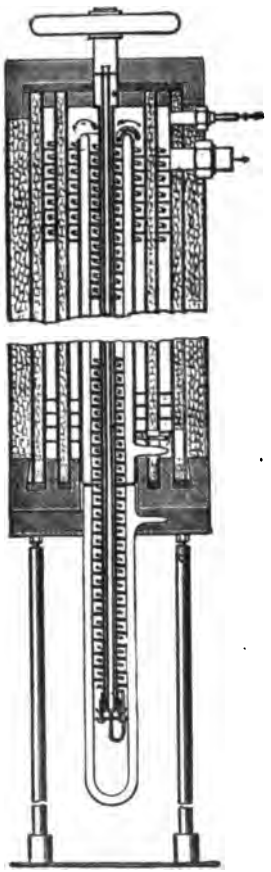


FIG. 2.—INTERNAL ARRANGEMENT OF COIL.

FIG. 3.—JET, REGULATING APPARATUS AND REGENERATIVE COIL.

HAMPSON'S GAS LIQUEFACTION APPARATUS.

temperature interchange is brought about, the coldest portions of the system being in the center.

The action is identical with that described for the Tripler apparatus. The liquefied air collects in a chamber at the lower end of the downwardly projecting extension, and is withdrawn as required.

A pressure of 1,200 lb. per sq. in. or more is required. A 3·5 h. p. compressor produces about 1·2 quarts of liquid air in an hour. At higher pressure,

1,400 to 1,600 lb. per sq. in., liquefaction begins in from 10 to 16 minutes. Instead of the pumping apparatus cylinders of compressed gas are sometimes used to furnish the supply. The gas or air which has escaped liquefaction flows out into the atmosphere. There is no closed cycle as is presented in Linde's apparatus.

The construction of the jet and its regulating apparatus is characteristic. The jet is directed upward and its capacity is adjusted by a flat or recessed diaphragm moved as desired by a screw worked from the exterior of the apparatus. This is clearly shown in Fig. 3.

Carl Linde's apparatus dates back to 1895. It has been given various forms, but the general principle is identical with that just described. Two concentric tubes are bent into a spiral, and air under heavy pressure is pumped into the inner one which is provided at its further end with a nozzle whence the air

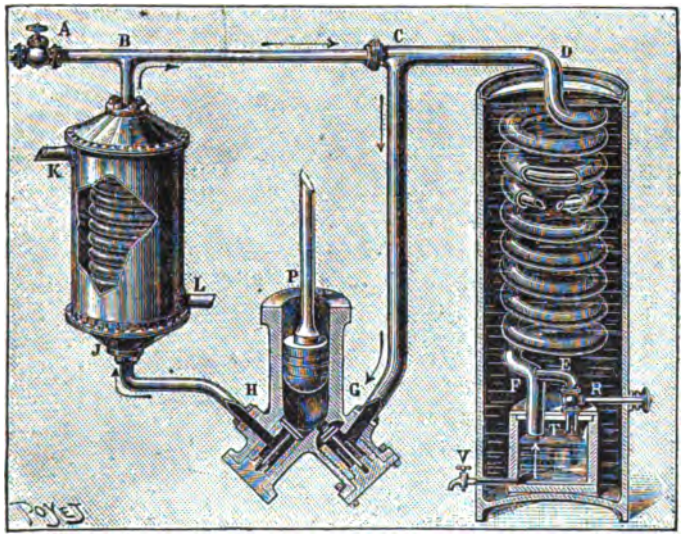


FIG. 4.—LINDE'S APPARATUS FOR LIQUEFYING AIR.

escapes and enters the outer pipe. Through this it returns in an opposite direction to the entering air and thus carries out the self-intensive cooling principle. The compressed air is cooled by a water-cooler so that it will enter the apparatus at a fixed temperature. The tubes are contained in a jacketed case.

For some reason, Linde claimed that his apparatus utilized the Joule-Thomson effect. This somewhat recondite principle may be thus illustrated: Assume two vessels connected by a tube with a stop-cock. Let one be filled with a gas at high pressure and the other be empty. Imagine them immersed in water and the stop-cock opened. The gas will flow through it until the pressure is the same in both vessels. In one vessel the temperature will fall as energy has been expended by its contents in forcing their way into the other globe vessel, while in the other, upon whose contents energy has been expended, the temperature will rise. If the gas operated on is a perfect gas, the water in which the vessels are

immersed will suffer no change of temperature, because the cooling and heating actions will exactly counterbalance each other. On trial with various gases no change of temperature in the water could be discovered, but more delicate observations carried on by Joule and Sir William Thomson (now Lord Kelvin) showed that with many gases a slight decrease in the temperature of the whole took place. With hydrogen, a slight increase takes place. In other words, the majority of gases are not perfect gases, while hydrogen is super-perfect. This cooling action is very small in amount compared with the general cooling due to expansion, yet Linde based his theory of operation on it, and complete calculations are given to show that it is sufficient in amount to produce liquefaction. There are reasons why it would be of some importance to Linde to claim that he is the first to utilize this principle, as it gives him a priority in the enunciation of a theory at least.

Fig. 4 shows the general features of construction of the Linde apparatus. In the center is a pump *P*, which draws air through the inlet *G*, and compresses it. The air then passes out through *H*, and loses the heat due to compression in the cooler seen on the left, through which water runs, entering and leaving by the pipes *K* and *L*, the compressed gas entering at *J*. The compressed gas follows the course indicated by the arrows. The pipe carrying the gas enters at *C*, the concentric larger pipe, and the double pipe thus constituted is bent into a coil as shown on the right side of the illustration. Part of the coil is cut away in the figure to show the interior disposition. The gas-conveying pipe finally emerges from the larger pipe at *E* and enters a vessel, the valve *R* regulating the flow. The compressed gas escapes through the lower end of the coil within this vessel, cools as it expands and passes up through the outer concentric pipe *F*, filling the annular space between the two pipes and cooling the gas within the smaller pipe. Thence it goes to the pump through *G*. In this manner the self-intensive action is established and liquefaction is produced. To supply the gas removed from the apparatus by liquefaction, fresh air or gas is pumped in as required through *A*. The liquefied air is withdrawn through the valve *V* and the idea of a closed cycle is carried out.

This form is typical, but Linde has sometimes expanded part of the air by steps or partial expansions. There is no essential difference brought about by this change.

One of the finest plants for the manufacture of liquid air is that erected under the designs of Ostergren & Burger, in New York, whose process is largely along the lines of those described. Self-insulation is a very prominent feature in their invention. By using properly disposed channels for the interchange of heat along the lines described for the Hampson processes, the colder portions of the conduits are in the center and are progressively surrounded by those nearer the atmospheric temperatures, so that the outer portions of the coils are of nearly atmospheric temperature while liquid air collects from the inner portions. They use a low pressure for liquefaction and their apparatus is very efficient; as now installed it produces liquid air in very large quantities. Their demonstration may rank among the best yet installed, and they are recognized claimants for priority in the fundamental invention of self-intensive liquefaction.

Among the principal investigators besides Pictet, Cailletet and others mentioned above are James Dewar, Sigmund von Wroblewski, and Karl Olszewski, constituting a most celebrated group of modern scientific investigators.

Uses of Liquid Air.—The uses of liquid air remain to be developed. It has proved hitherto rather disappointing in regard to its practical applications. A brief statement will cover this aspect of the question.

When liquid air evaporates, the nitrogen goes off in much larger quantity than the oxygen, so that on standing the liquid increases in its proportion of oxygen. This offers a method for producing oxygen, not of absolute purity but of commercial value, 60 to 98% O. Raoul Pictet is at present engaged in installing a plant for producing commercial oxygen in this manner.

If liquid air, especially that which has been enriched by standing, is mixed with finely divided or porous carbonaceous matter, an efficient explosive is produced. Linde has had extensive experiments made on this explosive and good results were attained. It must, however, be exploded within a few minutes of the time of mixing—an apparent objection, offset by the fact that no danger exists of a cartridge exploding after missing fire the first time. In this respect it is the safest conceivable form of explosive.

Many treatments in medical cases have been carried out with liquid air. It affords a powerful cauterizing agent, the effect of which entirely disappears after the treatment. Its use is proposed not only for simple cases where cauterization is required, but also for the treatment of cancer.

It has been proposed seriously to use it for reducing the temperature of electric conductors. As the temperature is lowered, copper becomes a better conductor of electricity and by carrying a copper electric main through a pipe kept full of liquid air, the wire would conduct a much greater current for equal drop of potential than if at ordinary temperatures.

Another use proposed for liquid air is refrigeration. In the case of fruit, the great difficulty in applying ice refrigeration is the presence of moisture, and in transportation the weight of the ice is also a great objection. By the use of liquid air as a refrigerant weight is saved, and no moisture is introduced. The question of making practical success of liquid air as a refrigerant on a large scale is yet to be solved. The prospect of utilizing liquid air for power is not very favorable. It provides a fluid of easily regulated evaporation and capable of developing enormous pressures, but with its practical efficiency greatly reduced by the low temperature produced in its expansion. It can develop a pressure of about 12,000 lb. per sp. in., about one-half that developed in the firing of a cannon.

The use of liquid air for cooling the air in the working places in very deep mines has been suggested editorially in the *Engineering and Mining Journal*, September 16, 1899, and it may possibly permit the working of mines at much greater depths than have heretofore been considered possible.

The great problem always anticipated with liquid air was how to keep it. But the great readiness with which it assumes the spheroidal state makes its transportation easy. It is carried long distances in tin buckets double walled and packed with boiler felting or other non-conducting material.

REVIEW OF THE LITERATURE ON ORE DRESSING IN 1899.

By Robert H. Richards.

No radical departure has taken place during the past year, but any changes that have come have been more in the nature of perfections and extensions of existing methods.

Richards' Classifier.—A special form of this apparatus was introduced at Longdale, Va., where limonite ore was treated by a log washer. The tailings, which were formerly allowed to run to waste, were sent, with the exception of the very finest, to the classifier. During a seven-day's run the apparatus handled

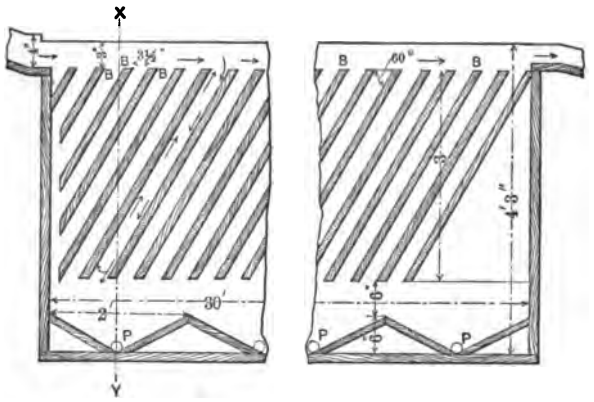


Fig. 1.—Longitudinal Section.

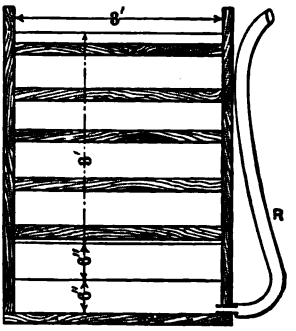


Fig. 2.—Transverse Section on X. Y.

Baffling Board Settling Tank for Fine Slimes used at Cocheño, Mexico.

51·5 tons of stuff, not over 5% of which was of larger diameter than 1·5 mm., and less than 10% smaller than 0·159 mm. The feed contained 41·8% Fe, while the spigot product, which amounted to 26·19 tons per 24 hours, contained 45·56% Fe, and was worth $3·45 per ton. The tailings contained 38·47% Fe. The classifier on this work required 377 lb. of water per minute with the feed and 460 lb. of hydraulic water, of which latter amount 169 lb. went with the concentrates out of the spigot, while the balance rejoined the feed water.

A baffle board settling tank is used at Cocheño, Mexico,* for the settling of

* Courtenay De Kalb, private communication.

fine slimes. Its form is shown by Figs. 1 and 2. There are three of these in use. The feed contains on an average 4·67% solid material, the overflow averages 0·64% solid, and the settlings 21·91% solid material. The surface velocity of the current in the tank is 7·5 ft. per second. Formerly instead of these three tanks, settling boxes and five settling ponds, each about an acre in extent, were used. The circulation was always through two of these ponds, and sometimes through three. Treating the same material as above (4·67% solid) these ponds gave an overflow which averaged 0·7% solid material, and the settlings pumped out carried 20% solid.

The tank is not adapted to slimes carrying more than 60% granular matter, of which the maximum grains must not be more than 0·08 mm. diameter. The effectiveness of the apparatus appears to be due to the fact that grains have only to settle 0·25 in. in a length of 30 ft. before they are caught by the currents moving slowly between the baffle boards as indicated by the arrows.

Hooper's vanning jig used in this country to treat garnet is a movable sieve jig 4 ft. 2·5 in.×2 ft., which has the tail end of its jig box suspended from a pivot by a vertical rod with rigid connection to the box, and of adjustable length. The head end is given an approximately up and down motion by a vertical connecting rod leading to an eccentric, while the tail end has a motion very nearly horizontal. The screen extends only to within 14·5 in. of the tail end, being replaced by a solid bottom. The vanning motion is such, and the adjustments can be so made that the stuff fed at the middle forms heads which travel continuously to the head end, and tailings which pass over the other end. It is applicable to difficult jigging problems on stuff from 0·25 in. to 30-mesh, and can treat 5 to 7 tons per 24 hours. The average slope of the screen is 2·5°.

Shaking Tables.—Several forms of these have been designed on the same principle as the Wilfley table (THE MINERAL INDUSTRY, Vol. VI.). The Cammett table made by the Denver Engineering Works has the riffles extending the entire length, and has a lengthwise, as well as a sidewise, adjustment of the slope. The wash water is introduced by three or four individual spray pipes. The surface is of redwood, specially painted, and the riffles are cut directly upon the surface of the table.

On the Hallett table the riffles extend parallel to a diagonal drawn from the tailings side to the concentrates end. The surface is of lead. An adjustable end as well as side slope is used, and the wash water is small in amount and fed over a distributor.

The Bartlett table made by the Colorado Iron Works, has three decks arranged in step form, with riffles extending the whole length. The whole of the material is treated on the upper step with little water, the tailings of this are re-treated on the second step with more water, and the tailings of the second step are re-treated on the lower step with still more water.

The Woodbury Imperial Concentrating Table, made by G. E. Woodbury, derives its motion from a cam and spring.

The use of tables of the Wilfley type is continually on the increase, and they have as a rule proved very successful.*

* See "Ore Dressing," elsewhere in this volume.

Laboratory Experiments on Concentration after Roasting.—These tests were made by F. H. Mason.* The ore contained galena and blende in a metamorphic schist (sp. gr. 2·5) with some oxides and sulphides of iron and some chalcopyrite and arsenopyrites. The best result of wet concentration on raw ore was 33% Pb, in the concentrates, while the average was not over 25%. After roasting at a dull red heat for an hour, concentration gave a product, running 68·24% Pb, 6·66 oz. Ag, 0·25 oz. Au from an ore containing 19% Pb, 17·23 oz. Ag and 0·56 oz. Au per ton. Roasting should be stopped while the galena and blende are practically unchanged, but leaving the arsenopyrite and pyrrhotite oxidized and porous. It may be difficult to accomplish these results when working on a commercial basis.

Concentration of Black Phosphate.—David Levat† crushed black phosphate rock through an 80-mesh sieve, and treated it by levigation. Before treatment the ore contained 9·76% P_2O_5, after treatment the residue amounted to 60%, and contained 6·10% P_2O_5. The remaining 40%, at that rate, should show a concentration to 15·24% P_2O_5, and could be saved by settling.

Slime Tables.—A comparison of different circular slime tables has been given by Hübner.‡ The bump of the Bartsch table is advantageous only for coarse pulp (0·5 to 0·25 mm.). Ordinarily it has somewhat greater capacity, but produces higher tailings than when run without the bump.

The concave round table has several advantages over the convex. First of all the pulp is more easily and uniformly distributed, and the pulp distributor is easily accessible, and easily kept clean, while the openings in the pulp distributor of a fixed convex table are easily stopped up. This difficulty does not occur in a revolving convex table.

In the four government dressing works in the Harz there are 32 simple concave tables, 29 combination (concave followed by convex) and only 6 convex tables; but in the private plants of Upper Silesia the concave tables are not common.

General Milling Considerations.—Needed reforms in metallurgical standards have been discussed by F. T. Snyder.§ He advocates: (1) The adoption of a standard ton, preferably the metric ton. (2) All weights to be "dry" weights. (3) Abandonment of the estimation of weights from volumes. (4) Abandonment of the method of designating screens by "mesh." He makes a new point that in double-crimped screens the maximum opening is not at right angles to the plane of the screen. (5) The use of a micrometer caliper with very large pole pieces to get the actual size of grains in fine products. (6) A standard method of plotting curves of sizing tests. (7) The calculation of the relative surface in all sizing tests, as affording a better means of comparison than the mere percentages of different sizes.

The agitation against the system of designating screens by meshes, and for the abolition of wire gauges still continues.‖ It is refreshing to note that one

* *Journal of the Mining Society of Nova Scotia*, Vol. V., Part I., 1899–1900, p. 41.
† *Annales des Mines*, Series IX., Vol. XV., 1899, p. 5.
‡ *Zeitschrift für Berg- Hütten und Salinenwesen*, Vol. XLVII., 1899, p. 239.
§ *Canadian Mining Institute*, Vol. II., 1899, p. 3.
‖ *Iron Trade Review*, March 23, p. 12, and March 30, p. 16, 1899.

of the largest wire cloth screen manufacturers in this country, the W. S. Tyler Co., has given the actual size of hole in their screens in their latest catalogue.

MAGNETIC CONCENTRATION.

A new magnetic separator, patented by O. A. P. Trustedt,* has no movable parts whatever, but while the ore is being carried down over a slope by a current of water, the magnetic particles are gradually worked to one side by a set of alternating magnets underneath which act on a principle similar to that of a polyphase motor.

Magnetic separation especially in Sweden is the subject of a very full article by H. C. McNeill.† The Wenstrom magnetic separator which he describes is a simple machine, especially adapted for roughing work. It can treat ore of fairly large size (up to 4 in.), and not necessarily although better dry. A machine with drum 2 ft. 2·5 in. diameter, and 2 ft. long treats 5 tons per hour. At Grangesberg the ore which is magnetite in a gangue of quartz, pegmatite, schistose material, etc., goes to: (1) Grizzly with 5 in. spaces. Oversize is hand picked into concentrates and wastes. Undersize to 2. (2) Three shaking screens, one above another with holes 1·25, 0·75 and 0·25 in. respectively. A stream of water also washes the stuff while on the screens. Material through 5 on 1·25 in. goes to 3; that through 1·25 on 0·75 in., and through 0·75 on 0·25 in. go together to 5; that through 0·25 in. goes to draining pits and thence to market. (3) Trommel with 1·5 in. holes. Oversize to 4. Undersize to 5. (4) Large Wenstrom separator. It yields concentrates which go to market and tailings which are hand picked yielding concentrates to market (pieces too large to be held by the machine) and tailings to waste. (5) From 2 and 3. Small Wenstrom separator. It yields concentrates and waste.

The original ore runs 38 to 42% Fe, and the concentrates 58 to 62% Fe.

The Monarch separator (improved Ball & Norton) is described. For this the ore must be dry and best below 1 mm. It is especially applicable where the magnetite is more or less associated with gangue, or to ores where the magnetite is of secondary importance. At Luleo in the north of Sweden the ore contains 0·01 to 3% P in the form of apatite, and is correspondingly low in iron. Only hand selected ore containing the most apatite comes to the mill to 1.

(1) Blake breaker to 2. (2) Rolls crushing to pass through a 0·5 in. screen to 3. (3) Revolving cylindrical dryer 48 ft. long, 4 ft. 8 in. diameter sloping 1 in 18, and rotating once in 5 seconds. By chute to 4. (4) No. 1 rolls to 5. (5) No. 1 trommel with oblong holes. Oversize (80% of total) to 6. Undersize to 9. (6) From 5 and 8 No. 2 rolls to 7. (7) No. 2 trommel. Like No. 1. Oversize to 8. Undersize to 9. (8) Elevator to 6. (9) From 5 and 7. Elevator to 10. (10) Monarch separator. It yields concentrates, tailings and middlings. The last are re-treated on a second Monarch separator. All the screens, rolls, separators, elevators, chutes, etc., are boxed in with wood, and the dust is drawn off by exhaust fan to a separate house, where it is allowed to settle.

* *Engineering and Mining Journal*, April 29. 1899. p. 503.

† *Journal of the Iron and Steel Institute*, Vol. LVI., 1899. p. 18.

and removed at intervals for further treatment. In this plant it is the tailings that are the most valuable product. They are further treated for the separation of small residual iron compounds, and are finally converted into soluble phosphate of soda by the Wiborgh process.

The Dellvik-Gröndal separator is used for finely disseminated ores, and especially to treat slimes. It has been used at Pitkaranta, in Russian Finland, since 1894, for concentrating lean iron ores.

The Heberli separator is made in two forms, the first for separation and concentration of low grade magnetic ores only; the second is for the treatment of galena blende ores when occurring, with magnetic oxide of iron, also for the separation of roasted spathic iron ore and blende, etc., or in the treatment of a complex ore consisting of galena, blende, magnetic oxide, pyrrhotite and quartz. Full details and illustrations of these separators are given in the article referred to above.

At Laxburget, in Sweden, an ore containing approximately 11% Pb, 22% Zn, 14% Fe_3O_4, 15 to 20% SiO_2, and 2 to 5% pyrrhotite, is first cobbed and picked over to roughly pieces of 6 in. diameter, and then goes to 1: (1) Blake breaker crushing to 0·75 in. Elevated to 2. (2) No. 1 trommel with 0·375 holes. Oversize to 3. Undersize to 5. (3) Coarse rolls to 4. (4) No. 2 trommel with 0·03125 in. holes. Oversize to 5. Undersize to 7. (5) From 2, 4 and 6 fine rolls to 6. (6) No. 3 trommel with 0·03125 in. holes. Oversize to 5. Undersize to 7. (7) From 4 and 6. Heberli separator of the second form. It yields concentrates and tailings. The latter are separated into coarse and fine. Coarse tailings to 8. Fine tailings consisting mostly of blende and silica with 3% magnetite go to settling pits for subsequent treatment as fine slimes. (8) A 4-compartment jig. It yields concentrates (galena) and tailings. The latter are re-ground in a Heberli mill and concentrated by pointed boxes, continuous revolving buddles, Rittinger tables, etc.

*The magnetic concentration plant at Pitkaranta, Finland,** treats magnetite which occurs in large quantities, but of low grade and mixed with sulphides of copper and zinc, and finely disseminated. The ore seldom contains over 25% Fe and often below 20%.

At the New plant, which has a capacity of 350 tons a day, rolls, which gave unsatisfactory results both technically and economically, and Gruson mills, which at first gave good results, though latterly these were not satisfactory, were replaced by continuously fed and discharging tube mills. These have the advantage over the Gruson mill in producing greater product with the same power, reducing the working costs one-half, and that water may be used avoiding the formation of dust.

There are four of these, each 1·75 in. diameter, and 0·8 in. long, and each grinds 50 tons of medium ore per 24 hours, using 8 h. p. There are four more of 2 in. diameter, and 1 in. long, each of which grinds 30 tons of hard ore per 24 hours. They crush to 1 or 0·5 mm. The wear of balls and ring is 15 kg. per 100 tons of ore. For the magnetic separation a Dellvik-Gröndal separator

* *Oesterreichische Zeitschrift*, 1899, p. 51. *Stahl und Eisen*, 1899, p. 271.

described above is used. It requires a half h. p. to drive it, and uses 8 ampères with 35 volts for its magnets. In most other machines a middling product is formed, and the tailings carry 6 to 10% Fe, but with this machine there are no middlings, and it is claimed that the tailings carry only 1 or 1·5% magnetite. The concentrates contain 65 to 71% Fe, whether the feed runs 18 or 40% Fe. They are briquetted.

With a water power of 125 to 150 h. p., the cost of erecting a plant for the yearly treatment of 120,000 to 150,000 tons was as follows:* Two rock breakers (12 h. p.) 2,900 fl.; two elevators (4 h. p.) 1,600 fl.; eight tube mills (96 to 120 h. p.) 20,900 fl.; eight separators (4 h. p.) 15,800 fl.; dynamo and electrical apparatus (4 h. p.) 1,600 fl.; pump for about 600 liters per minute (4 h. p.) 650 fl.; transmission and belting, 5,200 fl.; building, 6,500 fl.; installing, etc., 9,850 fl.; total, 65,000 fl., or about $32,000. The total h. p. is 124 to 148. The running cost, including transportation to mill (presumably per year) is: Maintaining crushing machinery, 11,000 fl.; maintaining conveying machinery, 1,300 fl.; maintaining building, 650 fl.; wages, 6,500 fl.; administration and sundries, 13,000 fl.; total, 32,450 fl., or about $16,000. This makes the cost per ton about 26 kreutzers, or about 13c. Where hard ore is treated the capacity is reduced to 7,500 tons per year, and the cost per ton becomes 46 kreutzers, or about 23c. If steam power were used the costs per ton in each case would become 32·5 and 65 kreutzers respectively.

The Waugh and Bignell dry concentrator† is much like a Frue vanner, run without water, except that a belt of canton or cotton flannel is used, and air is forced up through its top side. Concentrates are carried up to the head while tailings pass off at the tail. One mill uses them at Lewiston, Wyo., two or three in the Black Hills, S. D., and one at Denver, Colo.

New Coal Washing Plant at the Blanzy Colliery, Saône et Loire, France.‡ The capacity is 800 tons in 10 hours.

(1) Storage bins delivering ore to 2.

(2) Bucket chain elevator lifting 22 m. (72 ft.) to 3.

(3) No. 1 trommel. It is run dry and makes: Through 60 on 45 mm. (2·1875 to 1·2031 in.) to No. 1 jig, 5; through 45 on 25 mm. (1·2031 to 1 in.) to No. 2 jig, 6; through 25 on 10 mm. (1 to 0·2031 in.) to No. 3 jig, 7; through 10 mm. (1 in. to 0) to 4.

(4) No. 2 trommel. It makes: Through 10 on 7 mm. to No. 4 jig, 8; through 7 on 3 mm. to No. 5 jig, 9; through 3 mm. to No. 6 jig, 10.

(5) From 3. No. 1 jig. It has two compartments with a 7 mm. sieve on the first compartment. It yields, first compartment hutch through 7 mm. to 11, second hutch 60 to 45 mm. to 15 mm. to 15, and tailings (coal) to 19.

(6) From 3. No. 2 jig having two compartments with a 6 mm. sieve on the first compartment. It yields first compartment hutch through 6 mm. to 11, second compartment hutch 45 to 25 mm. to 15, and tailings (coal) to either 18 or 20.

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* The florin equals $0·48.

† *Engineering and Mining Journal*, Oct. 14, 1899, p. 453.

‡ *Comptes Rendus Soc. Ind. Min.*, 1898, p. 204.

(7) From 3. No. 3 jig, a two compartment jig with 3 mm. sieve on first compartment. It yields first compartment hutch through 3 mm. to 11, second compartment hutch 25 to 10 mm. to 15, and tailings (coal) to 21.

(8) From 4. No. 4 jig, a two compartment jig with a 4 mm. sieve on the first compartment. It yields first, compartment hutch through 4 mm. to 11, second compartment hutch 10 to 7 mm. to 11, and tailings (coal) to 22.

(9) From 4. No. 5 jig, a two compartment double jig with a 4 mm. sieve on the first compartment. It yields first compartment hutch through 4 mm. to 11, second compartment hutch 7 to 3 mm. to 11, and tailings (coal) to 22.

(10) From 4. No. 6 jig—two 2-compartment double jigs with a 6 mm. sieve on the first compartment. It yields first compartment hutch 3 to 0 mm. to 11, second compartment hutch 3 to 0 mm. to 11, and tailings (coal) to 22.

(11) From 5, 6, 7, 8, 9, 10. A bucket elevator for "fine shales" lifting 19 m. (62 ft.) to 12.

(12) No. 3 trommel with 5 mm. (0·1875 in.) holes. It yields oversize to 13 and undersize to 14.

(13) No. 7 jig, a two compartment jig. It yields hutch products to 26, and tailings (coal) to 22.

(14) From 12. No. 8 jig, a two compartment jig. It yields hutch products to 26, and tailings (coal) to 22 and 24.

(15) From 5, 6, 7, 8. A bucket elevator for "coaly shales" lifting to 16.

(16) No. 4 trommel with 25 mm. (1 in.) holes. It yields oversize to 17, and undersize to 18.

(17) No. 9 jig. It yields hutch to 26, and tailings (coal) to 24 or 25.

(18) From 16. No. 10 jig. It yields hutch to 26, and tailings (coal) to 25.

(19) From 5 and 6. Draining tower or store bin for "coarse coals." The coal is first rinsed on a shaking screen before falling into the bin. The rinsings go to 29.

(20) From 6. Draining tower or store bin similar to 19. The rinsings go to 29.

(21) From 7. Draining tower or store bin for "fine coals," similar to 19. The rinsings go to 29.

(22) From 8, 9, 10, 13, 14. Settling tank and bucket elevator lifting 22 m. (72 ft.) to 23. Overflow to 28.

(23) From 22 and 29. Seven draining towers or store bins for "washed slack." They have sieves at the bottom to drain them when full. The drainings go to 29.

(24) From 14 and 17. Draining tower or store bin for "mixed coals." The drainings go to 29.

(25) From 17 and 18. Draining tower for "second class coarse coals." The drainings go to 29.

(26) From 13, 14, 17, 18. Bucket elevator delivering to 27.

(27) Draining tower or store bin for "definite schists" which go to waste. The drainings go to 29.

(28) From 22. A box with 8 hoppers. The spigots go to 29, and no mention is made of any overflow.

(29) From 19, 20, 21, 23, 24, 25, 27, 28. A pulsometer pump lifting 20 m. (65·75 ft.), and delivering into the launders leading to 23.

(30) Water. One large centrifugal pump at a level of 5 m. (16·33 ft.) fed from the upper portion of the large storage bin or at a level of 8 m. (26 ft.) delivers 12 cu. m. (2,640 gal.) per minute to a level of 16 m. for the jigs. There is also a smaller centrifugal pump drawing from the delivery of the large one at a level of 16 m., and raising 1·5 cu. m. per minute to a level of 22 m. (72 ft.) for supplying No. 2 trommel, and the conveying water for the launders.

The estimated quantity of water to be renewed per minute is 2·5 hectoliters (55 gal.).

Other details in the shape of sizing tests, details of jigs, etc., are given.

Treatment of Culm.—At the washer working on the Reynolds culm bank in the borough of Plymouth, Luzerne County, Pa.,* the culm first goes to a bar screen, the oversize of which goes to rolls, and is then jigged forming coal and waste, while the undersize goes to a set of circular screens which yield: (*a*) Chestnut—jigged forming coal and waste; (*b*) Pea—jigged forming coal and waste; (*c*) Rice—shipped; (*d*) Buckwheat—shipped; (*e*) fine coal and dirt which is run into the mine to fill old chambers in order to remove pillars later. The results from 1890 to July, 1898, are as follows:

Cubic yards of culm washed, 382,000; nut coal made 3,145 tons, or 2·6%; pea coal made 13,649 tons, or 11·5%; buckwheat coal made 78,580 tons, or 66·0%; rice coal made 23,491 tons, or 19·9%; total, 118,865 tons, or 100·0%.

One cubic yard of culm yields 0·008 tons of nut coal, 0·036 tons of pea, 0·205 tons of buckwheat, and 0·062 tons of rice, making a total of 0·311 tons.

* *Engineering and Mining Journal*, April 15, 1899, p. 440.

PROGRESS OF METALLOGRAPHY IN 1899.

By Albert Sauveur.

The activity in this department of metallurgy, to which I alluded a year ago, has continued unabated during 1899. Many new recruits have joined the ranks of those engaged in metallographic researches and the industrial and technical world has shown a continuously increasing interest in their results and possibilities.

In the following pages I have endeavored to present in a concise, still comprehensive manner, the most important additions to our knowledge of the structure and properties of metals for which we are indebted to metallographic methods of inquiry.

Methods of Revealing the Microstructure of Metals and Alloys.—Osmond and Cartaud* have made a systematic research into the relative value of a number of reagents for the development of the microstructure of iron and steel, reaching the conclusion that the combined polishing and etching method with a dilute solution of ammonium nitrate yields the best results. A piece of parchment spread tightly over a smooth board is soaked with a dilute solution of ammonium nitrate (two parts in weight of the crystallized salt to 100 parts of water), and the previously polished surface of the specimen is rubbed upon it until sufficiently etched.

Ewing and Rosenhain† have obtained some beautiful metallic surfaces suitable for microscopical examination without further polishing or etching, by casting the metal or alloy upon a very smooth surface, as plate glass or polished steel. Gold, bismuth, cadmium, lead, tin, zinc and the fusible alloys, yielded in this way some very fine preparations.

The method cannot be applied, of course, to metals which oxidize below their melting points.

Moreover, in the case of lead, some surfaces were obtained, which needed no further treatment to reveal the structure, by heavy pressure of a freshly cut section against a smooth surface.

The Microstructure of Pure Metals.—I called attention in The Mineral

* *The Metallographist*, Vol. III., No. 1, p. 1.

† *Philosophical Transactions of the Royal Society of London*, Vol. CXCIII., p. 353; *The Metallographist*, Vol. III., p. 94.

INDUSTRY, Vol. VII., to the remarkable work of J. E. Stead by which he had shown conclusively that the polyhedric grains seen in the structure of iron or soft steel, are not true crystals, but are made up of small cubic crystals, always oriented in the same direction in the same grains, but differently in adjacent grains. To this fact is due, as was then explained, the brightness of some grains and the dark appearance of others, as well as the kaleidoscopic effect produced in rotating the specimen under examination.

Ewing and Rosenhain have now shown us that such structure is common to all metals, at least as far as their experience go, and they have examined a large number of metals. All metals are made up of crystalline grains, with irregular polyhedric junction lines, each grain being made up of small crystals similarly oriented in the same grain but differently in different grains. All such structures exhibit the kaleidoscopic effect described in the case of iron.

Fig. 1 shows, under a magnification of 1,000 diameters, the structure of cadmium cast upon plate glass. Air pits are formed which are negative crystals, and it will be noticed that they have the same geometrical form and, in the same grain, the same orientation.

The authors also give the following evident explanation of the fact that the junction lines between the crystalline grains are made visible by etching. "It is due to the differential action of the acid which has produced differences of level by attacking one grain more energetically than its neighbor. Each of the short sloping surfaces which connect one grain with another, appears black because it does not reflect the normally incident light into the tube."

Phosphides and Sulphides under the Microscope.—J. E. Stead, in a recent lecture before the Cleveland Institute of Engineers, in England, has described a method of devolving the structure of cast iron and steel which reveals at once the presence of phosphides and sulphides. The brightly polished piece of iron is heated until, to the naked eye, it assumes a purple color. When viewed under the microscope, the iron is then found to be colored light blue, the sulphides orange, the phosphides pale lemon yellow and the carbides a slaty blue.

Mr. Stead expects to present at the May meeting of the Iron and Steel Institute a comprehensive paper on this most important addition to our knowledge of the microstructure of iron and steel.

The Thermal Critical Points of Iron and Steel.—The critical points which occur during the cooling or the heating of a piece of iron or steel and which are accompanied by important and sudden changes in the physical, chemical and structural properties of the metal have been the object of several important researches during the past year.

To make the matter clear I shall briefly outline the condition of our knowledge of the phenomenon previous to these recent additions.

Gore and Barrett called attention long ago to the fact that when a piece of steel is heated to a high temperature and allowed to cool slowly, the cooling proceeds at first at a uniformly retarded rate until a temperature of about 700°C. is reached when there is a sudden retardation in the fall of the temperature, indicating an evolution of heat often so considerable as to cause a momentary stop in the cooling, or even an actual rise of the sensible tempera-

ture, a "recalescence" of the cooling metal. After this retardation the fall of temperature resumes its normal rate, which is continued until the surrounding temperature is reached.

In heating, as might be expected, there is a reverse phenomenon, an absorption of heat causing a retardation in the rise of temperature. Although the two retardations do not occur at exactly the same temperature, the retardation on heating taking place at a temperature some 30°C. higher, they, undoubtedly indicate the opposite phases of the same phenomenon, for in order to induce one of the retardations, the opposite one must first be produced; just as in order to melt ice it is necessary first to freeze the water. The fact that one retardation lags behind the other is merely due to hysteresis, so frequently found at work in physical phenomena.

This critical point is commonly named "the point of recalescence." Brinell designated by the letter V, the retardation occurring during cooling and by W, the retardation during heating, while Tschernoff called these two retardations respectively *a* and *b*. Finally Osmond, whose researches have been so fruitful, designated by Ar the retardation during cooling and by Ac the opposite transformation taking place during heating, and his notation has now been quite universally adopted.

These sudden changes in the rate of cooling or of heating are of course the manifestation of some important transformations taking place spontaneously within the metal when the critical temperature is reached. Many of the properties of the steel undergo during the retardations deep changes.

Designating by "critical range" the two critical points considered collectively, we find that above the critical range the carbon exists in a condition different from its condition below that range, it being present as hardening carbon (soluble in cold dilute acids) above the critical range and as cement carbon (insoluble in cold dilute acids) below that range. If the steel be quenched immediately after it has been heated past the critical range it will be fully hardened, while if it be quenched from a temperature below that range, however near it, it remains unhardened. It is during the retardations that steel acquires and loses its hardening power.

By quenching steel, moreover, from a temperature above the critical range, no retardation occurs during cooling, from which it is rightly argued that certain conditions which are stable at a high temperature are retained by sudden cooling, being denied the time required for their transformation, and to this retention the various theories of hardening attribute the hardening of steel. It is only in regard to the nature of what is thus preserved that they differ.

Above the critical range the metal is non-magnetic, while below the range it is magnetic. The electrical resistance of steel increased abruptly during the critical point on heating, being ten-fold what it should have been, had the increase remained normal. Above the critical range, steel is made up of a single constituent (martensite), and during the retardation on cooling this constituent breaks up into two components (ferrite and pearlyte in the case of steel containing less than 0·80% of carbon, cementite and pearlyte in the case of more highly carburetted samples).

Osmond discovered that when the steel contains a small percentage of carbon, say less than 0·40%, a retardation can be detected both during heating and cooling at a temperature higher than that of the critical point just described, while if the steel contains less than 0·25% of carbon, a third critical point is detected at a still higher temperature. In electrolytic iron or in iron very free from carbon, moreover, only these two upper retardations occur, which might have been anticipated, seeing that the lower critical point is closely related to a change in the condition of the carbon.

The upper retardations occurring in carbonless iron and in low carbon steel have been designated by Osmond by the notations Ar_3 and Ar_2, the third retardation being then designated by the symbol Ar_1. The evolution of heat corresponding to the critical point Ar_2 occurs generally between 700 and 750°C., and takes place progressively while the point Ar_3 is sudden and occurs at about 860°C.

It has been conclusively shown that the retardation Ar_2 is closely identified with the appearance of magnetism during cooling, while the point Ar_3 is identified with a sudden increase in the electrical resistance of the steel.

The critical point in steel possessing only one such point is then in reality a triple point which is better designated by the symbol $Ar_{3,2,1}$ indicating that the transformations which in very soft steel occur separately during the critical points Ar_3, Ar_2 and Ar_1, here occur simultaneously at the temperature of recalescence.

Finally a fourth critical point has been first noted by Ball in iron and in very soft steel at a temperature of about 1,200°C., the meaning of which has not yet been conclusively ascertained.

These critical points with their accompanying changes of properties are of course the manifestation of changes of internal energy, such as occur for instance during the solidification or the melting of all substances. Here, however, these internal changes are not accompanied by any change of state and they must therefore correspond to some molecular, *i.e.*, allotropic changes. The existence of several allotropic conditions of iron, stable at different temperatures is no longer disputed. Osmond calls the condition of iron stable above the point Ar_3 γ, the condition stable between Ar_3 and Ar_2, β, and the condition stable below Ar_2 α. According to this theory the hardening of steel by sudden cooling is due to the retention in the cold metal of the iron in the γ condition, it having been denied the time to pass into the soft α condition which is the one normally stable in the cold.

The position and magnitude of the three critical points just described are very clearly shown graphically in Fig. 2, in the case of steels containing various amounts of carbon. In this diagram taken from Osmond the abscisses represent the temperatures and the ordinates the time required for the temperature of the cooling metal to fall 10°C. as indicated by a Le Chatelier pyrometer.

It is seen that when the metal contains 0·02% of carbon we have a pronounced evolution of heat at Ar_3, a slight one at Ar_2, while the point Ar_1 is absent. With 0·14% of carbon, the point Ar_3 occurs at a little lower temperature and the evolution of heat is less, while the point Ar_2 is sharper than in the previous

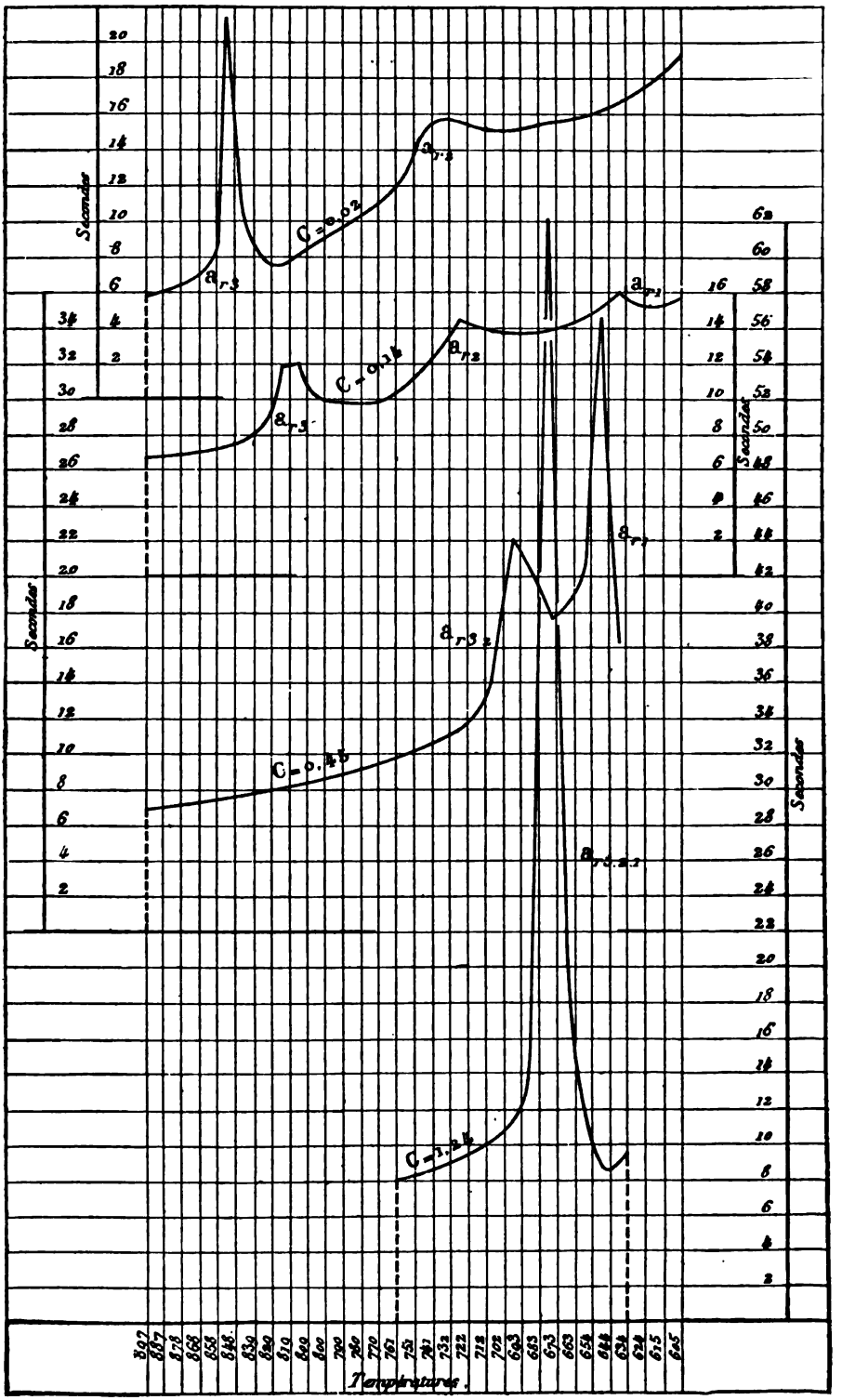
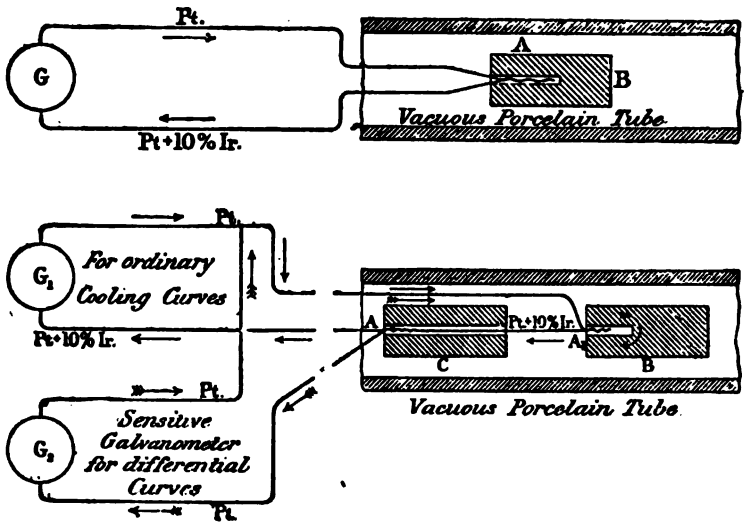


FIG. 2.—THERMAL CRITICAL POINTS OF STEEL.

sample; finally the point Ar_1 appears here, although remaining faint. When the steel contains 0·45% of carbon the two points Ar_3 and Ar_2 combine into a double point which may be called Ar_{32}, which is accompanied by a marked evolution of heat. In this steel, of course, the point Ar_1 is of considerable magnitude. Finally in the sample containing 1·24% of carbon, we see that the three critical points are united into a single point, Ar_{321}, accompanied by a considerable evolution of heat, producing a decided recalescence of the cooling metal.



FIGS. 3 AND 4.—THE NEW ARRANGEMENT OF THE LE CHATELIER PYROMETER.

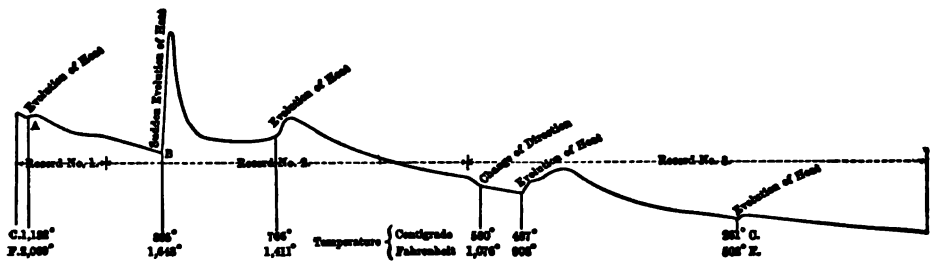


FIG. 5.—PHOTOGRAPHIC RECORD OF THE COOLING OF ELECTRO-IRON.
One-quarter of the actual size.

Sir W. C. Roberts-Austen's Recent Cooling Curves.—In his fifth report of the Alloys Research Committee,* Sir W. C. Roberts-Austen describes a very ingenious arrangement of the Le Chatelier pyrometer, by which the sensitiveness of the instrument is greatly increased and which led to the discovery of at least two additional thermal critical points in electrolytic iron.

In his new installation of the pyrometer, W. C. Roberts-Austen instead of measuring the difference of temperature between the thermo-junction and the cold junction of the pyrometer, (see Fig. 3) uses a second thermo-junction,

* *Proceedings of the Institution of Mechanical Engineers*, 1909; *The Metallographist*, Vol. II., p. 186.

(*A*, Fig. 4), placed in a compensating cylinder *C* made of copper, platinum or fire clay. *B* is the piece of metal experimented upon and its actual temperature is recorded by the galvanometer G_1, while any evolution of heat occurring spontaneously within the metal is recorded in a large scale on the sensitive galvanometer G_2. It will readily be seen that by this arrangement, the magnitude of these evolutions of heat is not masked by the fact that the mass itself is rapidly losing heat since it is the difference of temperature between the two thermo-junctions which are recorded, and this difference is not affected by the general fall of the temperature of the system.

With this extremely sensitive instrument W. C. Roberts-Austen found the cooling curve of electrolytic iron shown in Fig. 5.

The following is quoted from his own description of this curve:

"There is at *A* the point at 1,132°C., which was first observed by Dr. E. J. Ball. Then at *B* there is the ordinary Ar_3 of Osmond, which in this case occurs, not as in mild steel at the normal temperature of 850°C., but at 895°C. When the mass continues to cool down there is, as it was anticipated there would be, the point Ar_2, which in this case occurs at 766°C. The carbon point Ar_1 could not be expected to occur in iron of so high a degree of purity, and it does not exist; but there is evidence of evolution of heat at a point which I believe to be between 550 and 600°C. It is difficult to fix this point accurately; it seems to vary somewhat in successive curves. The next point, at which heat evolved, is a new one of extraordinary interest. It occurs between 450 and 500°C., and evidence will subsequently be adduced to show that it is connected with the retention of hydrogen by the mass of iron, even though it had been heated to 1,300°C. Finally there is a small point at 261°C., that is, at a temperature of no less than 400°C. below redness."

W. C. Roberts-Austen considers the presence of these two newly discovered critical points to be due to the presence of hydrogen in the metal. The upper point, the author says, "may represent the separation of an iron hydride from solid solution in the mass of iron, and the lower point, at 270°C. may be the corresponding eutectic point."

Further investigation will, of course, be required before we can accept these conclusions as demonstrated facts. Might there not be some relation between these critical points and the brittleness of soft steel at a blue heat?

The Constitution of Steel.—The true nature of the constitution of steel has continued to engage the attention of many investigators and valuable contributions have been made to the subject. As pointed out a year ago in THE MINERAL INDUSTRY, Vol. VII., it has been conclusively shown that steel should be regarded as a solution of iron and carbon. This view has been further confirmed by W. C. Roberts-Austen in his fifth report of the Alloys Research Committee already alluded to, and by a very able paper of Stansfield.*

From the cooling curves of many samples of steel containing various amounts of carbon, W. C. Roberts-Austen has plotted the curves shown in Fig. 6. The branches *AB*, *DB* and *ac* indicate the evolutions of heat taking place during

* "The Present Position of the Solution Theory of Carburized Iron," *Journal of the Iron and Steel Institute* 1899, II.; *The Metallographist*, Vol. III., p. 24.

the solidification of steel of different carbon contents, while the lower branches refer to the liberations of heat occurring when the metal is in a solid condition; in other words they indicate the position of the critical points described in the foregoing pages of this review, with the exception of the branch marked "separation of cementite," the corresponding heat evolution of which had not been considered on that occasion.

It will be noted that both the upper branches and the lower ones bear a striking analogy to the solubility curves of certain metallic alloys, and from this analogy a similarity of constitution is inferred which is fully confirmed by the appearance of the structure of steel under the microscope, as pointed out in my last review on the progress of metallography.

It is now conceded on all sides that the evidences point to the fact that steel at a high temperature, *i.e.*, above the critical range, is a solid solution of iron and carbon or more probably of iron and the carbide Fe_3C, and that by

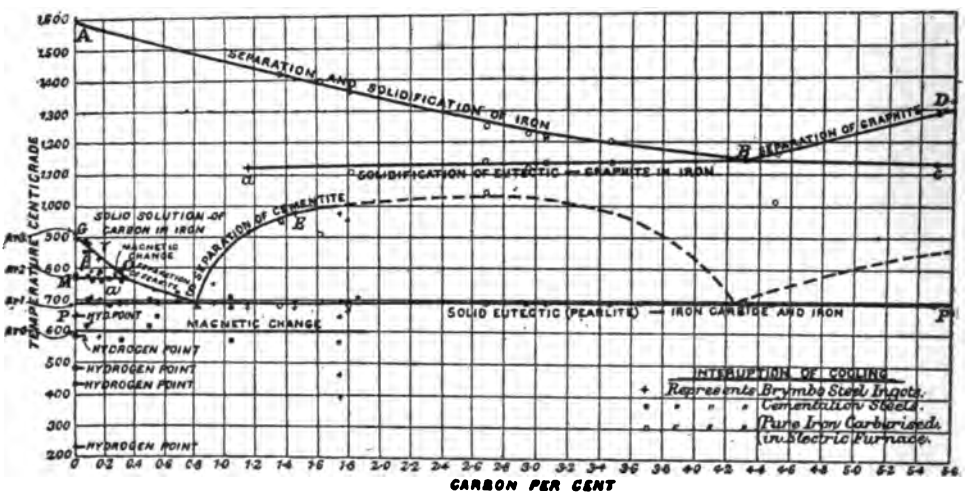


FIG. 6.—COOLING CURVES OF STEEL.

sudden cooling we retain this solution in the cold as we deny to the constituents the time to fall out of solution, and to this fact must be attributed the hardening of steel. As Stansfield has remarked, in the mind of many this solution theory of steel was thought to oppose the allotropic theory, but this is far from being so. On the contrary the latter theory is in perfect agreement with such conception of the constitution of steel. Indeed, from a critical examination of the allotropic theory we would naturally be led to the conclusion that quenched steel is a solid solution of carbon or of the carbide Fe_3C in γ iron, and that since γ iron is hard, according to the theory, quenched steel also must be hard.

Prof. J. O. Arnold and Andrew McWilliam have published a valuable paper* on the diffusion of elements in iron. They find in their results additional

* "The Diffusion of Elements in Iron," *Journal of the Iron and Steel Institute*, 1899, I.; *The Metallographist*, Vol. II., p. 278.

evidence in favor of Prof. Arnold's contention of the existence at a high temperature of a carbide of iron $Fe_{24}C$; few, however, will be able, I think, to follow their argument.

Prof. E. D. Campbell* reasserts his belief in the existence of a series of carbides of iron which he calls ferro-carbons, similar in chemical structure to the hydro-carbons, and which vary in proportion and nature according to the carbon content in the steel and the treatment received. His belief is based chiefly on the fact that when steel is dissolved in hydrochloric acid, hydrocarbons of various formulas are evolved. He maintains his position in his usual scholarly way.

Relation between the Structure of Steel and its Thermal and Mechanical Treatment.—The writer has summarized† the relations existing between the structure of steel and its thermal and mechanical treatment in the following propositions:

Changes of Structure Brought about by Heat Treatment.—I. When a piece of steel, hardened or unhardened, is heated to the temperature Ac, all previous crystallization, however coarse or however distorted by cold work, is obliterated and replaced by the finest structure which the metal is capable of assuming, the structure of burnt steel, which cannot be effaced by such treatment, being the only exception.

II. When a piece of steel, hardened or unhardened, after being heated to the temperature Ac, is allowed to cool slowly, it retains the fine amorphous-like structure which it had acquired at that temperature. It possesses then the finest structure which hardened steel is capable of assuming.

III. When a piece of steel, hardened or unhardened, after being heated to the temperature Ac, is suddenly cooled from that temperature, by quenching it in cold water for instance, it is fully hardened, and retains the fine amorphous-like structure acquired at that temperature. The metal possesses then the finest structure which hardened steel is capable of assuming.

IV. When a piece of steel, hardened or unhardened, is heated to a temperature above Ac and allowed to cool slowly and undisturbedly, the metal, whose crystallization had been obliterated by its passage through Ac, crystallizes again, the crystals or grains increasing in size until the temperature Ar is reached, below which there is no further growth.

Corollary to III. and IV.—When a piece of steel, after being heated to a temperature above Ac, is allowed to cool to Ac and then quenched, it will be fully hardened, but its structure will be coarser than if it had been quenched from Ac without having been previously heated above that temperature.

V. The higher the temperature above Ac from which the steel is allowed to cool undisturbedly, the larger the grains.

VI. The slower the cooling from a temperature above Ac the larger the grains.

Corollary to V. and VI.—Pieces of steel finished at a temperature above Ar will have a coarser grain in those parts which have been finished hottest,

* "The Constitution of Steel," *Journal of the Iron and Steel Institute*, 1899, I.; *The Metallographist*, Vol. II., p. 18.

† *Journal of the Iron and Steel Institute*, 1899, II.; *The Metallographist*, Vol. II., p. 294.

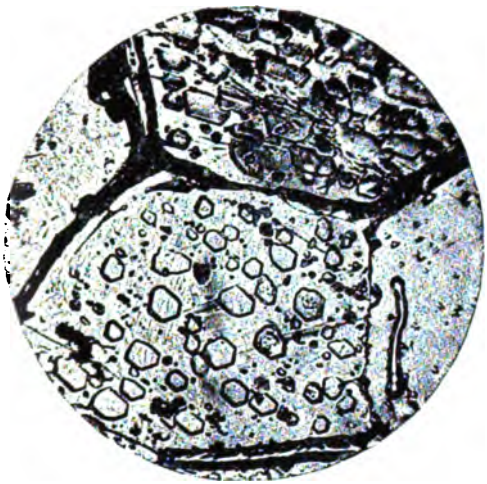


FIG. 1.—GLASS-CAST SURFACE OF CADMIUM. MAGNIFIED 1,000 DIAMETERS.

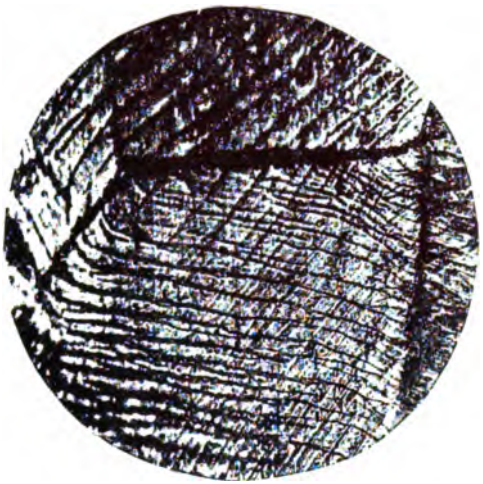


FIG. 8. — SLIP-BANDS IN SWEDISH IRON STRAINED BY TENSION. MAGNIFIED 400 DIAMETERS. VERTICAL ILLUMINATION.

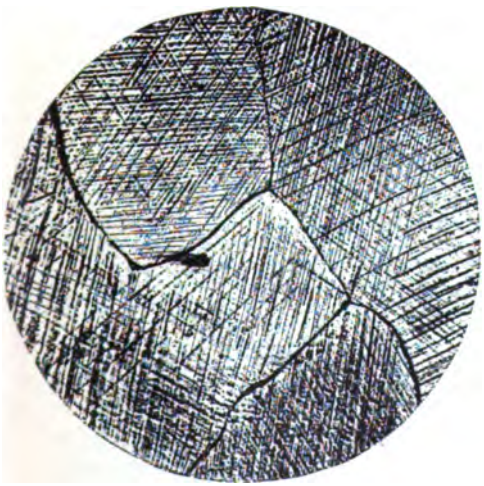


FIG. 9.—SLIP-BANDS IN LEAD STRAINED BY TENSION. MAGNIFIED 100 DIAMETERS. VERTICAL ILLUMINATION.

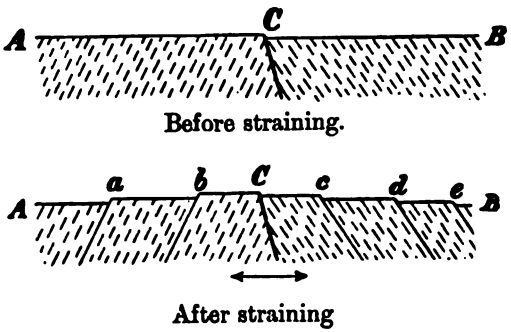


FIG. 10. — DIAGRAM ILLUSTRATING THE EFFECT OF STRAIN UPON THE STRUCTURE OF METALS OR ALLOYS.

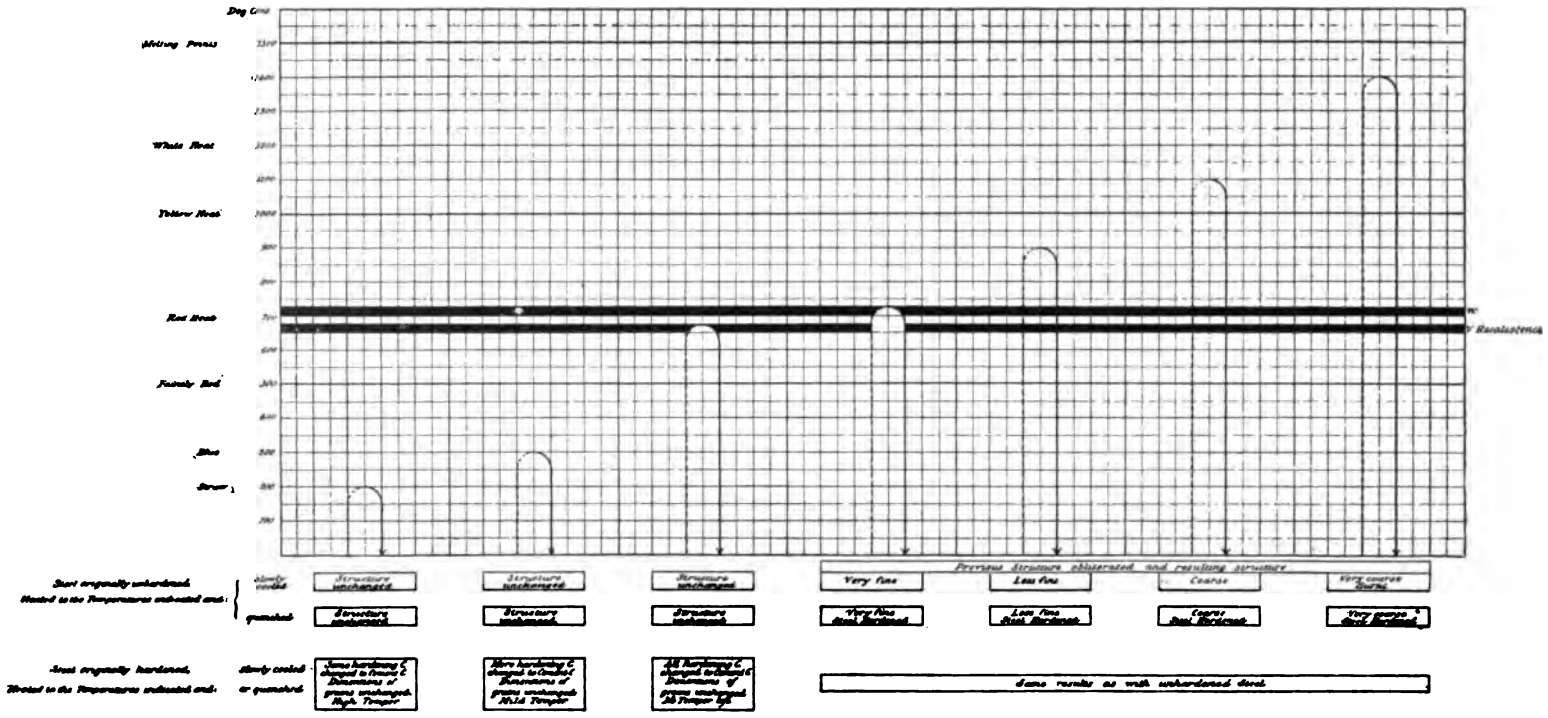


FIG. 7.—CHANGES OF MICROSTRUCTURE BROUGHT ABOUT IN STEEL BY HEAT TREATMENT.

and where subsequent cooling has been more gradual, *i.e.*, the central portions, or portions farther away from the cooling surfaces.

VII. When a piece of steel, hardened or unhardened, is heated to a temperature above Ac and suddenly cooled, it is fully hardened, but its structure will be coarser than when quenched after having been heated to Ac.

VIII. When a piece of unhardened steel is heated to a temperature below Ac and quenched or slowly cooled from that temperature, no change takes place in its structure.

IX. When a piece of hardened steel is heated to a temperature below Ac, some of its hardening carbon is changed spontaneously into cement carbon, and the metal is thereby softened. The tendency of the hardening carbon to pass into the cement condition increases with the temperature, and is the greatest at the temperature Ar. This transformation, however, is not accomplished by any change in the dimensions of the grains.

These propositions are for the most part illustrated graphically in Fig. 7, in which Brinell's mode of representations was adopted.

Attention is called to the exception of the behavior of soft steel to proposition VII, since it has been shown by Stead that when a piece of iron or of very soft steel is subjected to a prolonged heating at a temperature between 600 and 700°C. (therefore below Ac) the previous structure is obliterated and the structure becomes coarsely crystalline. It should be noted, however, that it is only the softest brands of steel (containing less than 0·20% carbon) whose structure is thus altered at a temperature below Ac.

Changes of Structure Brought about by Work.—We must at the outset consider two kinds of work: hot work and cold work. By hot work is meant work performed above the critical range, *i.e.*, above Ac; by cold work is meant work performed below the critical range, *i.e.*, below Ar. The work done between Ac and Ar will be cold work if the temperature of the metal has just been raised from below Ac, and it will be hot work if its temperature has just been lowered from above Ac.

The effect of work upon the structure of steel may be summarized in the following propositions:

I. While steel is being worked it does not crystallize (provided, of course, the working of the metal is sufficiently vigorous to affect all parts of the mass).

II. Hot work as such has no influence upon the structure of the metal. Indirectly, however, by retarding crystallization until a lower temperature is reached, it may influence its structure most decidedly; but the same results could be accomplished by heat treatment alone, *i.e.*, by reheating the unworked metal to the temperature from which the worked piece was allowed to cool undisturbedly.

Remark.—A certain amount of work is, of course, necessary to expel the slag, close blow-holes and other similar irregularities, and otherwise render the piece sound; but once this is accomplished, further work will not improve the structure, except indirectly as stated above.

III. Cold work distorts the grains or crystals of steel, flattening them and elongating them in the direction of the forging or rolling.

IV. The lower the temperature the more pronounced the effect of cold working.

Remark.—The structural distortion caused by cold working, with its accompanying alterations of the physical properties of the metal, may be removed by heating the metal to Ac. (See Proposition I.)

From the above considerations, we may draw the following conclusions of industrial interest:

Finished Pieces of Unhardened Steel.—Since it seems to have been conclusively shown that the smaller the grains of the metal, the more ductile and tough it will be, it is evident that we should endeavor to impart such a structure to all finished pieces; and as the finest possible structure results from heating to Ac, we naturally infer that every finished piece of unhardened steel, in order to be in the best possible condition, should, as a last treatment, be heated to Ac.

While such treatment cannot always be applied—being in many instances, altogether impracticable—nevertheless, manufacturers should endeavor to approach this desideratum as much as is consistent with other conditions and requirements of production. Forged or rolled pieces should be finished as near the temperature Ar as possible, since finishing them at a temperature much above Ar leads to the development of a coarse structure during subsequent cooling, while if they be finished below Ar they will suffer from the effects of cold work.

The problem, however, is further greatly complicated by the fact that thick pieces cannot be finished at a temperature uniform throughout, the differences of temperature between the central portions and the outside increasing rapidly with the cross section of the piece. If it be finished at a temperature above Ar the whole mass will begin to crystallize during subsequent cooling, but the interior, being hotter and cooling more slowly, will have a larger grain than the outside portion; the resulting structure will be far from uniform, and the physical properties of the finished piece in its various parts will also necessarily lack uniformity, as may be ascertained by testing specimens cut from different regions. If the outside or thinnest parts have reached Ar when the piece leaves the forge or the finishing rolls, the central portions may be considerably above that temperature, and may assume a coarse crystalline structure; while if the working be continued until the interior of the piece has reached the temperature Ar, the thinnest portions may be much below that temperature and suffer from the effects of cold working.

Uniformity of structure (and therefore of physical properties) in all parts of a worked piece of steel, together with freedom from distortion caused by cold work, can only be secured by reheating throughout to the temperature Ac; and while such treatment, as already stated, is impracticable in many cases, beside necessitating the use of pyrometers, it remains true that with a clear understanding of the effect of heat treatment and of work upon the structure of the metal, each manufacturer has it in his power to improve the quality of its product.

In the manufacture of costly pieces of steel, when large outputs, necessitating

the highest possible speed of production, is not the all-important and all-absorbing factor—in the production of armor-plates, for instance, and of expensive forgings—the metal may be treated on these scientific lines, and the highest efficiency secured.

Annealing.—From our previous considerations it is evident that to anneal in the most effective manner the metal should be reheated throughout to Ac, and slowly cooled from that temperature, maintaining it a sufficient length of time at the temperature Ar to assure a complete change of the hardening carbon into cement carbon. By heating the metal to Ac all previous coarse crystallization or distortion is obliterated and replaced by the finest possible structure. In cooling slowly through the Ar range, its hardening carbon passes back into cement carbon, but as the complete transformation requires time, it is desirable to keep the metal for some time at that temperature. After such treatment the steel will undoubtedly be in its softest and most ductile condition.

Hardening and Tempering.—It is almost superfluous to state here, that in order to harden steel, and at the same time preserve the best structural arrangement, the metal should be heated to Ac, and then quenched, without allowing the temperature to rise above Ac. This rule is quite universally followed, and although smiths seldom use pyrometers, some of them become such experts in judging the temperature of the metal by the color, that it is probable that the majority of their pieces are, at the instant of quenching, very near Ac indeed, or at the refining temperature, as they properly call it.

The temperature of tempering is also very universally regulated by the color of the metal, and generally with great skill and accuracy.

The use of pyrometers in connection with the hardening and tempering of steel should nevertheless be commended, as it removes all uncertainty, secures uniformity of treatment, and may be the means of avoiding the destruction in the quenching bath of many costly pieces.

It is customary, the writer believes, to quench the metal (merely for convenience?) after it has been heated to the proper temperature for tempering, instead of allowing it to cool slowly from that temperature. It is, however, probable that, if it be left to cool slowly, its softness will increase, since the change of hardening carbon into cement carbon, which is arrested by the sudden cooling, will continue, awhile longer at any rate, if the steel be slowly cooled.

The Effect of Strain upon the Structure of Metals and Alloys.—Ewing and Rosenhain have published in a recent and extremely valuable paper,* the results of their investigation dealing with the effect of strain upon the structure of metals.

Polished sheets of metals were strained very gradually while being examined under the microscope. When the yield point is reached, *i.e.*, when plastic deformation begins to occur, black lines are seen to cross the crystalline grains of which the metal is made up. These lines are generally quite straight and are parallel in the same grains, but have different directions in different grains. Fig. 8 shows the appearance of Swedish iron, strained by tension, under vertical light and magnified 400 diameters.

* "The Crystalline Structure of Metals," *Philosophical Transactions of the Royal Society of London*, Vol. CXCIII., p. 353; *The Metallographist*, Vol. III., p. 94.

As the strain increases other systems of lines, inclined to the first, make their appearance and eventually two, three and even four systems of intersecting lines may be seen in each grain. Fig. 9 illustrates these systems of lines in the case of strained lead, under a magnification of 100 diameters.

These lines are not cracks but steps in the surface, which steps are due to slips along the cleavage or gliding planes of the crystals. In Fig. 10 *AB* represent the polished surface of two grains, *C* the junction line between these two grains. The pull takes place in the direction of the arrows.

Yielding occurs by finite amounts of slips at a limited number of places *a, b, c, d, e,* resulting in short portions of inclined cleavage or sliding surface appearing black under the microscope, because they do not send back any light into the tube. By oblique light some of these slip bands appear black while others are bright. When the surface is rotated some of the bands which were black become bright and *vice versa* owing to the change of their position in regard to the incidence of the light, which is conclusive proof that the black lines are not cracks, but inclined surfaces as described above.

It is seen, then, that contrary to the general belief, metals remain crystalline after the severest strain, and that the flow or plastic strain in metals is not a homogeneous shear such as occurs in the flow of viscous fluids, but is the result of a limited number of separate slips, the crystalline elements themselves undergoing no deformation.

MINE TIMBERING.*

By Wilbur E. Sanders.

In this necessarily brief article the systems of timbering dealt with are those in use among the mines of the mountainous regions of Western United States. This qualification, of itself, requires no apology; for the cosmopolitan character of our miners—managers and engineers, superintendents and foremen—and their shrewd keenness in devising ways to meet the problems presented in underground workings, in selecting means peculiarly adapted to the end in view, and in improving upon well-known methods already in vogue have placed the science of supporting mine excavations by timbers, as developed by them, far in advance of that in use among the older and less progressive mining communities. This monograph does not include the methods used in coal mining in the East, or that in use in the copper and soft iron ore deposits of Michigan and Minnesota; nor does it treat of wood and iron cribbing for round shafts or of iron supports now used in many European mines, nor of timbering and metal supports used in large tunnels.

The mines operated under these methods present every known characteristic of lode formation. The veins and ore deposits lie at all angles of inclination or *dip;* they are of all shapes and sizes, from the small seam to immense masses hundreds of feet in width, and of all lengths; while the materials comprising them and their country formations vary in texture from rock of strength sufficient to overlie considerable excavations without extraneous support, to the soft ground which requires not only immense quantities of timber and waste-filling to carry the workings safely, but an eternal vigilance upon the part of those conducting the operations.

During preliminary work it is important to explore and develop the ore bodies in the surest and least expensive ways, and, except in the case of prospecting hidden or blind deposits, by means of bore-holes from the surface downward, metal mines are tested and all are exploited during their earlier

* A comprehensive book is now being prepared by Mr. Wilbur E. Sanders on the subject of "Mine Timbering," to be published by the Scientific Publishing Co., in the near future. The work will be eminently practical in character and profusely illustrated by plates from working drawings. The subject will be treated exhaustively, and will include the methods in vogue at metalliferous and coal mines, as well as the support of large tunnels and other allied subjects.—Editor of The Mineral Industry.

stages by shafts sunk or adit-levels driven in and with the ores. This latter is an axiom in mining during this period of development, and should be invariably followed where possible. When once the ores have been opened up so that an estimate may be made as to their extent and general characteristics, more expensive works necessary to prepare the mine for the larger operations of economic ore extraction may be safely entered upon. It sometimes happens that the required information as to ore bodies beneath the surface of a mining claim is sufficiently answered in and by the workings of adjoining properties to make preliminary prospecting of the deposits unnecessary; then systematic plans for operating on a large scale may be properly inaugurated.

It is not the province of this article to touch upon methods of mining in use above ground, whether by open-work, hydraulic mining, or other processes, but rather to deal with the support of underground excavations by the use of timbers, and the details of mining therewith connected. Nor is it intended to explain methods technically foreign to the subject, although such will be touched upon when used as adjuncts to systems of timbering, as waste filling, etc. In the plates drawn to illustrate the article, sizes of timber most frequently used have been arbitrarily taken for convenience. The plates giving dimensions are working drawings showing the methods of framing, as explained, and can easily be applied to frames and timbers of any desired dimensions.

In developing and exploiting mines the miner should remember that unnecessarily large openings for levels, shafts, and other similar workings, mean not alone the breaking down and transporting of needless quantities of material, but also the added expense of keeping in repair larger passageways than are necessary, an item of considerable importance in heavy or creeping ground. On the other hand the larger excavations are relatively the easier and cheaper to drive. The rule should be that the size of workings must be ample to carry out their purposes properly, but not larger than is necessary for economy in operation.

It often happens that conditions, local or otherwise, are such that the strongest timbering fails to withstand the pressure to which it is subjected, and other means of support must be employed. In exceptional cases large excavations may be supported with little or no timbering, but usually waste-filling must be extensively used as an adjunct to timbering if the mine is to be kept open. Swelling or "creeping" ground, resulting from the exposure of certain rocks and clays to the air, whereby they expand with a force no timbering can resist, demands prompt attention that the timbers may be relieved from abnormal strain. This is done by making use of an open lining of lagging, through the interstices of which sufficient material may be removed to relieve the unusual pressure upon the frames; a process that is continued as long as the conditions demand. The above scheme is employed at the Ontario mine, Park City, Utah, and in the Never Sweat mine, Butte, Mont., where a system of narrow "square-sets," with open lagging, placed outside of the timbers of the large three-compartment working shafts, has been successfully employed to meet just such conditions in swelling ground.

There are certain established principles connected with the use and framing of mine timbers that should be borne in mind. Pressure is best resisted in line

with the grain of the wood rather than across the grain; which may be made clear by the following explanation. (See Plate 1.)

Let Fig. 1 be a section of timber supporting upon end and side the weight-

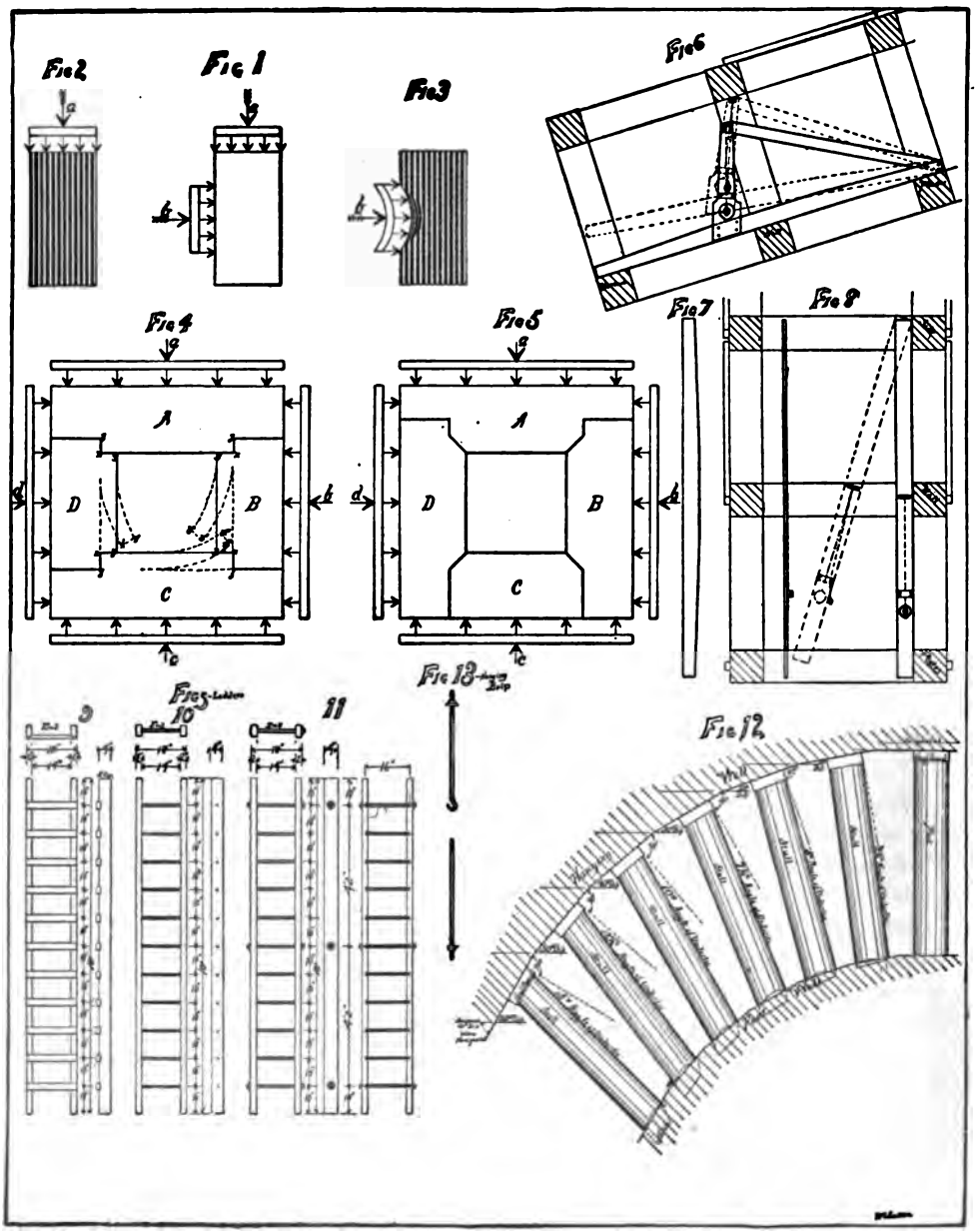


PLATE 1.

pressures represented by the arrows *a* and *b*; *a* acting in line with the grain of the piece, *b* at right angles to the grain. As shown in Fig. 2 each individual grain or fiber of which the block is composed resists the pressure *a* by the strength

of the combined fibers of the timber, or in other words by the full strength of the timber itself. On the other hand, Fig. 3, the pressure *b* acting across the grain is resisted by the power which binds together the bundle of fibers that make up the piece, the weight tending to crush them down one against the other until the shape and strength of the timber is destroyed. The writer has himself seen in Clark's Colusa mine, Meaderville, Mont., 24 in. of square-sawed yellow pine crushed down to a thickness of 8 in. by the weight pressing upon its side, at right angles to the grain of the wood, while the supporting post still retained its integrity.

In framing timbers the sets should be made with especial reference to the direction of the pressure-thrust. (See Plate 1.)

In Fig. 4 let us suppose the pressure upon the frame comes from the direction of the arrow *a*, in which instance it will be seen that the full cross-section of the timber *A* is supported at either end by the pieces *B* and *D*. This joint is without doubt an excellent one when, and only when, the entire pressure upon the frame comes from the direction *a* or *c*. The frame, however, is likely to be subjected also to pressures from the directions *b* and *d*, to resist which the timbers *B* and *D* offer only a portion of their cross-sections while the remaining part *x y* of the pieces tend to split off from the larger portions, thereby weakening the timbers by an amount equal to the sections *x' y'* so removed. A similar result from the pressure *c* might now cause the portion *x z* to split off from *C*, in which event there remaining as against the pressure *b*, no shoulder upon *C* to support *B* in place, the timber *B* would be forced from its position, causing the frame to collapse. Therefore for resisting pressure from two or more directions the framing shown in Fig. 4 is not applicable.

The only satisfactory remedy for this inherent weakness of square-shoulder framing is to make use of the mitered joint or beveled hitch, as shown in Plate I., Fig. 5. In this method, because of the support afforded a timber by the miter of this joint, the pressure from any of the directions *a*, *b*, *c*, and *d*, is resisted by the strength of the full cross-section of the piece against which the force acts. However, as shown in the piece *C*, the simple miter is not in itself sufficient to sustain any considerable thrust without a tendency to wedge apart the timbers *B* and *D*, and thus destroy the set. Without doubt the simplest and strongest joint obtainable is some combination of the square-shouldered tenon with the miter or beveled hitch. This combination is shown in the joints supporting the piece *A*. Here the full strength of the timber is obtained, with no tendency to split or slip, and weight up to near the point of crushing only serves to bind the set more firmly together.

Heavy ground is supported by the heavier timbers; by sets placed near together; and by sets strengthened by reinforcing sets or timbers.

Round timbers are stronger than square-sawed pieces, in which the grain of the wood has been cut and weakened by the saw. Used in the mine round timbers are less easy to handle than are the squared; they are less easy to align properly, and it is impossible to reinforce satisfactorily sets framed from such timbers by the usual false sets or pieces. The bark should invariably be removed from round timbers, as it collects moisture and fungus, and thus hastens the decay of

the wood. It also prevents the pieces from becoming properly seasoned before they enter the mine.

Prospecting technically belongs to that part of mine development which deals with the discovery of ores or rather products of mining, while exploitation deals with the internal working or extraction of the deposits themselves.

Timbering of Shafts, Stations and Levels.

Shafts.—Shafts are of two kinds, vertical and inclined. The former is more frequently used in large operations, where speed and convenience in hoisting are the prime necessity—particularly in connection with the more steeply inclined deposits, and with flat ones and pockets lying entirely beneath the surface. The usual size for single compartment shafts, and for the hoisting compartments of the larger shafts in metal mining is from 4 to 6 ft. in the clear of the timbers; the compartment used to carry water-columns, air- and steam-pipes, and ladders, is frequently made larger in cross-section than the working compartments. The expense of sinking shafts and of keeping them in repair in average ground increases rapidly beyond a certain size; it is therefore considered good practice to make the shafts as small as possible, keeping in view the work to be carried on through them.

Inclined Shafts.—Inclined shafts are used largely during the preliminary stages of development in veins, and other outcropping deposits that dip below the horizontal at angles too small to allow of the economical use of vertical shafts. Similar methods of timbering both classes of shafts are in use, although at times the timbering of inclined shafts approaches more nearly to that employed in supporting the level workings; as when application is made of the three-piece and four-piece level sets to inclines (Plate 2, Figs. 1 and 2). The single stull-piece, with head-board, is often used in the mountains when the hanging-wall or top rock is of such strength as to require little support.

When the three-piece level set is employed the cap is usually lengthened, and the top of each post fits into gains cut near the ends of the cap. Where a sill piece is desirable the sill is framed in the same manner as the cap, and the posts act as dividers. (See Plate 2, Fig. 2.) In such use technically the posts become end-plates, the cap and sill side-plates, while posts when used are placed lengthwise of the shaft as distance pieces to separate the sets. (See Plate 2, Fig. 5.)

A two-compartment shaft is constructed by placing a third post or girt in position at or near the center of the set in much the same manner as are located the end posts or plates.

The three-compartment shaft is similarly constructed by locating two such girts at their proper position, the tenons of the girts being V-shaped. (See Plate 2, Figs. 3, 4 and 5.) Behind or back of the side-plates and in line with the end-plates and girts, the set is tightly blocked and wedged in place. Pole or plank lagging is used where it is necessary to prevent falls of loose rock from the walls and sides of the shaft. Skip-ways are carried by the sill piece or bottom side-plate. Guides also are attached to the end-plates and center-girts when safety devices are used upon the ore-skips.

Other methods of framing the four-piece set, as applied to the inclined shaft, differ from the above framing in minor details, and at the same time allow of the use of the full width of the shaft. (See Plate 3, Fig. 1.) The halved

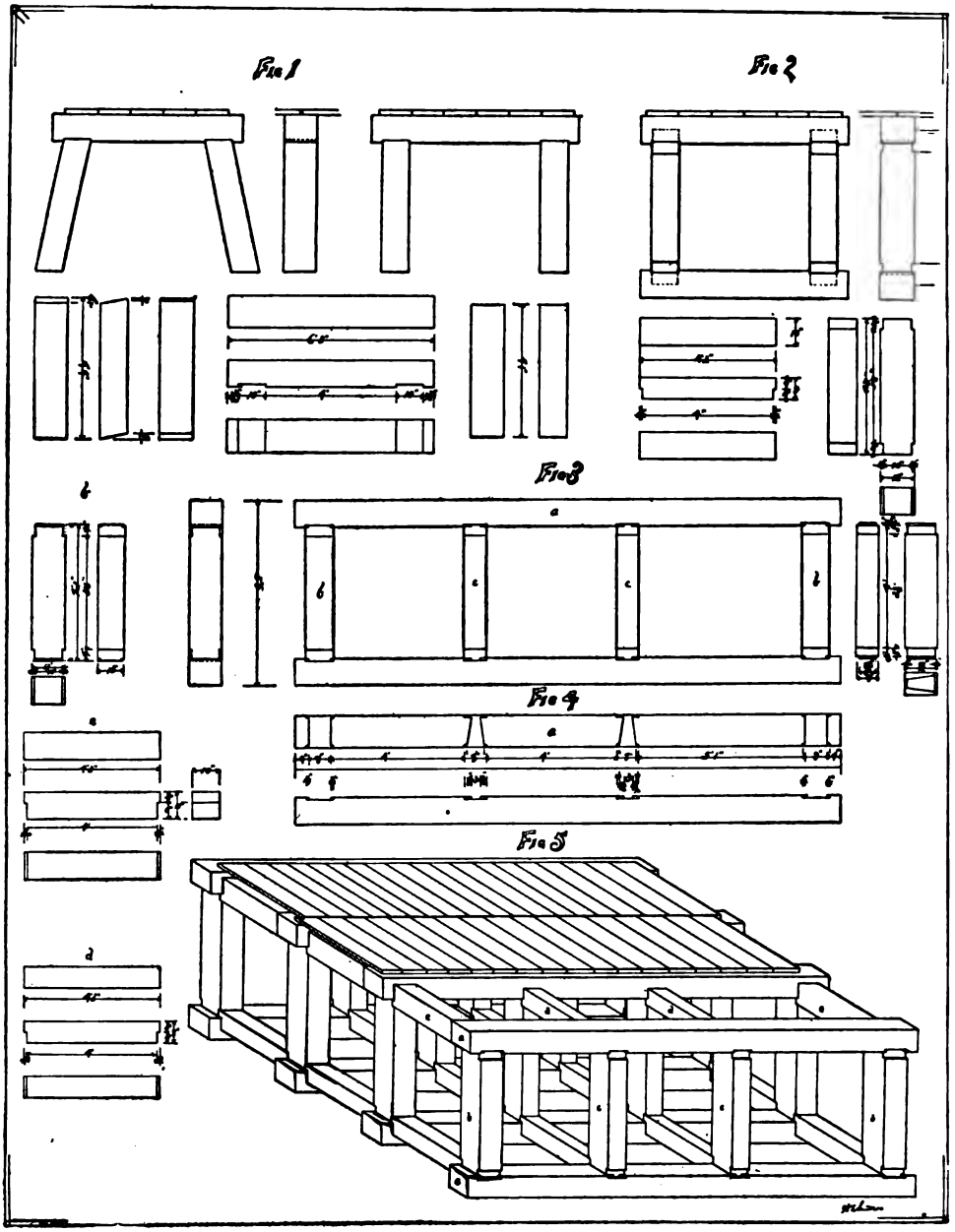


PLATE 2.

system of framing, as explained under vertical shafts, is rarely used for the inclines, and then only when posts are employed to form the complete square-shaft set.

Stations for Inclines.—The stations constructed for inclined shafts are of two kinds, one being so arranged that the ore-cars dump directly into the hoisting skip, held in position just beneath (Plate 3, Fig. 2), while in the other a 25- to 75-ton ore-bin is placed beneath the station and above the shaft, from which bin the ore is drawn into the skip for hoisting to the surface at intervals. This station, while requiring more excavating to construct, is by far the most economical in the end as the skip can be run entirely independent of the trammers or carmen. (See Plate 3, Fig. 3.)

Alignment of Incline Sets.—Probably the simplest method of aligning the side-plates of inclined-shaft sets, in order to get them in line one with another, is by the use of the combined straight-edge and plumb-bob.

A straight edge is made of a length greater by a foot or so than the distance between two sets. From the side opposite the true edge is built up a frame, one piece of which is so set that a plumb-line attached at its upper end will hang vertically along a fixed line, marked upon it, when the straight edge coincides with the true inclination of the shaft, and at the same time simultaneously rests upon three bottom plates. To prevent the plumb-line from swinging too freely it is confined at its lower end within comparatively small limits by a cleat attached to the upright piece. (See Plate 1, Fig. 6.) The straight-edge alone is used to locate the end-plates evenly in line with each other. (Fig. 7.) When the sets are placed they are bound in position by hanging-hooks or bolts, (Fig. 13) as explained under vertical shafts, and when so held are blocked and wedged firmly in place; the straight-edge, as above described, being employed to locate the sets in their true position during this operation.

Vertical Shafts.—The timbering of shafts varies according to the nature of the ground and the size of the shaft. Shafts sunk in some localities require little if any timbering, while in other places they are supported only by huge timbers that have been framed with the utmost precision.

Cribbed-Shaft Timbering.—In small shafts usually some form of cribbing is used. This system of shaft timbering is the simplest and often the cheapest in use, but it becomes cumbersome and expensive in large shafts. As usually employed it requires little framing, is easy to place, to repair or renew, and to keep properly aligned, and its use enables the timbering to be kept even with the bottom of the shaft in sinking, if that be necessary.

The simplest form of cribbing is that of poles, cut to required lengths and placed in pairs across each other, either from above or below. Located in this manner it forms an openwork lining to the shaft. Strips *aa* nailed to the poles upon the inside corners keep the cribbing in position. (See Plate 4, Figs. 1 and 1*a*.)

Cribbing is also formed from sawed timbers of various dimensions, the most simple method being that in which planks of the required lengths are placed around the shaft, upon edge and resting upon similar sets below or supporting similar sets above. These sets are made to resist the outside pressure, usually, by so placing the two shorter or end pieces that they will hold apart the two longer side pieces, the former in their turn being held in their position by corner

strips *b* within and nailed up and down the shaft to the side pieces. (See Plate 4, Figs. 2 and 2*a*.)

While the method of timbering is extremely simple it is unsatisfactory, and

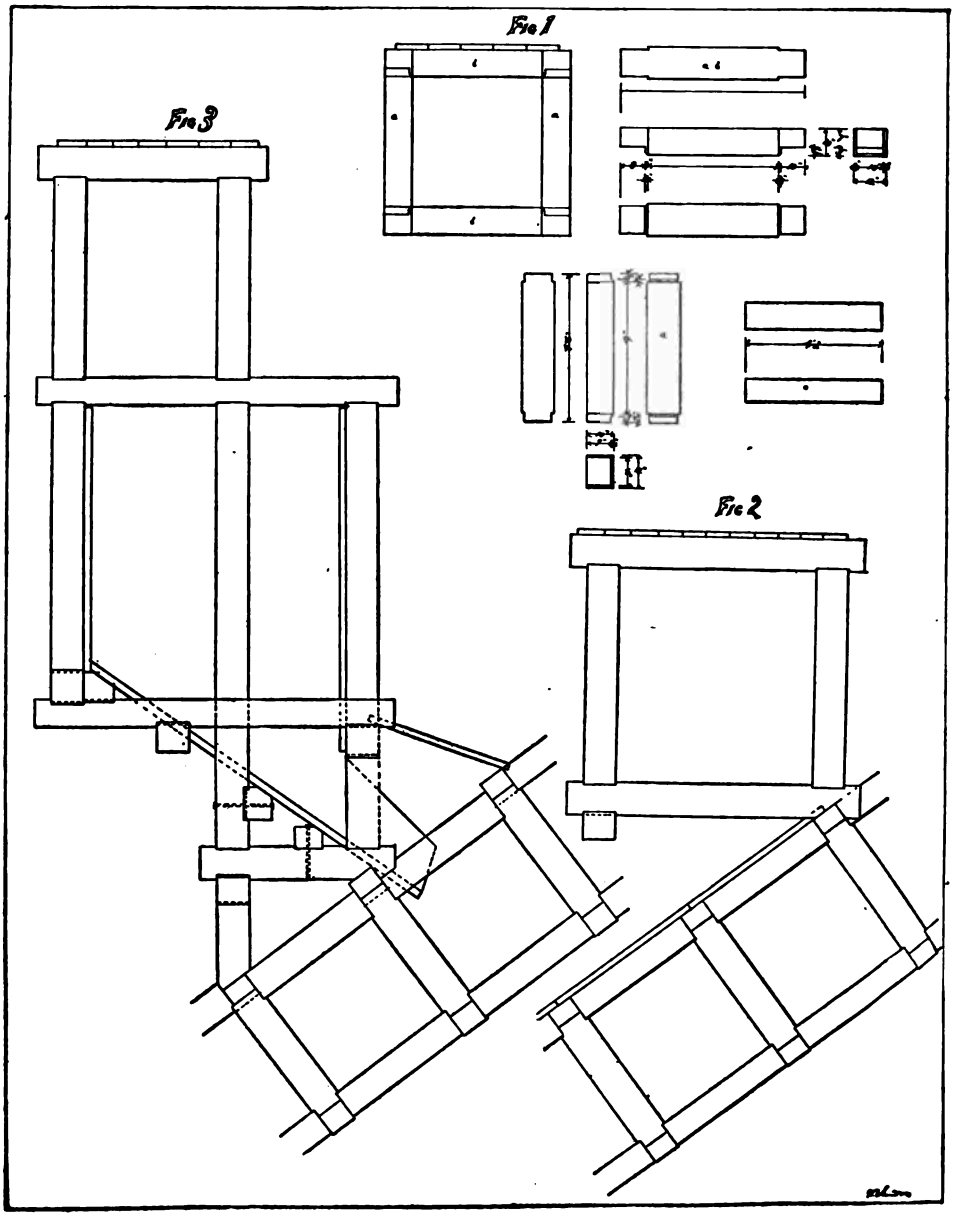


PLATE 8.

good mining practice makes use of the framed set as being stronger and in every way better. The basis of this system is some form of the tenon and mortise, whereby the ends of the two timbers forming a joint are framed with both a tenon

and what might be called an open mortise, the mortise of one piece engaging the tenon of the other and *vice versa;* to the end that each piece supports or keys its mate in place. See Plate 4, Figs. 3 and *a, b* and *c,* which show the more simple forms of this method of cribbing, the latter *c* being excellent for the reason that it causes the edges of the planks on the sides to break joint with the edges of the end planks in a way to stiffen the shaft and prevent the sets from moving horizontally one upon another. The methods of framing planks for these styles of cribbing are shown in Figs. 3, *d* and *e.*

Halved Framing for Shaft Sets.—A development of the tenon and mortise framing of joints is of almost universal application in advanced methods of supporting vertical shafts. This method as applied to the cribbed system is shown in Plate 4, Fig. 4, *a, A, b,* and *B.* The method of framing the pieces for openwork cribbing is shown in Fig. 4, *a,* and for tight cribbing, Fig. 4, *b.*

Fig. 4 *B,* also shows the framing of the opening from the shaft to the levels. False timbers or struts are used temporarily to hold the sides of the shaft intact while this opening is being framed into the level, or the framing can be placed before the planks are removed from the cribbing. A stronger and more satisfactory frame, however, is obtained by the combined beveled hitch and halved joint. The most satisfactory use of this combination is that in which the top and bottom of the side-plates are made to break joint with top and bottom of the end-plates. (See Plate 4, Fig. 5.) Details of this framing are given in Fig. 5, *a* and *b,* while the method of placing the timbers is shown in the isometric perspectives *A* and *B.*

Compartments are formed by cutting the side plates to receive a center girt that is framed very similarly to the end-plates.

The tight cribbing has been used for large shafts in heavy ground. On the Comstock lode, Virginia City, Nev., several of the important shafts were timbered with a solid cribbing of 14-in. pieces.

Square-Set Shaft Timbers.—In the square-set system, as applied to the timbering of vertical shafts, the heavier timbers of a cross-section of 6 in. and upward are employed. A set consists of the side and end pieces, with posts used to separate the horizontal frames. In the larger shafts divisional timbers, called girts, are used to separate the compartments. The side and end pieces are called wall-plates, for the reason that they frame the sides or walls of the shaft. The longer pair of plates are designated as side wall-plates—usually called side-plates—and the shorter pair as end wall-plates, or end-plates. Shafts of a single compartment are characterized as one-compartment shafts; and those which are divided by inner struts or girts into two, three, and four divisions, as two-, three- and four-compartment shafts. It is doubtful if shafts larger than with four-compartments can be successfully operated in deep mining, unless in exceptional cases. The framing of the various sized shafts is very similar, differing only in details that will be explained later.

One-Compartment Shafts.—The timbering of a one-compartment shaft consists of two side-plates, *a,* two end-plates, *b,* and four posts, *c,* which technically make a single set, successive sets being used to the bottom of the shaft to support the sides, and a lining of plank or lagging, *d,* is employed to prevent falls of

loose material into the opening. (See Plate 5, Fig. 1.) For the wall-plates the halved method of joint framing is employed, and at the same time a hitch or square shoulder 1 in. deep is cut in the tenon as a support for the post. This

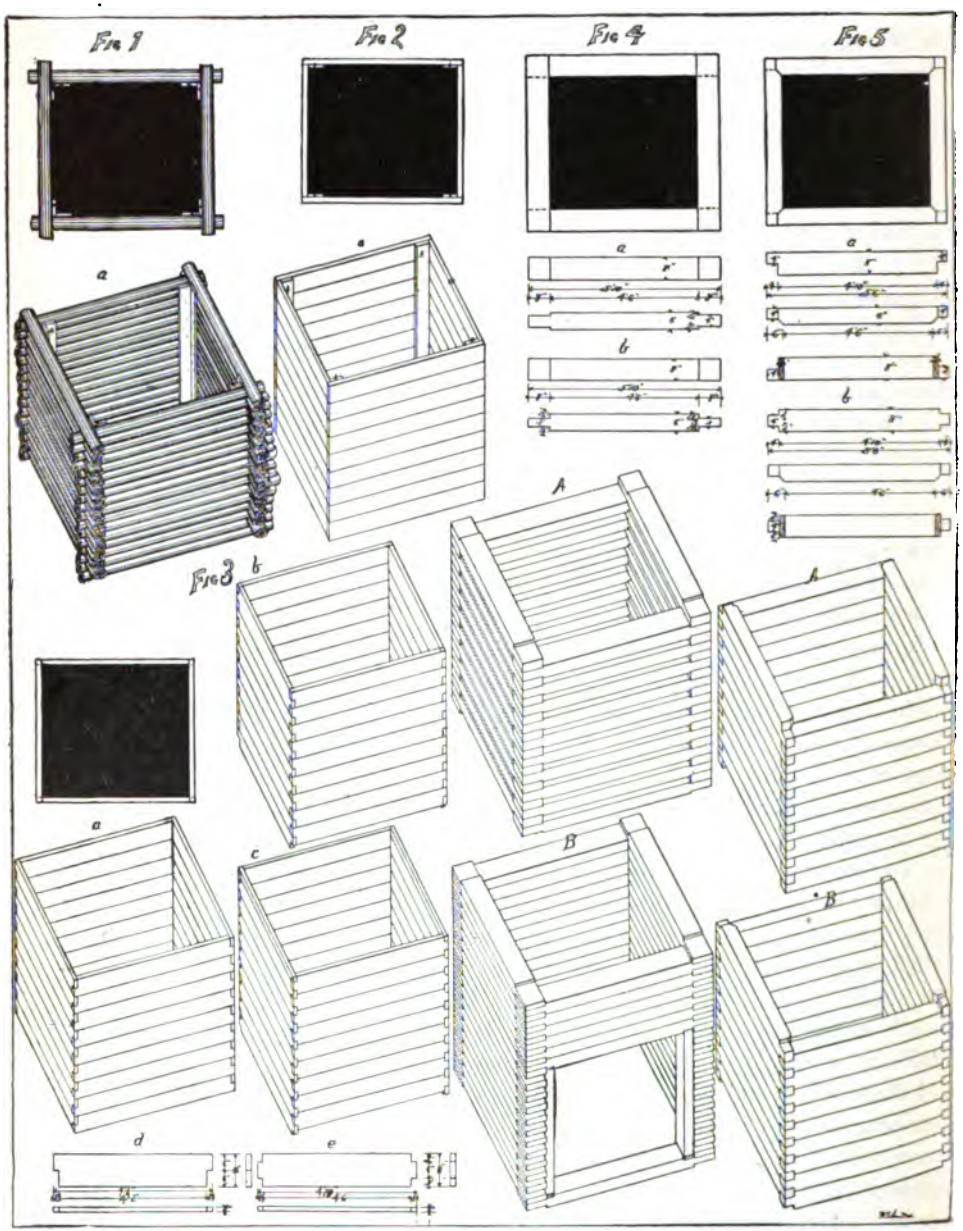


PLATE 4.

halving of the timbers, if used alone, greatly weakens them and the beveled hitch is framed from their inner faces so that their full strength may be brought to the support of the shafts. (Plate 5, Fig. 1, *z*.) The dimensions of the set

are such that the plates when fixed in position are separated 5 ft. from center to center, the practice being to increase the size of the timbers in heavy ground rather than to place the frames nearer together.

In framing the sets the utmost care is taken that the measurements shall be exact, and that the timbers shall be cut true to the line, especially for all important working shafts, large and small. A center line is laid off upon the inside faces of the plates and the measurements for tenons, mortises, and miters, are taken from this line. The faces of the tenons and shoulders are made at right angles to or parallel with, and measured from the face of the plate. These precautions are necessary because of variations in the dimensions of timbers. The halved tenons of each side-plate occupy the lower portion of that timber, those of the end-plates the upper part, so that when in place the side-plates support the end-plates by their tenons. See Plate 5, Fig. 2, *a*, showing the isometric perspective of this shaft.

Locating Shaft-Sets.—When a shaft is to be sunk from a surface too level to furnish possibilities for the disposal of the waste material coming up from below, it is necessary to elevate the top or collar of the shaft above the surface of the ground. This is accomplished by building up a cribwork of rough timbers to the desired height by placing logs of sufficient length in layers by fours or more across each other, with the shaft opening in the center. This cribbing is reinforced by waste filled in against it, and with this as a backing the shaft sets are located in position and blocked securely against the cribbing.

It also happens at times that the top material in which a shaft is to be sunk is too loose to support the sets by simple blocking. In this case the tenons at both ends of the side-plates are made to extend beyond the limits of the shaft, and these extensions are bolted to cross timbers above or rest upon such timbers as a support, the latter being of a length sufficient to bear upon the ground to either side of the shaft, and thus support its weight until it shall have entered rock firm enough to afford secure support to the sets by blocking and wedging in the usual manner. (See Plate 5, Fig. 3, *a* and *b*.)

The process by which shaft sets are located and fixed in position as integral parts of the shaft is as follows: The side-plates of the set to be put in place are swung to the set above by hanging hooks or bolts, which are made usually of 0·875-in. round iron, hooked at one end and threaded at the other as far as may be necessary. (See Plate 1, Fig. 13.) These hooks are used in pairs, the length of each being about 4 in. greater than one-half the height of each set from outside to outside of the wall-plates, measured vertically. Thus, with plates of 12-in. cross-section in a 5-ft. set, the length of each hook would be about 3 ft. 4 in., with the bolt end threaded for 6 to 8 in. These bolts are the simplest and most easily manipulated device yet constructed for hanging a set into place. Holes are bored through each side-plate, two at fixed points near either end, to receive the bolts, one hole at each end being used to hang the plates to the set above while the other holes are intended for the next succeeding set. Cast-iron washers are used between the plate and nut to give bearing to the latter when binding the sets together. The bolts having been located in the plates, the hooks attached to the loose timbers are caught upon the hooks of the set above,

the end-plates are put into place, their tenons resting upon the tenons of the side-plates, the posts are set in the hitches cut to receive them and the nuts are screwed down until the frame is tightly bound to the set above. If this

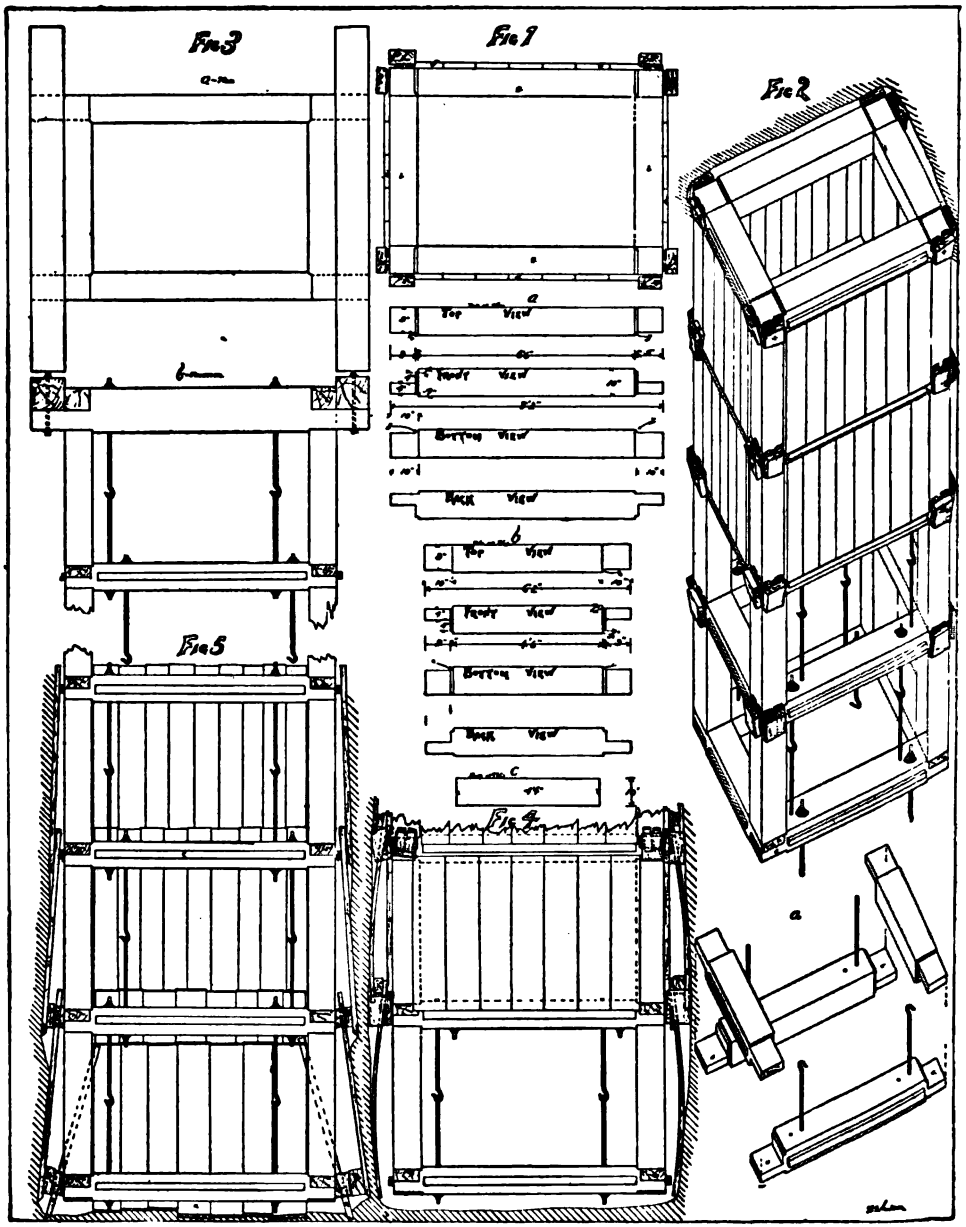


PLATE 5.

upper set is properly level, and the framing of all the parts correctly done, the center line marked upon the new set must be level. Blocks x are placed on the two sides of each corner in line with each plate, between wall-rock

and frame, and wedges are driven to tighten the set in its proper position. (See Plate 5, Figs. 1 and 2.) The back of each plate carries a strip, nailed thereto, and resting upon this as a ledge for support is placed the plank lagging or lining of the set. Filling is stowed behind the lagging, as the planks are put in position, sufficient to prevent movement of the surrounding ground that would be likely to throw the shaft out of plumb. The same process is repeated as the sinking progresses to the bottom or sump of the shaft. In dangerous ground the practice is not to remove the bolts after the sets have been located, and it is well in any case to leave them in place for several sets from the bottom of the shaft in order to bind the frames firmly together at this point.

Alignment of Vertical Shafts.—Various methods of aligning the timbers of vertical shafts are in use, the most satisfactory probably being the combination straight-edge and plumb-bob. A double straight-edge of a length sufficient to extend over three wall-plates in position—about 11 ft. for 5-ft. sets—is made, near the center of which is attached a plumb-line of a length of about 4 ft. A hole is cut in the straight-edge near its bottom end, in which the bob may swing freely, while a cleat attached just above this point serves to confine the line so that it is quickly located at the center mark, and a line is drawn upon the flat side of the piece parallel to the true edge, with which mark the plumb-line must coincide when the true edge is exactly vertical. (See Plate 1, Fig. 8.)

The set having been bound to the one above, and blocked to its approximate position, is then aligned truly with the two sets above by means of the straight-edge (Fig. 7) and by the combined straight-edge and plumb-line (Fig. 8) and is brought to its exact position vertically, the wedges being driven first at one side and then the other until the set is in place. Usually the sets are aligned first at the side, the side-plates first at one end and then at the other being brought into position by the wedges, when the process is repeated with the end-plates in order to align the ends of the shaft. Should it be a shaft of two or more compartments the side-plates are aligned by blocks and wedges in line with the divisional girts separating the compartments after the corners of the shaft have been brought to their places in the same manner as has been described. If the timbers are rightly framed the inner faces of the wall-plates should exactly coincide vertically with the inner faces of the sets above. The frame having thus been brought into and fixed in its true position, the lining is placed and the set is complete. (See Plate 5, Fig. 2.)

Repairing Shafts.—When by reason of undue strain, weakness develops in one or more of the timbers of a shaft, the faulty pieces must be removed and replaced by new ones. Preliminary to this work several sets, particularly those next above the point at fault, are tightly bound together by the hanging-bolts. If posts only are to be replaced it may be accomplished by removing the lagging adjacent, excavating enough ground from behind each post to allow of its being driven back from the shaft until it is clear of the timbers, or it may be chopped out with little trouble. The new post is then placed in position from behind, being driven or wedged into place and fitting into the hitch framed to receive each post in the plates. When necessary to replace the wall-plates the lagging of the adjoining sets above and below are removed, the blocks are knocked away

and the posts taken out, when the plates may be released and new ones put in place. The posts are then returned to their position, the set is bound to the plates above and below by bolts, blocked, wedged, and aligned, lining is put in, and the repairs are complete. It may happen that the ground will not stand during this process, in which case false timbers and lining must be used to hold the walls of the shaft in place.

Timbering Shafts in Loose Ground.—Shafts are frequently sunk in ground that breaks away from the walls before a set can be placed in position, and a quick process of lining the sides of the excavation is necessary. A method of false lining, largely in use throughout the West, keeps the loose earth from falling. It consists of planks of desired lengths placed vertically, and so blocked and wedged into position as to press each piece outwardly against the walls. The top end of the plank *a* is blocked, *b*, from the wall against the lagging of the last set placed in position, reaching a foot or so above the wall-plates of that set. The plank is further blocked and wedged, *c*, away from the wall-plates themselves, the effect of this being to throw the foot of the piece backward from the shaft against the side of the excavation, and thus prevent the material from coming in. This lining is often carried completely around the shaft. (See Plate 5, Fig. 4.)

Timbering Shafts in Running Ground.—In soft running ground, or loose ground too heavy to allow of placing the false lining above described, a method of spiling is employed that is practically identical with that used for driving levels through similar material. The process consists of supporting the dangerous ground beneath the last set placed in position, by what might be termed an enclosing and protecting shield of plank spiling or forepoling, that is advanced downward from that set piece by piece as the material is excavated from within. The spiling, *a*, sharpened at the foot, and often shod with iron at the head, is driven with a sledge, one plank at a time being advanced for a short distance as the material is withdrawn from before it. The spiling is held in position by the set and the material through which it is being forced, only enough of this being removed at a time to allow it to be driven a short distance; otherwise the pressure from without may force the lining into the shaft. Each plank around the shaft is driven successively one by one, until the entire shield has been advanced, when the process is repeated, and continued until the shaft has been excavated to a depth sufficient to allow of the placing of another set in position, the idea being to advance the shield by successive small stages during the work.

The spiling is started at a considerable angle, but as it is driven downward it tends to approach nearer and nearer the vertical until, when the new set has been permanently located, tail-pieces or bridges, *b*, are placed to hold the bottom of the planks in position, and at the same time to furnish an opening between the plates and the foot of the shield through which to drive the spiling for the next succeeding set. These tail-pieces may be permanently left in place, or removed in order to allow the planks to settle against the top of the spiling below, binding the latter in place and making a closer lining. Ledge-strips *c* may be attached to the plates, and the usual close lining placed about the shaft as additional

security, and to keep from the shaft material that otherwise might work in at the corners. The posts prevent spiling from being placed vertically so as to form a continuous close lining, which difficulty may in a measure be overcome by diagonal spiling so placed as to cover these openings, whether they occur at the sides or at the corners of the compartment shafts. Where possible the sets should be blocked and wedged to place in order that the shaft may be kept plumb, and the hanging-hooks should always be retained in treacherous ground. The above process is repeated successively until the shaft has entered firm ground, when the usual methods of timbering may be resorted to. (See Plate 5, Fig. 5.)

Two-Compartment Shafts.—In preliminary operations, in a mine where pumping is necessary, two-compartment shafts are employed, one of the divisions being given up to hoisting and the other to pumping and ladders. Both compartments are made of the same size, the usual practice in the West being for each division to be 4 ft. along the length of the shaft by 4 ft. 6 in. across its width, hoisting cages being most frequently constructed for operating in compartments of those dimensions. The timbering of a shaft of this size is framed in a manner almost identical with that of the one-compartment shaft, with the exception that the side-plates are made longer, and that a divisional piece, called a center-girt, is made to fit by tenon and mortise across the center of the side-plates. Center posts are also used to strengthen and stiffen the frame. The methods of locating and aligning the sets are those used for the one-compartment shaft.

Three-Compartment Shafts.—Should the mine warrant more extensive hoisting appliances, a third and larger compartment for pumping is added to the two-compartment shaft, while the smaller compartments are given up to hoisting. Because of the jar and strain upon the timbers from winding, this work should be done in the compartments that are supported by the solid side-plate as they are more rigid and self-sustained.

Although three-compartment shafts are often enlargements from two-compartment shafts, nevertheless most of the large working-shafts throughout the West are those of three compartments that have been commenced and carried to the bottom as such. Plate 6 gives the isometric perspective of a shaft of this type. This framing is such as obtains the greatest possible stiffness and strength for the wall-plates, and represents the most advanced timbering in use. The arrangement is excellent. The small cage for the use of the pumpmen, traveling closely against one of the side-plates to allow space at the opposite side of the compartment for locating the water-column, air- and steam-pipes, is hoisted by an independent engine, and the safety arrangement for the ladders is carefully designed.

The sets are located in the same manner as are those of the single compartment shaft. The side-plates are hung to the plates above, the end-plates are placed in position, the solid-center-girt is fitted into the mortise cut to receive it in the center of the solid side-plate, the divisional girt is located at the joint between the long and short side-plates, the eight posts are placed, and the set is tightly bound by the hanging-bolts to the set above, blocked, wedged, and aligned. The side-plates are sometimes made of a single piece, framed to receive two solid center-girts, which make the shaft more rigid. Long timbers of this

kind are difficult to handle in a shaft, and it is not always possible to use them. Where possible they should be placed at the stations, both above and below, in

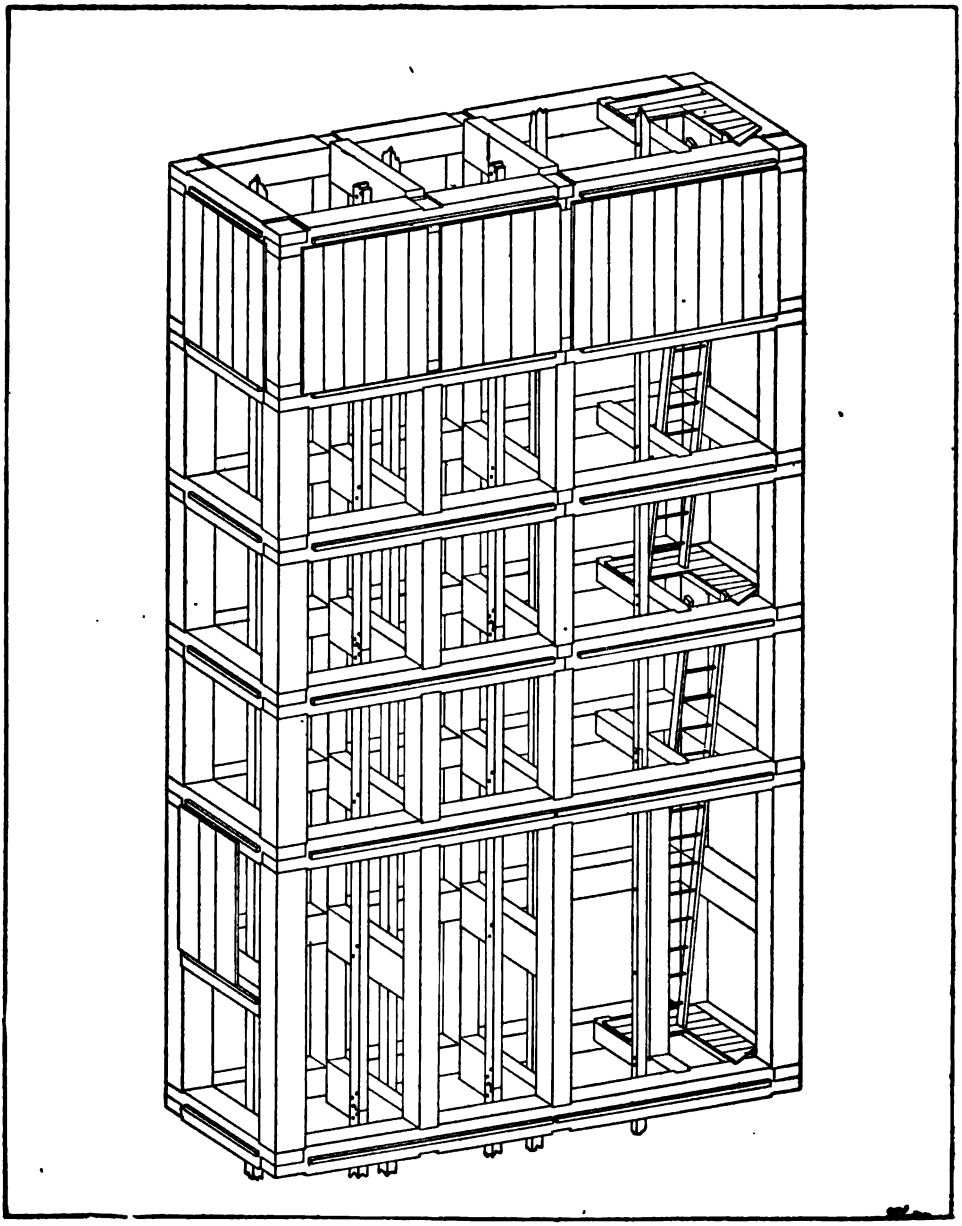


PLATE 6.

order to make the frame of the station set as strong as possible, as shown in Plate 9.

The principal reason for the almost invariable adoption of the double-hoisting-compartment shaft in large operations throughout the West is that of balanc-

ing the loads in winding. One of the cables winds over, the other under the same engine-shaft, and when the two drums or reels are both clutched to the shaft the weight of one cage and load acts in a measure to balance the other, thus saving power.

Four-Compartment Shafts.—Four-compartment shafts, with three hoisting divisions, may be divided into three classes, viz.: the single-width shaft, largely used throughout the Witwatersrand gold fields of South Africa; the "L" shaft, now practically abandoned for good reasons; and the double-width shaft, which it would seem is likely to come into general use as being one peculiarly adapted to the vast operations of extensive mining.

In the single-width four-compartment shaft the two end hoisting-compartments are employed for raising ore, and, on occasion, for lowering timbers; the third is used in sinking and by the pumpmen, and likewise for lowering timbers into the mine, while the fourth compartment is given up to the pumps, the water-columns, air- and other pipes, and to the ladders. The framing of the timbers for this shaft is almost identical with that for the three-compartment shaft, except that the divisions are such that each section of the side-plates supports two compartments. (See Plate 7, Fig. 1.)

The double-width four-compartment shaft practically comprises two two-compartment shafts placed side by side, the end-plates being lengthened in order to form a double-width shaft. Two of the end compartments are used for the hoisting of ore and the lowering of men and timbers, one of the two remaining is employed as a cageway for pumpmen and timbermen, with its station cut on the opposite side of the shaft from that for the hoisting divisions, while the other is given up to the uses of the pump and for carrying the water-columns, air and other pipes, and ladders. See Plate 7, Fig. 2, giving the plan of the shaft, and *a, b, c* and *d* the method of framing the timbers, which is similar to that for the three-compartment shaft, and Fig. 3, showing the isometric perspective of the construction.

The wall-plates of this shaft are solid throughout their length, giving great strength and rigidity to the frame; none of the timbers are so long as to render their handling in the shaft difficult, while at the same time the excavation is nearly square in cross-section, and of a size to facilitate the breaking of ground, and the placing of the frames in position. Furthermore the bracing received by the plates from the interior cross-girts, both long and short, greatly solidifies and strengthens the set. The great capacity of this shaft, together with its compactness, strength and rigidity, and its accessibility for repair, renders it especially adapted to extensive mining operations. As compared with the three-compartment shaft this shaft requires the excavation of but 450 cu. ft. more of material in each 100 ft. of its depth than does the former.

Ladders.—The ladders that are placed in the manways of a shaft, and in other inclined and vertical passageways throughout a mine, are usually made in 5-, 10- and 15-ft. lengths or thereabouts. (See Plate 1, Figs. 9, 10 and 11.) They consist of two supporting 2×4-in. scantlings, placed parallel about 14 in. apart, to which are fixed cross-pieces or rungs at regular spaces of about 10 in. The rungs may be of wood or iron, preferably the latter, unless the mine waters are

sufficiently acid to attack the iron—which is often the case in the deep workings. The wooden rungs are often simply spiked to the scantlings, or they may be set into hitches cut in the edges of the side pieces, and nailed firmly in place.

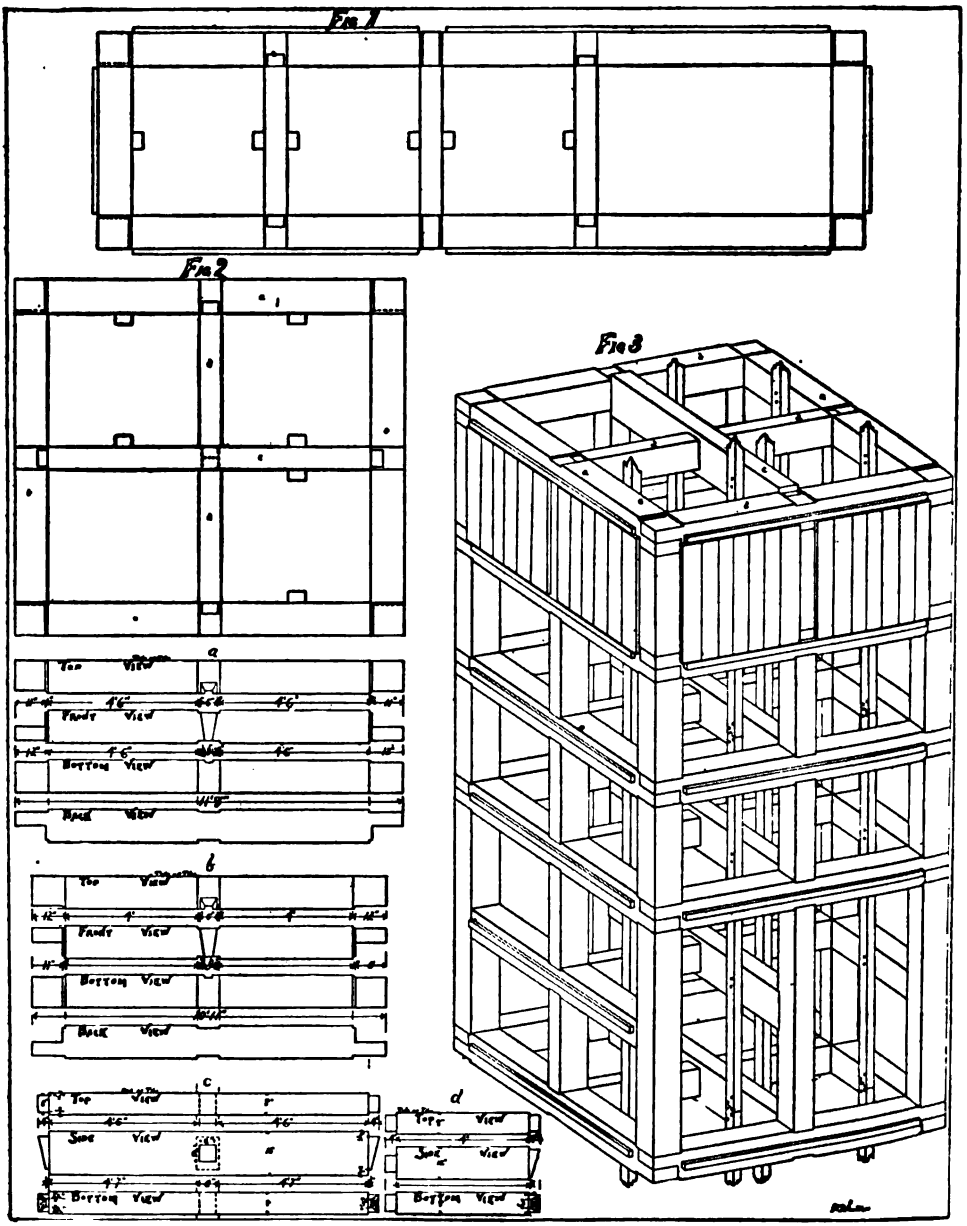


PLATE 7.

Frequently the scantlings are bored, and turned rungs are fitted into the holes, the ends of the rungs being wedged to hold them in place. While this method makes an excellent ladder, the holes weaken the scantlings materially, and

furthermore, it is almost impossible to replace a rung without destroying the ladder. The most substantial ladder, and one easily repaired is the following: The inner face of each scantling is bored at the required intervals to a depth of 1 in. in order to receive the ends of 16-in. lengths of 0·75- or 0·875-in. common iron pipe, cut exactly. At every fifth rung smaller holes, concentric with the others, are bored entirely through the scantlings, through which and through the lengths of pipe located at such points are passed 20-in. lengths of 0·75- or 0·875-in. round iron, threaded at the ends to receive nuts. When the different rungs of the ladder have been located in their places these nuts are tightened upon washers fixed between them and the scantlings, binding the ladder frame securely together. These rungs possess great strength while but little strength is lost to the scantlings, and the ladder is easily repaired.

Shaft Station-Sets.—At various depths, usually at intervals of 100 ft., levels are run from the shaft, to and through the inner workings of the mine, and at such points stations or large rooms are excavated in the walls of the shaft, and timbered to serve as centers for the storage of material of all kinds, whether coming from, or to be distributed to the various working places. The construction of these stations necessitates a change from the usual shaft framing at such points, in order to obtain the needed height for an entrance from the shaft to the station. Should the shaft have been carried below the point for a station, the obstructing wall-plates at the entrance must be removed, together with their girts, and the posts both above and below. The longer posts, fitting into the gains framed in the plates, are then located, and distance pieces for the walls and girts for the center of the shaft—the same being tenoned to mortises in the posts—are placed in position, occupying that of the wall-plates removed except that the station entrance is left free of such pieces. The remaining sides of the shaft are then lined with the usual tight lagging. See the isometric perspectives of the four-compartment shafts, (Plates 6 and 7). False pieces or temporary struts holding in place timbers to support the walls of the shaft during this operation are used when necessary.

STATIONS.—The ordinary working station is made of a width, in the clear of its framing, of the two hoisting compartments, or of a width which is enlarged at the pumping stations by an additional chamber usually equal in width to the length of the pump compartment, for the accommodation of the pump; the usual practice being to make the inner faces of the station-sets align with those of the shaft timbers.

In height the stations are made equal to that of the shaft entrance, less the thickness of the flooring, but the roof is made to slant downward from the second or third station-sets to the end of the room. The length of the station varies with the conditions, from 20 to 40 ft. being usual.

The timbering of stations consists of the four-piece level-sets, enlarged and placed at such distances apart as the nature of the ground requires, usually from 5 to 10 ft. Distance pieces or girts are used to hold the sets in position, this in connection with the usual blocks and wedging. Ordinary plank lagging prevents the fall of loose rock, and a flooring of 2- or 3-in. planking is laid to which is screwed a turning sheet of boiler plate, whereon ore-cars may be turned or

slewed around. A sump tank to hold the mine water that is to be pumped to the surface is framed and placed beneath the pump division of the station.

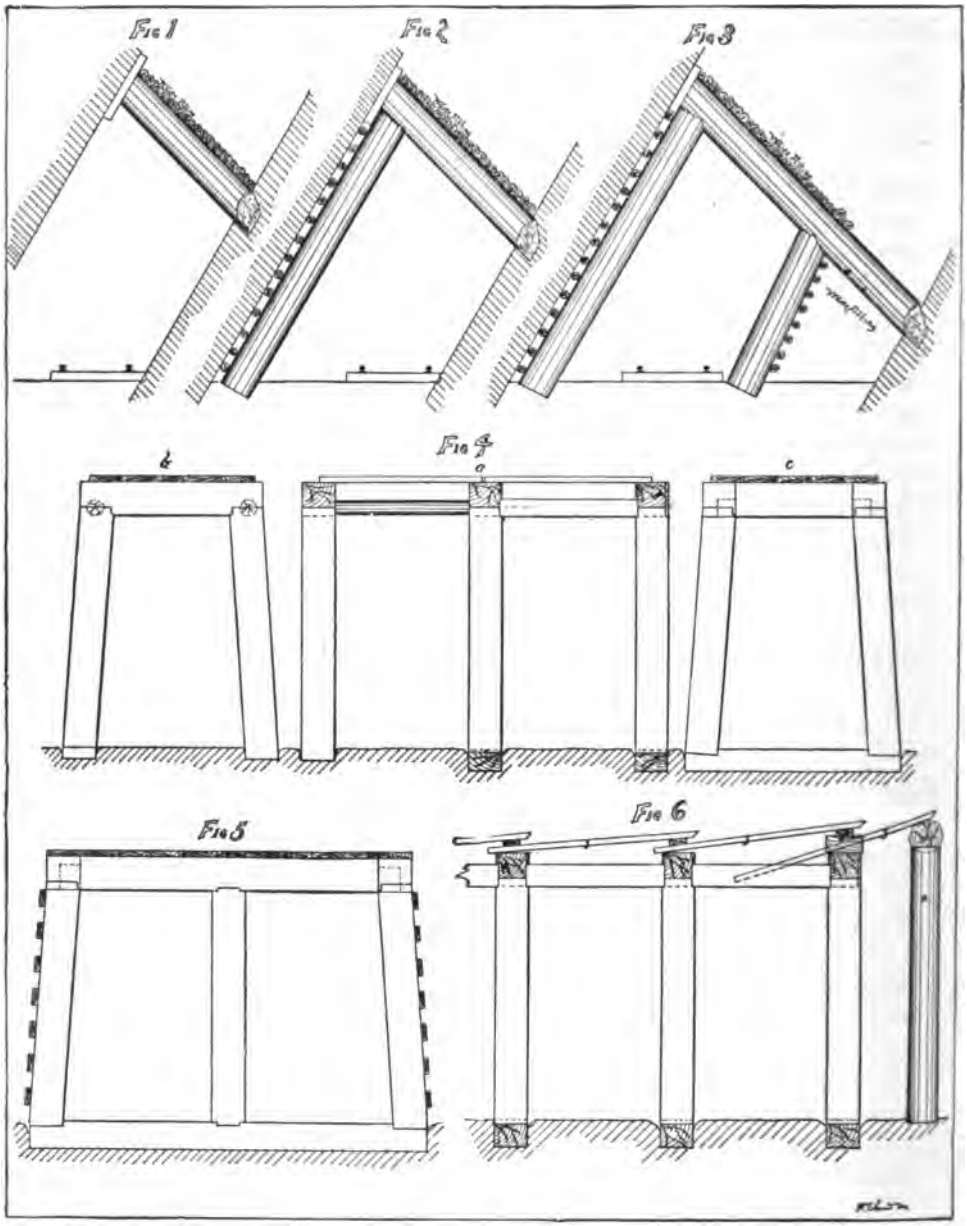


PLATE 8.

In loose ground it is necessary to timber the station by a system of loose spiling, analogous to the method explained in driving levels through similar material.

LEVELS.—Levels include all those approximately horizontal workings through

which mine transportation to and from the working places is carried on, and include adits, cross-cuts and drifts. An adit, usually miscalled tunnel throughout the West, is a horizontal tramway driven either within or from without the ore deposit, and connecting the interior workings of the mine with the surface, while an adit-level in contra-distinction includes only those portions which are contiguous to and immediately connected with the adit, and are operated through it. A drift is a horizontal opening driven longitudinally with and in the ore body, its function being to afford a means of communication along the lode. A cross-cut is that part of a level which is driven laterally across the country formation, or across the ore deposit to connect one part of the mine with another. The methods of timbering these various openings are identical, and will be treated under one head.

The timbering of levels is accomplished by what is known as the single-piece, or the two-, three-, or four-piece set, depending upon whether one, two, three, or four timbers are employed. They are also known as the quarter, half, three-quarter and full set. The pieces are known as the post, if approximately vertical, the stull if inclined, the cap or top piece, and the sill or bottom piece. Upon the latter rest the posts of the four-piece set, the sill being used to keep the feet of the posts of a set from being forced inward by exterior pressure, and also where the ground beneath will not support the weight resting upon the set.

Where the sides of a drift are sufficiently strong, when the deposit is of a width of not more than from 15 to 20 ft., a single piece is frequently used. (See Plate 8, Fig. 1.) Should either of the walls prove to be weak, this single piece is supported at either end as shown in Fig. 2, forming what is called the two-piece set. If both walls are too weak to support the single piece, or should the deposit be of considerable width, posts are placed under both ends, somewhat as shown in Fig. 3, forming the three-piece set. This set is, however, usually made from framed timbers, either round or squared; the posts of the set being of equal length, and the sets nearly or quite of equal size. These sets are held in position by distance pieces, either of poles sledged into position between the sets, or of squared timber, in which case the sets are framed with a hitch to receive and support the ends of the piece. Poles used for this purpose are called "sprags," while the square pieces are known as "girts." The framing of squared or round timbers for this set is practically identical, but round timbers because of their unevenness usually require that a pattern shall be made as an aid to systematic framing. The three-piece set is usually made of round timbers, with the posts set with a spread or slant outward at the bottom as an aid to resist the outside pressure. The feet of the posts are set into hitches of rests at the floor of the level; this also tends to strengthen the set against the inthrust (Fig. 4, *a* and *b*).

The four-piece set is usually framed from squared timbers, the posts being set upright, making the square set, or with a slant outward as in the three-piece set. The set consists of the cap, sill, and two posts, usually carefully framed. In adits the set is often aligned with considerable exactness, and when thus placed the passage presents a pleasing appearance. In adits and cross-cuts the posts are usually given a slant. Often, however, this set becomes an integral part of

the regular square set system as applied to the extraction of masses on the levels and in the stopes of the large metal mines. (See Plate 8, Fig. 5, *a*, *b* and *c*.) The framing of this set often becomes massive, especially in the heavier ground of

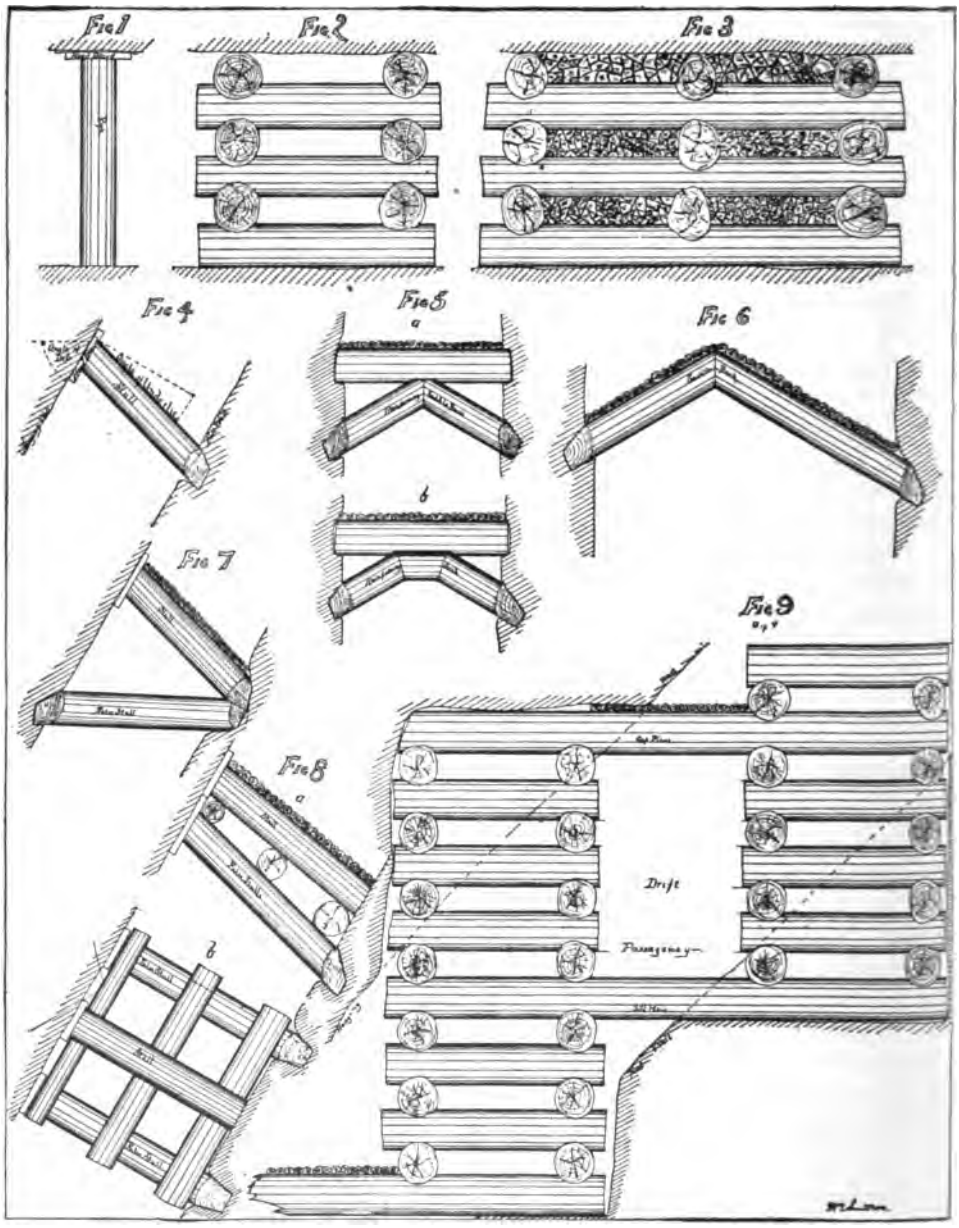


PLATE 9.

adits. A center post is often placed in position for forming the double tramways. (Fig. 5.)

The ground between the sets is held in place by lagging of poles or sawed

plank, which rests at either end upon the timbers. Being of comparatively small strength a lining of this character will yield to unusual pressure, and thus give evidence of incipient crushing that would soon destroy the timbers if not attended to.

Timbering Levels in Loose Ground.—For this purpose the process known as spiling or forepoling is employed, its use being somewhat similar to that described under one-compartment shafts. The spiling may be of sawed plank or of poles of the required length, sharpened at the forward ends, and with their heads protected by an iron shoe when necessary. A set having been fixed in position (see Plate 8, Fig. 6) a bridge *y* is placed upon the cap supported by the blocks *x* at either end. Between this bridge and the cap the spiling *z* is started, sloping upward at an angle. As it is driven forward, piece by piece, the material is picked away from the point of each plank as it is forced ahead a short distance at a time. In this manner the entire shield is advanced through successive small stages until it has been driven forward through about half the distance to its final position, when a temporary false set, *a*, is located to support the spiling. The driving is then continued until the shield has been advanced to its place. The regular set is then fixed in position, a bridge is placed upon its cap, and the false set removed, which allows the spiling to settle upon the bridge. The same process is continued in excavating for the succeeding sets while passing through similar material until more solid ground is reached. When necessary the same process is also applied to the sides of the opening, bridging the posts in the same manner as the cap is bridged and similarly advancing the shield. In very soft ground it is sometimes necessary to employ the same method in carrying the bottom of the level forward. In very soft or running ground the edges of the plank spiling must fit closely against each other, and at the same time the face of the working is retained by breast boards held in position by struts footing against the forward set. These boards are advanced behind the forward edge of the shield, being removed one at a time, and placed farther ahead as the material is removed from in front of it, a longer strut being used to support it in its new position.

Timbering of the Working Places in a Mine.—As regards the process of extracting the valuable materials from their places of deposit there are in use many methods well adapted to keeping the workings open under the varying conditions. Of these the most simple are those employed in the horizontal or bedded deposits, where often the overlying rocks are of such strength as to require little support other than that furnished by the occasional pillar of ground left in place for this purpose. Even the material left in these pillars is sometimes removed, and the roof allowed to fall, when it can be done without injury to future operations.

Posts.—The method of supporting the roof of horizontal deposits by posts or props is almost universally employed, and is the most simple artificial means of keeping open the working places of mines of this character. These posts are formed of sections of trees of various diameters and of lengths up to 20 ft. They are placed in a vertical position, normal to the roof and floor of the deposit, with a flat plank, called a cap piece or head-board, placed upon the top of each prop

to distribute the pressure evenly upon the timber, and to give greater bearing surface against the rock. (See Plate 9, Fig. 1.)

Cribs.—Another method employed is that of cribbing, or, as it is sometimes called, penning. This consists of building up a crib or pen from floor to roof of logs, laid in pairs or in greater numbers across each other. These cribs may be made solid if desired, but this is not often done for practice prefers to make a single or double pen and fill its interior with waste material, that is usually at hand in underground workings, the use of which greatly strengthens the crib. (See Plate 9, Figs. 2 and 3.) The stowing of waste in underground excavations from which the valuable materials have been extracted is often resorted to and forms a solid filling that will with comparatively little subsidence support any pressure. Waste filling is frequently used in connection with, and as adjunct to the various systems of timbering employed in supporting the walls of ore and other deposits. It forms the only permanent and certain means of retaining the walls of ore bodies in approximately their original position.

The metal mines of the West for the most part consist of deposits that dip below the horizontal at varying degrees up to the vertical, the dip of the blanket veins and other bedded deposits depending upon the uplift of the enclosing formations while that of the fissure veins follows the course of the fissures cutting through the earth's crust. Contact veins may present the characteristics of either of the above mentioned classes, and the chambers or isolated pockets of valuable materials may follow certain lines of deposit, or be without regularity or regular form. In size these different deposits vary from the deposits too small to be successfully worked in a commercial way, to immense masses of ore, the extraction of which brings into use all the science of the miner and of mining. Some of the methods in use for timbering these excavations during and after the extraction of the ores, are but the application of old methods to present use, while other systems are distinctly modern, both in origin and application.

Stulls.—Of the older methods there is principally and primarily the stull system, which is but the application of the post of the flat deposits to the use of the inclined veins. Stulls are almost universally employed in mining the smaller veins, with or without waste filling as an adjunct. They consist of sections of trees, pine, fir, oak, or other substantial woods, round, and peeled of their bark. These sections are of all lengths up to about 20 ft., as may be required at the points to be timbered, and in diameter up to about 4 ft. The greater the diameter the greater the strength of the timber. Length beyond certain limits decreases the power to resist pressure, as the piece is more liable to bend or buckle under the weight.

Like the post the stull is placed with a head-board to distribute the pressure, and to give greater bearing surface to the stull in supporting the hanging-wall of the deposit, while the foot of the stull is trimmed and squared to fit more closely into the "hitch" cut into the foot-wall to prevent the timber from slipping from its place. (See Plate 9, Fig. 4.)

Unlike the post the stull is not located in position in a line normal to the walls of the deposit, but at an inclination thereto approximating at a certain ratio

to the dip of the vein, the angle of underlie of the stull (see Plate 1, Fig. 12, and Plate 9, Fig. 4) being about one-fourth of the angle of dip of the deposit, thus:

Dip of Vein.	Angle of Underlie of Stull.	Dip of Vein.	Angle of Underlie of Stull.
10°	2½°	40°	10°
20°	5°	50°	12½°
30°	7½°	60°	15°

The reason for this underlie of the stull is that if the piece were placed at right angles to the walls of the vein a slight movement of the hanging-wall would cause the stull to fall. Also the stull usually carries the weight of waste filling above the levels, and this it would be unable to do if placed perpendicular to the wall, while this weight tends to wedge the piece more tightly into place if placed at an angle above the perpendicular to the walls.

Where the foot wall of the vein is too weak to support the stull in position a false stull is often placed to transpose the vertical weight upon the foot of the stull to a diagonal thrust against the hanging-wall. (See Plate 9, Fig. 7.)

In wide veins stulls are often reinforced so as to enable them to bear both the vertical weight of the waste filling above, and the side pressure of the walls by what is known as the double-stull method. This consists of false stulls placed beneath the stull proper, the former being placed with a foot-hitch and the stull, supported in its position by logs resting upon two or more of the false stulls beneath to either side, footing against the foot-wall without a hitch to receive it. This method is shown in Plate 9, Fig. 8, *a* being the end view, and *b* the plan.

In vertical or nearly vertical veins it is impossible to place the stull in position with the usual underlie, and some method is necessary that will change the vertical weight due to waste above into a diagonal thrust against the walls. This is done as is shown in Plate 9, Fig. 5, *a* by means of the saddle-back system of bracing, and *b* by the arch with key-piece. This saddle-back is sometimes used, as in Fig. 6,, to carry the weight of waste filling above, but it is without valve to resist side pressure.

Penning.—A method of timbering known as penning is sometimes employed in the inclined veins, and is nothing more than the crib of the flat deposits applied to the incline. It consists of cribs of logs built up from the foot wall of the vein to the hanging-wall, which it supports. Occasional longer timbers are used to tie the cribs together, and for the purpose of forming sills and caps for the passageways of the mine, as shown in Plate 9, Fig. 9. This method of using timbers for keeping open the working places of a mine is expensive, and requires quantities of timbering, but in connection with waste filling it is about as permanent as any method of timbering can be. It also has the advantage of a certain flexibility without weakening during movement.

Square-Sets in Stoping.—This system of timbering is peculiarly adapted to the extraction of ores occurring in large masses. In fact the size of the deposit matters little if waste filling be used in connection with the sets. The method requires vast quantities of timber, and the framing of the pieces is no small item of expense, but the handiness of the system is so great, and its adaptability

to all the needs of mining operations in extracting the valuable materials from their places of deposit is such that it has replaced many of the cheaper systems of timbering. Indeed it is a fact that its use in large operations is often found

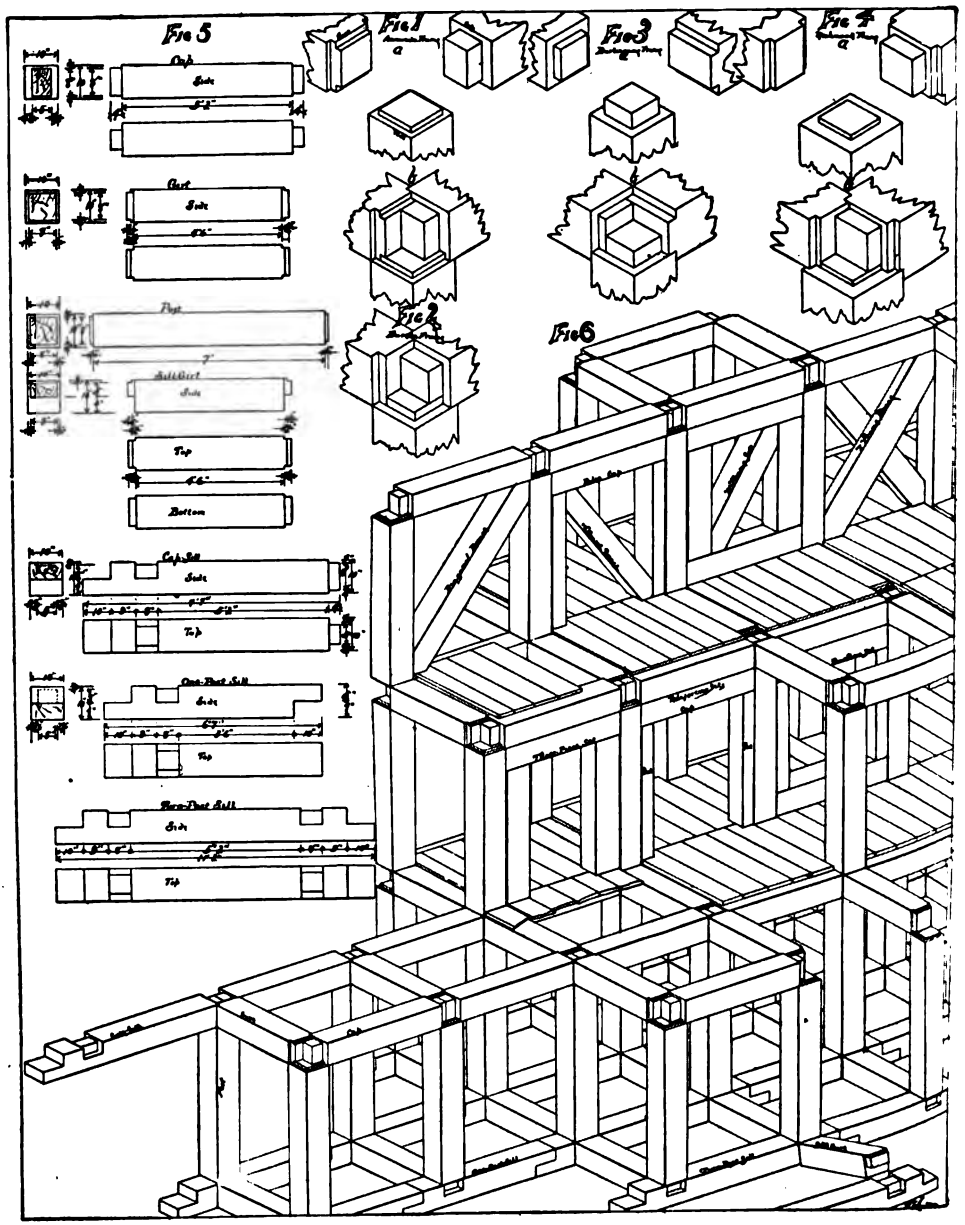


PLATE 10.

to be cheaper in the end than are many of the supposedly more economical methods, and this in spite of the fact that the framing of the sets involves no small item of outlay. Briefly, the system consists in filling up the excavations

resulting from the extraction of ores with what might be termed open blocks or cells of timber that may be added to and extended indefinitely in every direction, lengthwise of the deposit, across it, and between the levels, while the slope of the body matters little for the reason that in following the ores between their walls, sets may be extended laterally outward from the main body of the timbering at any point of the foot wall or left out when the slope of the hanging rock encroaches upon the timbering.

The set is made up of posts, cap, and girt, the former being as usual placed in an upright position, in line with the posts above and below. The cap rests upon the top of the post, and is invariably placed across the deposit, the cap of one set becoming in effect the sill of the set next above; while the girt, which likewise is set upon the post, is located along or longitudinally with the run of the ore body. The sets are, in the best practice, framed for a height of 7 ft. in the clear of the timbers; the reason for this being that this length obtains the full strength of the frame at the same time that it saves timber in the mine, and leaves sufficient height for passage without inconvenience. This height also allows of placing the reinforcing sets in position, and still permits passageway if necessary. Across the deposit the caps are made of a length such as will set the posts 5 ft. apart in the clear of timbers, which gives room for working the air-drills in the breasts of ore, and for other work at end points. Along the length of the ore body the girts are made to separate the cross frames by a distance of 4 ft. 6 in., adding somewhat to the number of such frames and thus giving greater resisting power to the sets against thrust from the side, from which direction the maximum pressure usually acts against the timbers. (See Plate 10, Fig. 6.) In this figure the method of locating the sets is shown in isometric projection, and also that of placing the different timbers of the system.

Usually a heading or drift is first run along the level, as near to the center of the deposit as may be, although this is not essential, and along this excavation the sets are placed from which to build up the more extended timbering. On the floor of this drift are laid the two-post sill pieces, the sill-girts are placed in position as is shown in the figure, then the posts are located, and upon them is placed a cap piece across the drift, while a girt connects this cross frame with the last one placed in position. The frame is now blocked and wedged against the top or back of the drift, and the set is completed. As material is removed from either side of this drift, one-post sills are laid upon the floor of the excavation and penned to the two-post sills already located, a sill-girt is placed, the post set upon the framing fitted to receive it, and cap and girt hold its top in position. It is then blocked and wedged firmly against the ground above, and also from the side. A 2-in. plank roofing is laid from cap to cap above the set, which performs the functions of a floor for the set next above as the timbering is carried up. When the foot wall of the deposit is reached, the sets are carried up along its slope by means of the cap-sill, a timber which combines the functions of the cap and the sill, also shown in position in the figure. Upon these cap-sills as a foundation are built up a new line of sets, and the process is carried on to any extent by repetition. Above the sill floor, as the ground is excavated, posts are set into the gains formed by the framing of post, cap and girt beneath,

and the timbers are continued upward, outward, and lengthways as far as may be necessary. The sets from one level are carried up to those of the level above, when short sets are placed in position to carry the weight of the upper framing.

Reinforcing Sets.—When the timbering is carried into unusually heavy ground, or where it is forced to carry a great weight, it is frequently necessary to reinforce the sets across the deposits in line with the greatest pressure, and this is done by what is known as the diagonal brace, the three-piece set, the full or four-piece set, the "N" frame brace, the "N" frame set, and the "X" frame brace. These are all shown in Plate 10, Fig. 6.

Ore-Chutes.—In delivering the ore from the stopes above a level to that level so that it may be loaded into cars and carried to the shaft for conveyance to the surface, a storage bin and passageway combined is necessary, and this is obtained by lining one of the sets as it is carried up from the level by a close lining of 2- or 3-in. planking, preferably the latter. These bins are designated as chutes, and the framing at the bottom of them whereby the ore is delivered into the cars is called the gate of the chute. This is made by constructing an inclined flooring, with sides, that shall project beyond the side of the chute into the tramway sufficiently far to allow the rock to fall from it into the car. Cross timbers are placed across the chutes at varying heights of about 30 ft. in order to break the fall of the rock so that it shall not destroy the gate frame. In keeping the chutes near the center of the deposit they are as often as is necessary offset toward the floor-wall on an incline from one line of sets to the next lateral set on the floor above.

Waste-Filling.—In average ground no system of timbering will long sustain the walls of large excavations, and waste rock from the vein and walls, especially the hanging-wall, is employed for filling up the spaces between the timbers in order to make a solid filling that shall hold the ground in place. Passageways are strongly reinforced, and the flooring removed from the different floors when waste is thrown into the excavation from above.

Methods of Framing.—There are several methods of framing the timbers of square sets in order that they shall come together from the six directions in a manner best suited to the needs of the occasion. Plate 10, Fig. 1 shows the method of cutting the timbers in use in the Anaconda mine, Butte, Mont., *a* giving the isometric perspective of the post, cap, and girt as shown separately, *b* showing their appearance when joined. The Eureka method of framing is shown in Fig. 2, and is very similar to the Anaconda frame, the latter having one more section cut from each side at each end of the girt. Both of these methods are used for opposing pressure from the sides of the veins, the caps abutting against one another to secure the greatest strength from the timbers, but by transposing the timbers of this set, so that the posts shall rest end on end upon one another, vertical pressure may be best resisted. This is shown in the Burlingame frame (Fig. 3), *a* separated and *b* joined, which is the Eureka framing with the post of the latter forming the girt of the former, the cap of the Eureka the post of the former, and the girt of the Eureka the cap of the former. Fig. 4 presents the Richmond frame which is a strong but complicated and expensive joint in which the cap and post are framed from 12-in. timbers and the girt of 10-in.

pieces. Usually the Anaconda and Eureka frames are made of 10-in. timbers, the reinforcing sets and solid waste-filling being depended upon to hold up the ground rather than by increasing the cross-section of the timber. Frequently, however, the upper levels of a mine are supported by 8-in. pieces, but this is infrequent at the deeper levels. The Eureka framing is now used in some of the Anaconda mines as being the simplest and cheapest method of framing. Plate 10, Fig. 5, furnishes the details of framing timbers for the Anaconda set. The square-set system of timbering was originated to meet the needs of the situation as developed in the workings of the Ophir mine, on the Comstock lode, Nevada, by Philip Deidesheimer.

Important details connected with the methods of timbering herein described, and other systems now in successful operation among the metal mines of this country are excluded from this necessarily abridged article.

ORE DRESSING.*

By Robert H. Richards.

General Principles.—The preparation of ores for the smelter by mechanical means, whereby the valuable minerals are concentrated into smaller bulk and weight by the separation of some of the waste, or whereby two valuable minerals are separated from each other, is called ore dressing.

The main advantages to be derived from ore dressing are that the rejected material does not have to undergo any further treatment and, consequently, costs of this and its proportion of transportation charges are eliminated; furthermore minerals undesirable in future operations may be separated. Thus, not only are costs reduced, but the profits increased.

Ore dressing makes use of the physical properties of minerals; the difference in behavior between the valuable minerals and the waste minerals affords methods for the separation of the former from the latter. Physical properties of interest in ore dressing are: hardness; tenacity and brittleness; structure and fracture; aggregation; color and luster; specific gravity and settling power; adhesion; greasiness; magnetism; change in mechanical condition by heat from non-magnetic to magnetic; change in mechanical condition by heat from dense to porous; decrepitation by heat. Though hardness does not effect the tendency of the mineral to produce fine slimes in crushing, the harder the mineral the greater the wear upon the crushing machine. Sliming, however, depends to a certain extent on the brittleness of the mineral. The tenacity of certain minerals makes them difficult to crush. The shape of mineral particles after crushing depends largely upon the structure of the original mineral and has an important bearing on the settling of the particles in water or in air. Where minerals occur in separate masses or in large crystals the conditions are favorable for a complete separation. On the other hand, difficulties are encountered when the minerals are in granular or acicular structure thoroughly intermingled with the waste. Laminated structure of the mineral and waste also causes difficulty in separation.

The distinctive coloration or luster of a mineral tends to an easy separation

* This paper summarizes in a general way many of the important chapters of Prof. Robert H. Richards' magnificent treatise on *Ore Dressing*, now being published by The Scientific Publishing Company, and which is unquestionably the most exhaustive and practically useful work on the subject that has yet been written in any language. The reader is referred to this book for more detailed information.

Editor The Mineral Industry.

in hand picking. A difference in specific gravity affords one of the surest means of separating minerals from each other. This property may be employed in two ways; namely, as affecting settling power in liquids or air; as affecting momentum. The separation of minerals through their difference in settling powers is due to the heavier mineral settling faster than the lighter, and of two particles of different specific gravities and the same settling velocity, that with the higher specific gravity will be of smaller diameter than the other. The ratio between these two diameters will have approximately constant values for each set of conditions, and these are called settling ratios. The length of trajectory of a particle to which high velocity in a horizontal direction has been given depends upon the specific gravity of the particle. The greater the specific gravity the longer the trajectory provided the size and shape remain the same. With particles of differing specific gravity but with the same trajectory, the particle with the higher gravity will be of the smaller diameter.

The relative adhesion of minerals to a surface may effect a separation; thus the diamond adheres to a greasy surface while quartz does not, and gold may be separated from quartz particles by an amalgamated copper plate to which gold particles adhere.

Greasiness is the term used to express the tendency which certain minerals have to remain dry when in contact with water and to float on its surface. With certain minerals this tendency is marked and may be regarded as a difficulty to be overcome.

Certain minerals are readily attracted by the magnet while some are only weakly magnetic and others are uninfluenced. By using properly constructed magnets this property may be made of great value, not only separating the magnetic from the non-magnetic, but those that are more magnetic from those that are less so. After certain minerals are heated they become magnetic and this property also may be utilized in separation.

Heating certain minerals such as pyrite causes them to lose a volatile constituent and to become of lower specific gravity, when they can be separated from minerals whose specific gravity was equal to theirs before heating. Heating of other minerals causes them to decrepitate after which screening may separate them from minerals which do not.

Ore dressing may be divided into two parts: severing and separating. The severing or crushing detaches the valuable mineral from waste, but as they are mixed together the operation of separating the values from the waste must follow.

Breaking and Comminuting.

Preliminary crushing is done by blasting in the mine, calcining for friability with or without quenching, by sledging, spalling and cobbing hammers, steam and drop hammers, rock breakers, log washers and wash trommels.

While blasting in the mine considered strictly is outside the province of ore dressing, still the latter may be affected by the degree of division of the ore. The fine division caused by high explosives, not only may cause the production of slimes, but in narrow veins may introduce an excess of barren rock.

When an ore is heated by fire the minerals are cracked and fissured in all directions, and if they are dropped into water when hot, their friability is increased. This increases the capacity of the crushing machinery, but also the production of slimes.

For sledging, spalling and cobbing, hammers of various shapes are used, varying from heavy sledges and spalling hammers requiring the use of both hands to the lighter single-handed cobbing hammer. This latter is rarely used in this country.

Steam hammers of great weight were formerly used at Lake Superior for crushing lumps which were too large for the rock-breakers, but have since been done away with although a small steam hammer is used in places for freeing mass copper from adhering rock. Drop hammers raised by a rope and running between guides are used at Lake Superior for the same purpose as the light steam hammer.

While hand breaking is more expensive than machine breaking, metallurgically it may be more economical, as a smaller percentage of fines is produced by it. Its use moreover being governed by intelligence, a cleaner severing of ore from waste occurs.

Rock Breakers.—There are two chief classes of machines: (1) jaw-breakers, crushing intermittently; (2) the spindle or gyrating breakers, crushing continuously. The jaw-breakers may be further divided into three classes: the first of which, breakers with the greatest movement on the smallest lump, is represented by the Blake breaker. The details of machines manufactured by different firms differ greatly, as when the Blake patent expired in 1878 manufacturers began to seek improvements upon the original. Out of 142 Blake breakers in mills visited by the writer, 114 are of the pitman pattern and 28 are of the lever pattern. Out of 136 Blake breakers, 116 are with solid cast-iron frame while 20 have sectional cast-iron frames. The second class includes jaw-breakers with equal motion on coarse and fine lumps, such as the Forster breaker and the Blake multiple-jaw breaker.

The Dodge breaker is one of the leading examples of the third class of jaw-breakers, which have the greatest movement on the largest lump with a swinging jaw pivoted below.

Considerable discussion has been given to the relative merits of the Blake and Dodge types of breakers. There seems little doubt that the Dodge gives higher percentage of fines than the Blake with the same movement at the throat, especially when run at nearly full capacity. This may be accounted for by the fact that the Dodge is putting in more work at the mouth than is needed to prepare the lumps for the throat. The machine is acting under the condition of choked crushing, owing to the excess of fines combined with a large movement at the mouth. Each machine has its place, however, and the machine to be used will depend upon the special conditions of each case.

The Gates breaker is typical of that type of gyrating breakers which has the greatest movement on the smallest lump. It differs from the Comet and other breakers mainly in details of construction, as does the McCully breaker.

The Lowry National breaker is representative of the class of gyrating breakers which have the greatest movement on the largest lump of rock.

ROLLS.—These machines receive ore from the rock breakers or middlings from the jigs and crush to sizes that are suitable for separating the valuable minerals from the waste in the washing process. As they produce a very small proportion of fines they are especially adapted to crushing galena, chalcopyrite, blende and all the soft brittle ores, producing from them a minimum amount of slime. These machines have their limitations, however, as when crushing gold ores the gold is not sufficiently brightened to allow of ready amalgamation, nor do they readily crush fine enough to liberate the gold from the accompanying minerals. In treating malleable minerals such as metallic copper, they are found unsuitable, as the copper is separated in forms which cause heavy losses in dressing. There are some minerals, however, which on account of their malleability are not crushed as fine as the accompanying rock, so that separation may be made by screening.

Records of practice show that 63·5 mm. or 2·5 in. is a maximum feed for rolls, while lumps are rarely fed larger than 38·1 mm. or 1·5 in. Rarely is the feed finer than 2·12 mm. or 0·083 in. diameter, though a certain amount of fine dust is often fed with coarser material in order to obviate screening.

The efficiency of rolls depends largely upon the character of the shells. These are made of cast iron deeply chilled on the outside, cast steel, rolled or hammered steel, and chrome and manganese steel.

Out of 86 sets of rolls in operation in various plants 52 had steel shells, the remaining 34 being chilled iron. This latter metal has the advantage of low first cost and the worn shells have a market value. The wear, however, is uneven and the life short. Another difficulty is that they cannot be readily turned true, owing to their hardness. Wrought iron and mild steel have the disadvantage of flowing or beading at the ends. Cast steel is of medium price and has a medium life.

Either rolled or hammered forged steel is the most desirable material that exists, though it costs from 6·5 to 10c. a lb. against 2 to 4c. a lb. for chilled iron. Chrome steel has given good results, as has also manganese steel. The latter in one case wore down to 0·5625 in. thick, having had a remarkably long life. Manganese steel, however, has a tendency to work loose on the cores owing to expansion when working. This sometimes causes the shell to split.

The greatest care is needed to make the rolls wear evenly, and especially prevent flanges from forming at the ends. While the flanges may be removed by turning, or buffing down with an emery wheel it is far better to prevent their formation. This is done mainly by controlling the feed. Movable bars are placed in the steep sloping feed trough, which permit of the ore being fed to any desired part of the roll. Another way is to set the rolls one day with a lap of 0·25 in. on the end; next day change the lap to the opposite end. Automatic feeders are commonly employed.

In order to compensate for the wear of the shells and to regulate the size of the material crushed one of the rolls is mounted on movable pillow blocks. They are held apart at the required distance by putting in shims of equal thick-

ness on each side or by the use of compression bolts on the springs. The two movable boxes may be independent or they may be connected by a rigid frame.

Another way is to support on two vertical levers swinging from pivots, the two movable boxes, called swinging pillow blocks, and to hold the rolls up to their work by tension rods and springs, holding them apart by lock nuts on the tension rods.

Springs are used to give mobility when a drill or any other piece of steel happens to be fed with the ore, thus preventing the rupture of the machinery.

Rolls driven by belts and gears, are called geared rolls; or by belts alone, and called belt rolls. The former appear to be preferred for slow speed coarse crushing, the latter for high speed fine crushing.

Speed of Rolls.—The periphery speed varies greatly, as does the opinion of various writers on this subject. Mathematically speaking, the same pair of rolls must run twice as fast to crush 0·5 in. cube down to 0·25 in. as they would to crush 1 in. cube down to 0·5 in., the amount crushed being the same in both cases.

Theoretically, increase of speed, provided the reduction in size is little enough, can be made to almost any extent, but, practically, high speed with much reduction, will give trouble owing to the refusal of the rolls to nip the lumps. The latter fly back until a dangerous amount collects and then the rolls choke.

Instances have occurred where by reducing the speed of a pair of rolls the capacity has been increased and the wear reduced.

Since the wider the space between the rolls the more favorable is the angle of nip and the greater the tendency to run smooth and avoid slime making, the ratio of this space to the size of the apertures in the following trommel is important. I am inclined to the opinion that 1 to 1·25 would be the proper ratio; for example 0·5 in. space between rolls would crush a large percentage fine enough to pass through a 0·625 in. hole in the trommel.

The quality of crushing depends to a considerable extent upon the way rolls are run. Rolls when run slowly upon a given quantity of ore, may be so crowded that the fine particles cannot tumble away from the coarse as soon as sufficiently broken. In consequence such fine particles may be subjected to still further crushing due to the action of the particles one upon another. This condition may be called "choke" crushing. If now the speed of the rolls be gradually increased the percentage of fines will gradually decrease until a speed is reached at which the particles are treated individually, and there is plenty of room for the crushed fine ore to drop away from the coarser part under the acceleration of gravity and so escape further fine crushing. This condition may be called "free" crushing and it is the condition under which the maximum coarse and minimum fine material will be made.

Rolls acting under "free" crushing conditions stand pre-eminently at the head of the list among crushers for producing a large proportion of coarser sizes with a small proportion of slimes. "Free" crushing when practicable is the more advantageous of the two methods. It cannot be used, however, for crushing very fine on account of the impracticability of maintaining the space

small and the surfaces true. For fine work, the feed must, therefore, be increased so as to produce "choke" crushing and even this will not give a high efficiency.

Capacity of Rolls.—The capacity of, or the quantity crushed by, rolls is the number of tons that can be crushed from a given size to pass through a certain size of hole in a given time. In free crushing, provided the spring is sufficiently stiff to hold the rolls to their work, provided also that the angle of nip is favorable, the capacity is dependent upon the speed, the width of face and the space or distance the rolls are set apart. In choke crushing the capacity depends upon the measures given above and upon the pressure. The greater the pressure, the greater will be the reduction in size; for example, in a certain mill the pressure upon the fine rolls is governed by periodical sizing tests of the crushed ore. If the rolls are making too much oversize more pressure is put on.

STAMPS.—Stamps are particularly useful in crushing ores of gold and silver preparatory to amalgamation and concentration, and native copper rock and cassiterite, chromite, graphite and other ores preparatory to concentration. They are particularly useful where fine crushing is desirable, but are not suitable where the valuable mineral is finely disseminated and friable.. They may be divided into three classes: (1) steam stamps; (2) pneumatic and spring stamps; (3) gravity stamps.

Steam Stamps.—While six varieties of steam stamps are in use, they differ but slightly, and that in matter of detail. It is necessary for the steam stamp to have a heavy mortar bed. The earlier type of stamp had a light mortar bed, which absorbed only a small portion of the effect of the blow and was placed upon spring timbers of oak. These latter requiring frequent renewal were a source of great expense, so that in the Allis and the later Leavitt stamp a very much heavier mortar bed weighing from 60 to 120 tons is used. This obviates the necessity for the spring timbers.

The stamp frames in earlier designs were of wood while the more recent ones are of cast iron. The Leavitt frame, which may be taken as typical of modern construction, consists of four posts of cast iron, standing upon and bolted to pedestals on the base. These posts lean toward the center so that the width of the top is 44% of the width of the bottom. The frame is in three sections, lower, middle and upper, with suitable cross bars, flanged and bolted together. The middle part carries the guides, the upper carries the dash pot and piston rod guide, the cylinder and valve gear. Some frames allow screens to be placed on all four sides of the mortar if desired, but others permit only of their being on two. The mortar is a cylindrical pot of cast iron upon which rests the housing, in which the screens are placed. In the later type of stamps, the four sides of the mortar housing slope downward and inward to suit the screens, besides giving room for the four posts. The housing has liners to take up wear, which may be renewed.

Punched steel screens are generally used, which may be turned upside down when worn on the lower side. The size of the perforations varies from 0·1875 in. to 0·75 in. and the screen area varies from 2,340 to 5,984 sq. in.; 3,100 may

be taken as the general average. The life depends upon the material and the ore and varies from 12 to 60 days. A cast steel plate was used in one instance and lasted 42 days, crushing 8,820 tons. Chilled cast iron is found to be the best material for shoes and dies and other parts subjected to wear. The life of the shoes is necessarily short, ranging from 27 hours for armor plate steel to 60 or 70 hours for chilled cast iron. Forged steel and manganese steel have been tried, but with results less advantageous than the chilled iron.

The capacity of the steam stamp on Michigan native copper ore varies from 150 to 300 tons per 24 hours, being greater when crushing soft amygdaloid rock than when crushing hard conglomerate. The accompanying table shows the re-

Size of Screen in Mortar. Inches.	Size of Trommel Hole. Inches.	Capacity of Stamp per 24 Hours. Tons.	Copper in Tailings of Mill. %	Size of Screen in Mortar. Inches.	Size of Trommel Hole. Inches.	Capacity of Stamp per 24 Hours. Tons.	Copper in Tailings of Mill. %
0·1875 in. round.	None.	341	0·634	0·500 in. round.	0·25	427	0·625
0·250 "	None.	375	0·666	0·5625 "	0·25	439	0·593
0·3125 "	0·25	346	0·616	0·250×0·625 in. slot	0·25	421	0·539
0·375 "	0·25	396	0·617	0·3125×0·625 "	0·25	402	0·550
0·4375 "	0·25	384	0·613				

sults of tests of a stamp in a mill crushing conglomerate with varying sizes of screen apertures. In the first two of the tests the pulp that issued from the mortars passed directly to hydraulic classifiers and to jigs as in the usual practice of Lake Superior mills; in the rest of the tests, however, the pulp from the mortar passed first to a little hydraulic classifier with one spigot, removing only a small percentage of the total pulp as rich copper concentrates; the remainder overflowed to a trommel with 0·25 in. hole, the oversize being returned to the mortar. This modification as is shown in the table causes increased capacity and a decrease in the tailings loss. On soft amygdaloid ore using a 0·5625 in. round hole mortar screen and a 0·25 in. trommel, 487 tons were crushed in 24 hours, and the percentage of copper in the mill tailings was 0·196. This system has been introduced in the new Osceola mill and the new Arcadian mill. The actual horse power consumed depends upon the type of stamp varying from 81 h. p. in the old Ball stamp to 171 in the new Allis. The Leavitt stamp requires an expenditure of 149 in one mill. The efficiency is practically constant, however, varying from 1·745 to 1·852 tons per h. p. in 24 hours. The efficiency of the blow varies from 63 to 69%. The capacity does not differ greatly from the average results of 24 mills using the California gravity stamp, which crushed 1·79 tons per h. p. in 24 hours. However the California stamps are crushing to an average size of 0·7 mm., while the steam stamps are crushing through 4·76 mm.

These steam stamps are the most powerful crushers known and seem indispensable on native copper ores. On brittle ores, however, engineers are generally of the opinion that they slime the ore too much. For this reason they have been replaced by rolls in certain mills not working on native copper ores, while in another case the whole of the slimes are settled and smelted. Sizing tests on Butte, Mont., ore when crushed by steam stamps showed a large percentage which would pass a 0·12 mm. screen, which assayed comparatively

high in copper and silver and low in silica and gangue minerals. The coarse ore that remained on a 2·8 mm. screen assayed 2·3% Cu and 8 oz. of silver, while the fines assayed 4·3% Cu and 18·9 oz. of silver.

Although small steam stamps, such as the Tremain, the Sharpneck and the Hammond, are used in crushing gold ores, the large steam stamps have not proved successful. A seven months' trial on gold ore at the Black Hills, S. Dak., showed that the large stamp crushed too fast for the number of amalgamated plates with which it was provided, also that the jar from the stamp loosened the amalgam on the plates; but the former difficulty could have been overcome by centrifugal pumps and banks of plates one above another, and the latter by suitable framing. The stamp used had an 11 in. cylinder, and 22 in. stroke and made 95 strokes per minute and used 900 gal. (3·75 tons) of water per ton of ore. The steam pressure was 85 lb. per sq. in. and the capacity was 125 to 135 tons per 24 hours, crushing through a No. 7 needle (0·024 in.) slot screen. With a steam pressure of 110 lb. the capacity rose to 192 tons per 24 hours. The screen lasted six days. A finer screen caused the ore to bank in the battery and break the screen. On account of scouring action no inside plates could be used. The outside plates were 12 ft. long, 4 ft. wide and sloped 1·75 in. per ft. The capacity per unit of fuel was the same as with the Homestake, S. Dak., (gravity) stamp, but the latter crushed finer, which is an advantage.

The suggestion has been made to use a large steam stamp to reduce gold ore to say 0·1875 in. in size and a Huntington mill, or some similar mill, to take the coarser portion and bring that down to gold mill sizes. This would make a much more compact plant than the usual gravity stamps.

Pneumatic and Spring Stamps.—Pneumatic stamps were used in several native copper mines in Lake Superior but were discarded. The Krause pneumatic stamp was in use in the Calumet and Hecla to crush included grains of native copper from 0·1875 in. down to 0·0625 in., but was replaced by Heberli mills which required less repair and had greater capacity. This stamp is somewhat similar to the Husband and Sholl stamps. The Husband stamp using 93 i. h. p. crushed 100 tons of moderately hard Cornish tin ore through a No. 36 screen; the cost of repairs, including shoes and dies was 9c. per ton. Morrison's high speed stamp is a recent invention and combines the principles of the gravity and pneumatic stamps. In spring stamps the cylinder and piston of the pneumatic stamp is replaced by some form of spring. The Ellenbecker stamp was used at the Calumet and Hecla mill for many years, but owing to the difficulty which arose from the heating of the spring, it was discarded.

Gravity Stamps.—As these have been frequently treated in detail in THE MINERAL INDUSTRY, it is unnecessary to enter into a full description.

Concrete mortar blocks have been lately introduced and are cheaper and more durable than those made of wood. In the 300 stamp addition to the Alaska-Treadwell plant on Douglas Island, the anvil mortar blocks weigh 7 tons, and rest on a concrete foundation. It is claimed that they lack the resilience of timber, though it has been proved at Lake Superior that the capac-

ity of steam stamps has increased with the solidity of the foundation without causing increased breakage or wear. The Banner mine near Oroville, Cal., has a 20-stamp mill with stamps weighing 1,065 lb. with solid concrete mortar block, and it is expected that 5 tons will be crushed per stamp per day, being an excess of 0·5 ton over the maximum results with wooden blocks. A rubber sheet 0·25 in. thick is interposed between the mortar and the concrete block.

The weight of the mortar should be proportional to the weight of the stamp to get like conditions of impact. A ratio of mortar to stamp of about 6 to 1 is practically a constant, although in cases where extremely heavy mortars or very light stamps are used, the ratio is as high as 8 or 9 to 1. A narrow, somewhat deep mortar is the best combination to obtain rapid stamping and good battery amalgamation. The space behind or in front of the shoe must be larger than the maximum size of ore fed to prevent the stamp from being wedged against the side. Mortars are now commonly provided with chilled cast-iron lining plates to take up the wear.

The method of punching screens is not without influence on capacity and quality of product. Clear punched round or slot holes are preferred to buhr punched for coarse or medium crushing, but for fine crushing buhr punched holes are preferred, as thicker metal may be used. Slots are punched either vertically, diagonally or horizontally and either in line or staggered. Round holes are arranged in rows making either 60 or 90° with each other, the former giving 1·154 times as much percentage of opening as the latter when the diameters of the holes and the spaces between them are the same. There seem to be three facts that are all-important for the ore-dresser to know in deciding upon the kind of screen to use: (1) the exact diameter of the hole controlling the size of the pulp; (2) the percentage of opening showing the freedom of the discharge and the strength of the screen; (3) the thickness of the plate or wire indicating the strength of the screen. The screens may be made of Russia sheet iron, steel plate, burnt tin plate and unburnt tin plate; besides these tin bronze, phosphor bronze, aluminum bronze and copper have all been tried. Cloth screens are commonly made of steel or brass wire; phosphor bronze and aluminum bronze wire have also been tried. The life depends much upon the acidity of the water, which is often greater when surface ores are crushed. Tin plate screens, either burned or unburned, have the advantage of low first cost. Brass cloth has moderately low first cost and resists acids, but has not the strength of iron or steel and must therefore be made with larger wire. The various bronze wire cloths have long life but high first cost. Results at a number of mills visited show that the cost for screens per ton crushed varied from a minimum of 0·163c. to a maximum of 3·390c. for steel cloth and 6·857c. where brass wire cloth was used.

The quantity of water consumed varies from 17·97 tons as a maximum, to 2·92 tons per ton of ore stamped; 21 mills show an average consumption of 7·11 tons.

The advantages and disadvantages of gravity stamps when compared with rolls may be expressed as follows:

Advantages.—(1) They are simple in design and simple and comparatively

economical in operation, not requiring skilled mechanics. (2) They not only crush fine at one operation, but they successfully combine this crushing with amalgamation. They are more successful than any of their competitors in brightening gold for amalgamation. (3) They are adapted to the treatment of a great variety of ores, and in many cases, give better results than any other process. (4) There is no great loss of power by friction. (5) A disabled battery may be hung up and repaired without delaying its associates.

Disadvantages.—(1) The first cost is large and transportation charges are high. (2) The strains are excessive, necessitating heavy frames and large, expensive foundations. In consequence, they are cumbersome and require a great deal of space. (3) They are not a positive machine; that is, the power consumed is a constant, whether the rock is broken or not. (4) There is danger of over-stamping and sliming. (5) They are not well adapted to dry crushing, because of their small capacity when so used. These disadvantages are not so important as they at first appear. The excessive first cost and cost of transportation appear much smaller when based on the total amount of ore crushed. Sliming the ore may be avoided to a great extent by making suitable adjustments, and in some cases sliming may be an advantage. In spite of the frequent attempts to replace stamps by other machines, their advantages so greatly exceed their disadvantages, that their continued use for gold mills is quite certain, especially for hard and free-milling ores; though for some ores other processes are more suitable.

Pulverizers, other than Gravity Stamps.—The Patent Office Records contain descriptions of many machines which have been designed to replace gravity stamps for crushing gold ores or jig-middlings as well as many others for grinding soft substances, such as phosphates, gypsum and talc. Yet in 1895 and 1896 out of 723 establishments in California 551 still retained the stamp. One hundred and eight Arastras and 43 Huntington mills were employed, the balance being divided between various patented designs. For grinding phosphates and other soft materials, however, the gravity stamps are not used.

These other pulverizers may be divided into four classes, according to their mode of attack upon the rock: (1) abrading or true grinding; (2) pressure; (3) blow upon an anvil; (4) blow in space. They may also be divided according to their construction. The arastra is the best example of the first class. Its capacity varies according to size and the fineness to which it is necessary to grind the pulp. At Guanajuato, Mexico, 12 ft. arastras worked by mule power grind from 600 to 1,100 lb. of ore in 24 hours; at Zacatacas 1,000 lb. is crushed in 13 hours, while at San Dimas 1,500 lb. only is crushed in three days.

The amalgamating pan has been evolved from the arastra and has developed to the modern combination pan. It consists of a flat bottom pan with an iron cone in the center with high sides nearly or quite vertical and with a horizontal revolving disk called a muller. The pan has three important duties to perform; it grinds the ore, furnishes iron for the chemical reactions of the process; and mixes the mercury with the ore that amalgamation may follow. This mixing or circulation of the pulp is the most important feature of the

operation. The pulp current is generated chiefly at the bottom of the pan and is effected by the centrifugal force due to the revolution of the muller and by oblique shoe and die channels. Die channels are frequently omitted and as the centrifugal force is weak in the center, this space is occupied by a cone. Where thick pulp is used the cones are narrow with outward channels in the shoes and dies, if die channels are used. In those mills which use thin pulp, the cones are wide in order to obtain the needed activity of pulp. There are three mill processes which employ pans: the Washoe, the Reese River and combination processes. The first and third processes are conducted either with or without grinding, while the second is conducted without grinding. The tank system is commonly employed, although the continuous or Boss system has many advocates. The horse power consumed is naturally higher when grinding than when amalgamating. By manufacturers it is estimated that from 3 to 5 h. p. is required for amalgamating and from 5 to 10 h. p. for grinding. The loss of iron when free-milling ores alone are ground is from 6 to 10 lb. per ton of ore, including that consumed chemically.

In preparing pulp for the amalgamating pans, the Jenisch ball mill, which is similar to the Gruson mill, has replaced stamps in the mill of the Commonwealth Mining & Milling Co., of Pearce, Ariz. A No. 5 mill using 12 h. p. is said to crush 23 tons in 24 hours through a 40-mesh screen. The pulp is said to be much more uniform than that from a stamp mill and the wear, it is thought, will prove low. The Dodge and Tustin mills are similar to the Gruson in principle, differing, however, in details of design. Generally speaking this class of ball mills is especially adapted to crushing gold ores where the sliming of tellurides is to be avoided, where the pulp is to be treated by one of the leaching processes or where concentration is the chief method of beneficiation.

The modern form of Chile mill has been used as a fine grinder. At the Haile gold mine in North Carolina a mill with rollers 4 ft. in diameter and with 8 in. face, weighing about a ton, having white iron tires 8 in. thick, driven at 40 revolutions a minute, crushed 90 tons in 24 hours of hard tough quartzite of 0·25 in. diameter to 40 mesh. A trommel was used and the oversize was returned by an elevator. Of soft ore 240 tons per 24 hours have been crushed with a wear of 12 lb. of iron per ton of ore.

The Bryan roller quartz mill is a mill of the Chile type with three rollers. At one mill three 5-ft. Bryan mills at 37 revolutions per minute crushed 24 tons in 24 hours of jig-middlings with the expenditure of 7·5 i. h. p.

The Huntington centrifugal mill is extensively used for crushing gold ores and jig-middlings. Although acting on the principle of rolls it has not their positive spring, so that it is necessary to feed ore not exceeding 0·75 in. in size through an automatic feeder run independently. The advantages claimed for it as compared with gravity stamps are low first costs, less freight charges, small cost of erection, small amount of power and that it is a good amalgamator, making much less slimes than the stamp mill. The running cost, however, is high when compared with rolls.

The Griffin roller mill consists of a single roller suspended upon a vertical

axis and rolling on the inside of a die ring. The roller revolves about 200 times per minute on its own axis. A solid foundation is necessary and from 15 to 25 h. p. are required according to the work it does. The mill is constructed for dry or wet crushing. When used for dry crushing, a current of air forces the material out through the screen to a screw conveyor and dust chamber. The mill is found to crush finer than the screen size would indicate. J. R. De Lamar reports that a mill crushed 20 tons of extremely hard rock to 40 mesh and finer without screening. The screen used was one of four meshes. The mill makes excellent pulp for cyaniding, better than either rolls or stamps. With rolls the tailings assayed $4·65, but with the Griffin mill they ran from $0·89 to $1·65 per ton.

The Stedman or Carr disintegrator is used for crushing soft material and when used in Germany under the name of the Brink & Hübner, yearly statistics showed the wear to be about four times that of rolls when crushing jig-middlings.

Laws of Crushing.—In discussing this question of crushing under different conditions there are four lines to be considered: (*a*) Compressive strength of stone; (*b*) extent of crushing desirable; (*c*) work or power required for crushing; (*d*) comparison of various machines.

There are four ways that the force may act in crushing rock: (1) By direct pressure as in rolls where there is a strong force acting at low velocity; (2) by a blow on an anvil, as in stamps where there is a medium force acting at a moderate velocity; (3) by a blow in space, as in the Carr disintegrator where there is a weak force acting at high velocity; (4) by grinding as in the amalgamating pan. In the first three cases the force acts perpendicularly to the surface to produce rupture by compression; in the last case it acts obliquely, producing rupture by compression combined with shearing.

It is an advantage in most cases, except where the mineral is all finely disseminated, to crush first to a much coarser size than the finest particle, then to separate out as much clean mineral and clean gangue as possible and to recrush the residue. This process can be repeated indefinitely, but in practice the added cost and the mechanical difficulties limit the number of repetitions to one or two. The sizes to be crushed to and the number of repetitions will vary for different ores and can only be determined by experiment. The things to be considered are the quantity and value of the products and the cost of crushing and separating. The ideal thing in crushing would be to have every grain of mineral remain intact and be entirely cleaned from all adhering particles of gangue. This is impossible to obtain in practice and there will always be some particles of mineral which have particles of gangue attached to them, or which are entirely surrounded by gangue. Such particles are known as attached or included grains and help to make up the "middling product" in the subsequent separation.

Rittinger has proved mathematically that the work of crushing is proportional to the reduction in diameter and that the work required is proportional to the number of planes of fracture, or in other words to the increase in surface of the particles. This theory has been recently investigated closely by Von Reytt at Przibram. The average Przibram ore consists mainly of quartz, calcite,

argentiferous galena and blende. The specific gravity of average ore is 3·13 and of calcite 2·75. The conclusions arrived at were that the ratio of work done to increase of surface is fairly constant with coarser sizes, but with finer sizes to increase of surface is much more rapid than the work required to produce it. For Przibram ore he advanced the conclusion that the increase of surface per horse power varies between 20 and 40 sq. m. and approaches an average of 25 sq. m. only under similar conditions of crushing when the quantities of feed are chosen to correspond, since the power used is dependent upon, but is not proportional to, the amount of feed. Incidentally the work brought out the following additional facts.' It should be borne in mind that in his tests the rolls were all doing free crushing. Slow running rolls produce more fines than fast running; the quantity of returns to be recrushed being the same, and the capacity per horse power less with the slow rolls. With spaced rolls, crushing moderately fine material, the quantity of returns is considerably higher than with close rolls, the amount of fines being the same and the capacity per horse power higher in the former case. Spaced rolls are to be recommended for coarse crushing only, and they counteract the tendency of slow running to produce fines. For economy, rolls should be run as nearly to full capacity as possible, since the capacity per gross horse power increases with the increased capacity. Strongly compressed rolls increase fines, decrease the returns and reduce the capacity per horse power. The Schranz mill produces less returns and more fines than rolls, but rolls would give just as many fines if the extra amount of returns was recrushed. The one runner mill gives more returns and less fines than the Schranz and has less capacity per horse power. Stamps give low returns, produce the most fines and have the least capacity per gross horse power. The most economical crushing machine is that which keeps the amount of returns as low as possible and at the same time makes the least fines, since it is the making of fines which consumes the greater part of the power. While Von Reytt was able to estimate, from the data obtained the amount of work necessary to produce a given weight of Przibram ore consisting of lumps all of the same size to any given size, his experimental figures show more favorable results than those obtained in practice at Przibram.

There are a few rough rules which are very often applied for estimating power which are well within limits for average ore. Among them are: the Gates rule that breakers need not over 1 h. p. to crush 24 tons per 24 hours to pass a 2·5 in. ring; a rule given by C. M. Ball that 1 h. p. will crush 3·84 tons of average ore per 24 hours to 0·0625 in.; a rule given by Hallett that 1 h. p. will crush 1 ton of quartz per 24 hours to 60 mesh. In addition to these rules one mill reports that 1 h. p. crushes 1·7 tons per 24 hours to 20 mesh; this figure includes also the power used for elevating and screening. At the Lake Superior steam stamp mills 1 h. p. crushes 1·6 to 1·8 tons per 24 hours from 3·5 or 4 in. down to 0·1875 in. All of these preceding figures agree more closely than at first appears, if one takes into account the amount of reduction in each case. Edison does much better than any of them. He is able to make 1 h. p. reduce 13·2 tons of mine ore to 0·06 in. or 8·2 tons to

0·02 in. His power measurements were made by the use of a motor and a wattmeter.

The writer has made 19 tests of various weights of pure quartz rock to get at the average pressure to be exerted in crushing by rolls. Incidentally some figures were obtained on the work required in crushing. These will be published with plotted results in his forthcoming work, *Ore Dressing,* with much other data on this subject, which is necessarily omitted here, owing to space limitations.

Comparisons of Various Machines.—The selection of the particular machine to be used will depend upon the character of the ore and the method of extraction adopted. Rolls are the standard machine for crushing all the brittle ores in preparation for concentration, except where very fine crushing is needed. An exception is the steam stamps used at or near Butte, Mont., though at all but the Anaconda works these are being replaced by rolls.

The large steam stamps are the standard crushers for ores containing native copper of Lake Superior. Gravity stamps hold their own for fine crushing. When acting on brittle minerals they tend to produce large quantities of slimes, owing to the fact that the particles cannot escape from the force of the blow until it is spent and the stamp lifted, and even then the stamp may fall again upon particles that are fine enough to be discharged. Rolls are to be preferred to stamps for fine crushing where the ore is to be roasted or leached and it is desirable to keep the percentage of slimes as low as possible. The Huntington, Bryan and ball mills are competing for the place occupied by gravity stamps for fine crushing. Grinding machines are employed to some extent for pulverizing the middling products from jigs, which contain portions of rich mineral that are most difficult to sever from the gangue. They are apt to make less slime than stamps and more than rolls.

SEPARATING, CONCENTRATING OR WASHING.

Having the ore crushed, the succeeding operation is that of separating the valuable minerals from the waste, or separating one valuable mineral from another. This operation, like crushing, can be divided into preliminary, final and auxiliary work. The preliminary machines are, as a rule, unable to deliver a finished product, but they divide the ore up into a set of intermediary products well suited for treatment by the final machinery. These, such as picking tables, jigs, vanners, slime tables and magnetic concentrators, separate the valuable minerals from the waste, but often yield middling products needing further treatment.

Preliminary Washers.—Preliminary washers include trough washers, log washers, wash trommels, washing pans and hydraulic giants. Their province generally is to disintegrate and float adhering clay or fine stuff from the coarser portion, which is done by using a considerable amount of running water, aided by some stirring device. They may be divided into three classes: (1) those using hand tools for stirring; (2) those using some form of rotating stirrers driven by machinery; (3) those using the force of a water jet.

Screens or Sieves.—These may be classified as follows: (1) Stationary

Screens: (*a*) Grizzlies or bar screens; (*b*) gravel screens of wire cloth; (*c*) perforated plate screens; (2) Moving Screens: (*d*) oscillating bar screens; (*e*) plain shaking screens or riddles; (*f*) revolving screens or trommels.

Grizzlies are made of stationary bars placed at a definite distance apart and set at such an angle that the ore will slide on the bars automatically. Where hand sorting is done, the slope is more gentle, but under ordinary circumstances they are given a pitch of 45°. The bars should have such a cross section that the spaces will widen from the top downward, thus insuring a free discharge of the undersize. Round bars are sometimes used. The wear upon grizzlies is so small that the cost per ton of ore is insignificant.

There does not seem to be much need of the grizzlies to relieve the breaker of the fines, for it is everywhere economy to have a breaker of greater capacity than the mill. A grizzly must be used, however, where the tendency to form slimes is to be avoided, and it saves wear on the jaws.

Plane stationary plates are sometimes used. At Edison's mill, there are two sets of this kind, both used for dry screening; the coarse set is about the same as a 14-mesh cloth screen. Each screen slopes 45° and has a net perforated surface 22 in. long and 16 in. wide. One inch of blank margin is left all around for attaching it to its frame. The screen is made of crucible steel 0·03 in. thick and has slots 0·5 in. long and 0·06 in. wide. The openings amount to 17·62%. The sand to be screened is fed to a tier of four screens, placed one above the other, and sloping opposite ways. The oversize of the first goes to the second, that of the second to the third, and so on. The final oversize has, therefore traversed 88 in. of screen. There are 60 tiers, making a total of 240 screens to turn out 300 tons of undersize per hour, about equivalent to 14-mesh wire cloth in size. It makes 200 tons of oversize per hour, which is recrushed and sent back.

In regard to the thickness of plates for this class of screening, he prefers thin plates, as the tendency to blind is much less where there are fewer chances for points of contact, as is the case in the thin plates. The screens of this plant have never shown the slightest indication of blinding. A test with a hand screen showed that 85% of the ore that was fine enough to pass through the mill screens did so. The writer understands that these screens have been adopted recently by the Sterling Iron & Zinc Co., to displace trommels.

Riddles are shaking screens with plain surfaces and are divisible into four groups: (*a*) Shaking machines which have an endwise or sidewise motion in the plane of the screen with or without a bump; (*b*) pulsating screens, which have an up and down motion perpendicular or nearly so to the plane of the screen; (*c*) gyrating screens with a circular or elliptical motion in the plane of the screen; (*d*) gyrating screens with motion in a vertical plane parallel to their length. In a mill visited by the writer the tailings from the No. 1 Gilpin County bumping tables go to flat screens, attached to, and bumping with, the No. 2 bumping tables at 120 bumps a minute. These screens slope 20° and are made of 50-mesh brass wire, the width of the holes being 0·015 in. (0·58 mm.). Their undersize is treated on the No. 2 tables; their oversize is waste. The attempt to use trommels for this work was a failure on account

of the chokage and excessive wear on the screen. The various forms of riddles, except in coal washing, are used to very slight extent in comparison with trommels. One hundred and sixty-seven positions in the mills visited by the author used trommels and three used riddles. In coal washing riddles are frequently used, as capacity is of more importance and the avoiding of slime less so than in ore screening, and because coal wears the screen less than ore.

Revolving screens or trommels may be divided into three classes: (*a*) Cylinders and prisms; (*b*) cones and pyramids; (*c*) spirals. They are designed to screen ore with but little fall and to avoid the vibrations caused by shaking screens. In the mills visited by the writer there are 143 cylinders, 14 cones, 1 cone and cylinder combined, 1 hexagonal prism, 1 hexagonal pyramid.

While the diameter of the cylinders varies from 20 to 48 in. the favorite diameter is 36 in. The number of revolutions per minute and other factors being the same, the larger the diameter of the trommel the wider and shallower will be the bank of ore, and consequently the better will be the screening and the less will be the tendency to blind up the holes, due to increased centrifugal force. Length varies from 34 to 168 in.; between 5 and 6 ft. are the favorite lengths. The trommel must be long enough to insure each particle a reasonable number of chances to pass through a hole when on its helical path, and this necessary length is limited by the allowable depth of bank. The greater the proportion of undersize that is nearly as large as the screen holes, the longer should the trommel be. The slope is a most important factor, as it largely affects both the capacity of the trommel and the quality of the product, and varies from 0·25 in. per ft. to a maximum of 5 in. per ft, the most common being from 0·75 to 1 in. per foot.

The revolutions range from 8 to 30 a minute, 16 to 20 being the most common. Although the capacity is increased by rapid revolutions, yet owing to the blinding of the holes by centrifugal force, it is not advisable to carry the speed beyond 20 revolutions per minute. A steady feed of ore to a trommel is essential to good work, for if it is overfed at times it will surely carry a larger proportion of the undersize than is allowable into the oversize. Wash water is generally fed on the outside of the up-coming side of the trommel by a spray pipe, although in instances the shaft is made hollow and holes bored in it to give a spray of water inside. A disadvantage in the use of water lies in the fact that wet or even moist ore wears out screens much more rapidly than dry ore; it also dilutes the pulp and so leads to loss in slimes. The capacity of a trommel depends upon the slope, the diameter, the speed of revolution and its length.

The system which meets by far the most favor in this country consists in the use of cylindrical trommels with only one size hole for each trommel, the set being arranged in a series beginning with the coarsest and ending with the finest size. This is the best system, because it proportions the wear to the ability of the screens to withstand wear—the coarsest has the hardest usage and the finest the mildest. It also makes a more perfect separation of the coarse from the fine ore and very much lessens the production of slimes. Abroad the series has sometimes been divided as follows: The ore first goes to

a screen with medium-sized holes; the oversize of this to a series of coarse screens; the undersize to a series of fine screens. This system saves some fall, but it causes excessive wear in the receiving screen, besides complicating the arrangement.

Laws of Screen Sizing.—The list of successive screen sizes used in any mill taken in order from coarsest to finest is called the sieve scale. Rittinger held that in such a set the diameter of the hole in any screen must bear some constant ratio to that of the one above it, thereby making the sieve scale a geometrical series. He adopted 1·414 ($=\sqrt{2}$) for this ratio, and his sieve scale starts with 1 mm. hole and ranges up and down from that point. Among the mills visited by the writer there is only one that has a constant ratio or geometrical series, although there are several that approximate this and there are two which have arithmetical series. There are many reasons why a single ratio running through the whole sieve scale may not be advisable. The scale may be divided, having an upper portion with one ratio, a lower with another. Practically the sieve scale is developed by the exigencies of the mill, the ratio decreased or increased between any two screens where the particular work seems to demand it. The ratio to be adopted depends mainly upon the specific gravity of the grains; in a general way the greater the difference in specific gravity between the values and the waste, the greater may be the ratio between the diameters of holes in successive screens. Other considerations, however, may modify the ratio, thus: (*a*) the difficulty in the subsequent separation increases with the difference in the size of the grains treated together, that is to say, with an increased ratio; (*b*) a product which consists mainly of cubes or compact forms can have a larger ratio than one which has a large per cent. of flat scales and elongated grains; (*c*) if the minerals are near each other in specific gravity close sizing will generally give cleaner products in the jigs following, but if the minerals are in a coarsely crystallized condition a larger ratio may be used than if the crystals are fine; (*d*) where a large quantity of material comes on to any screen, so that an increase will be required in the number of screens treating that size, or an increase of the subsequent concentrating machinery, it may be better to diminish the ratio, using two successive screens with different sizes of holes, rather than two screens, side by side, with the same size holes; (*e*) as a large number of screens tends to increase the loss in slimes and minerals, it may be better to use a large ratio; (*f*) in even so simple an operation as hand picking it is inadvisable to have too large a ratio, as the eye and mind cannot deal as well with 1 in. and 3 in. pieces together as with either taken separately; (*g*) if the ore is so friable and tender as to require carefully graded crushing, the upper part of the scale will need a smaller ratio than if such graded crushing were not necessary; (*h*) in case the tailings are so rich that it is necessary to recrush them, the ratio of sizes fed to the coarse jigs may be large, while on the other hand the tailings from the fine jig which are run to waste require closer attention and therefore a smaller ratio may be advisable.

The size of the hole used in the coarsest trommel will be determined by consideration of graded crushing and of hand picking. The size of the hole in

the finest trommel, down to which screening shall take place and beyond which the preliminary separation shall be made by hydraulic classifier, will be decided by three main considerations: (1) The hydraulic classifier can be run much more cheaply than the last one or two trommels; (2) on the other hand the tailings of the jigs treating classifier products are much richer than those of jigs treating sized products; (3) the finer the screening is carried (that is, the later the hydraulic classification begins) the denser will be the fine pulp sent to slime tables, because there will be fewer hydraulic classifiers, which are great diluters of the pulp. This is a distinct advantage for slime table work.

The first of these considerations is an argument against fine screening, but the other two favor it. Each mill manager must decide whether fine screening or coarse classification is better for his particular case. In this connection it is debatable whether the more common European limit of 1 to 1·5 mm. holes for the finest screen is not better than the more common American limit of 2 to 3 mm. The following figures show the coarsest and finest holes used in the American mills: Of 108 punched plate screens, the coarsest has 40 mm. holes, the finest 1·25 mm.; of four cast-iron screens, the coarsest has 54 mm. holes, the finest 25·4 mm.; of 43 wire cloth screens the coarsest steel screen has 25·4 mm. holes, the finest 0·5 mm., and the coarsest brass cloth screen has 6·4 mm. holes, the finest 0·25 mm. holes.

The practice almost universal in this country is to use round holes in punched plates, and approximately square holes in wire cloth screens; the round holes of the plate screens give the most even product. As cloth screens have a greater per cent. of opening, they have more capacity than punched plate. Holes in rows making 60° with each other ("staggered") give greater area of discharge than those making 90°. The limit of the opening is reached when the strength of the screen is too much reduced, consequently the thicker metal of coarse screens allows a larger percentage of opening than can be used in fine screens. The per cent. of opening for round holes with different arrangements and spaces is as follows: If the space equals half the diameter of the hole the per cent. of the opening is 40·3 with the 60° arrangement and 34·9 with the 90° arrangement; but if the space equals the diameter of the hole the per cents. of opening are respectively 22·6 and 19·6.

Classifiers.—Classifiers are devices for subjecting sands or slimes to the action of water under free settling conditions, either to obtain a series of products diminishing in size, preparatory to subsequent treatment, or to settle the whole material as completely as possible from the water.

They all have a carrying current, by which is meant a current of water which carries forward whatever grains may remain suspended in it. Some of them have also rising hydraulic, or clear water currents added from below. According to the purpose to be served, the design of the apparatus and the mode of employing the above currents, these appliances may be classed in nine groups, as follows:

(*a*) *Using Hydraulic Water.*—Trough or shallow pocket hydraulic classifiers, deep pocket hydraulic classifiers and tubular hydraulic classifiers.

(*b*) *Not Using Hydraulic Water.*—Surface current box classifiers, whole current box classifiers, distributing boxes, unwatering boxes, settling tanks and clarifying reservoirs.

These classifiers of group *a*, using hydraulic water, are of the greatest importance and subject the sands, while being moved forward by the carrying current, to a series of upward hydraulic currents, each of which acts in its own pocket. The effect produced is that grains of sand which are heavy enough to settle against the hydraulic current, may do so and can pass out at an orifice or spigot designed for that purpose, while the lighter ones are lifted and carried forward to the next hydraulic current. A series of these pockets, with their quantities of hydraulic water graduated so as to have less and less upward current, will yield products ranging from the coarse size of the first to the fine size of the last, and the finest grains will be in the overflow from the last pocket.

The tube or orifice up through which the hydraulic water passes will be called the sorting column and its size and shape are of great importance as affecting the quality of the products, as in this the true work of separation takes place.

The use of dial cocks for regulating the quantity of hydraulic water is to be recommended for restoring the conditions after flushing out the spigots, as well as for inspection to see that the mill boys are not deranging the machines. Constant hydraulic water is essential to good work. This may be obtained from a tank or reservoir with constant level. Constant feed water for the carrying current is likewise essential for good work. A strainer to remove fiber, chips and large, abnormal particles from the feed where fine pulp is treated is an important adjunct. This strainer may often be required for the hydraulic water. This fiber screen should be large enough to run some time without being cleaned, should be arranged for easy cleaning and should be cleaned at regular intervals.

Hydraulic classifiers of a great variety of forms are used for handling the products which are, in the judgment of the mill men, too fine to size economically with screens, and the products they yield form a decreasing series differing from those of the screens in that they are not truly sized but sorted products, obeying approximately the various laws of free settling in water. The spigot products are in almost every instance treated on jigs. The fine overflow of the last hydraulic treatment is usually sent to the box classifiers; then to vanners or tables, or occasionally to jigs; it is often sent directly to these machines.

The sizes fed to hydraulic classifiers in mills vary in maximum size from 5 mm. down to 0·635 mm. That in the final overflow ranges from about 0·50 mm. down to 0·25 mm. Linkenbach recommends 1·5 mm. as the best maximum grain to feed the hydraulic classifiers and 0·25 mm. as the best maximum size of grain in the overflow.

The following are the essential qualities of a good hydraulic classifier, which should be borne in mind in designing or studying this form of apparatus: (1) It should in mill phrase, "be able to stand up" and do good work with little

care; (2) the rising current in the sorting column should rise with uniform velocity over all parts of any given horizontal section; (3) it should require little mill height; (4) it should be capable of easy and positive adjustment; (5) it should not require an excess of water which may dilute the pulp too much, and (6) it should be capable of being fed and discharged continuously.

Laws of Classifying by Free Settling in Water.—In order to design hydraulic classifiers, box classifiers, tanks and jigs intelligently, the laws governing the rate of settling of particles in water must be understood, which with the various forms and practice with classifiers both of the type using hydraulic water and not using hydraulic water, will be discussed in full in the writer's work, *Ore Dressing,* now in press.

Hand Picking.—Hand picking is the process of separating into classes, by hand, ores that have already been broken. It saves rich ores from being made into slime; it saves the expense of dressing and in some cases of shipping waste rock, and at the same time it increases the capacity of the mill. It is especially applicable where the veins are narrow and much country rock is apt to be broken with the ore. By separating such minerals as blende and chalcopyrite, which are difficult to separate by water because they have so nearly the same specific gravity, hand picking relieves the concentrating machinery of its most difficult work.

Where labor is not too expensive, hand picking with sledging may save the expense of crushing and washing the richer ore; and where the ore is rich and concentration by machines is unavailable, hand picking is a necessity. Where the price of labor is high large crushers have replaced hand sledging, but the advantage of hand picking may still be retained.

Hand picking may be done in the mine, at the rock house or in the mill. Separation in the mine is not much favored in this country, probably owing to the loss of ore which occurs, owing to limited space and the difficulty of inspection. In several mines pure, soft, easily recognizable ores, such as galena, are selected to minimize the losses due to attrition. In certain European mines a small bonus is paid the workmen for selected ore.

The rock house, which may be an addition to the shaft house or a separate building, is usually supplied with grizzlies, picking floors or tables, and tracks for the removal of the waste material and the classified products. The material subjected to picking is the oversize to the grizzly. In certain gold mines 10 to 15% of rich smelting ore is picked from the chutes as the ore is loaded on cars; at two of the Lake Superior copper mines 8 and 30% of the total copper product was picked out in the form of native copper, while the quantity of waste rejected was respectively 12 and 10% of the rock hoisted. On the Rand, in South Africa, considerable attention is given to removing barren quartzite from the ore which passes through the rock houses. At some of the mines the ore is screened into three sizes, the fine going direct to the mill while the two coarser sizes are hand picked. The good ore remaining in the coarsest size is then crushed and sent to the grizzly, the oversize of which is again hand picked. In some cases 30 to 40 or even 50% of the rock hoisted is removed,

while it is estimated that the average for the district is probably from 12 to 16%. Hand picking in the mill is done on grizzlies, chutes, tables or floors.

The cost of hand picking depends upon the percentage that is picked out, the size of the ore, ease with which the different minerals can be distinguished, the mechanical facilities and the price of labor. The labor required is generally the cheapest about the mill. According to Hatch and Chalmers the average cost on the Rand is about 14c. per ton, while the cost of crushing in the mill was averaged at $1·10 per ton and the cost of cyaniding the mill tailings averaged $1·24 per ton at that time. Thus there was an average saving of about $2·20 per ton of waste picked out, besides the cost of transporting from the rock houses to the mills. While the native laborers employed in this work received only 50c. a day, the operation would have been economical had white labor been employed at $3 to $3·50 a day. As an illustration of the relative value of the waste, the annual report of the Geldenhuis Deep mine for 1897 states that the average assay of the hand picked waste (amounting to 9·7% of the ore mined) was $1·16 per ton, while the assay of the final tailings after passing through the stamp mill and cyanide works, was $1·08 per ton. At the Ferreira mine, the hand picked waste (amounting to 31·9% of the ore mined) assayed less than the final tailings from the cyanide plant.

There are five classes of picking tables in use: (1) stationary horizontal tables; (2) stationary sloping chutes; (3) shaking tables; (4) belt, rope or plate conveyors, and (5) revolving circular tables. The first two are fed automatically, but are discharged by hand; the remaining three are both discharged and fed automatically. Of the plants visited by the author two have stationary picking tables in the rock houses, four have stationary tables in the mills, seven have picking chutes in the mills, seven pick on fixed grizzlies, one picks on a vibrating grizzly and one picks on a rubber conveying belt. Of the 22 instances here noted, 14 picked on chutes or grizzlies, showing that it is more common to sacrifice height than to install the more complex conveying belts. The chief argument in favor of moving tables is that they furnish picking facilities without loss of height.

Jigs.—The work of hydraulic jigs depends, as a rule, upon the action of two currents of water, an upward and a downward, alternating with each other in quick succession, upon a bed of sand supported by a screen. Sands of two or more specific gravities, during the upward movement, called pulsion, arrange themselves according to the law of hindered settling. During the downward movement, called suction, small grains, wherever they are free to do so, move downward through the interstices between the large grains.

In continuous jigs there is generally also a surface carrying current which serves to transport the lighter grains forward until they are discharged over the tail; sometimes this is done by a mechanical device.

The machines to be described under this heading are of two classes; jigs with movable sieves, which obtain the currents by pushing the sieve up and down in the water, either by hand or by power; and jigs with fixed sieves, in which the currents of water are produced either by a plunger or by a stream

of hydraulic water brought from a hydrant into the hutch, that is, the space beneath the sieve, or by both. The fixed sieve jigs are by far the more common.

This hydraulic water acts to modify both pulsion and suction. Its increase adds to the former and diminishes the latter. It may be increased to an amount that will stop suction altogether; this, however, is more a theoretical idea than a practical method, on account of the large amount of water required to accomplish the result. If on the other hand, the hydraulic water is reduced to zero, then suction is increased to the maximum and suction equals pulsion.

The jigs have proved the most valuable concentrators yet devised for all the coarse products and they succeed also upon comparatively fine products, but have never been used to advantage for slimes. For this reason there is only one instance where they are used in gold stamp mills—where they are reported to be run successfully. They can be used in the separation of two, three or four minerals; for example, quartz, blende, pyrite and galena. The coarse jigs can be used to save clean lump ore and to send the lumps of included grains to the crushers to be recrushed. The fine jigs can be used to yield pure heads, middlings for recrushing and washing, and tailings clean enough to throw away. The feed to jigs may be sized products, sorted products, or natural products.

Jigs are, as a rule, final washers, that is to say, among the products which they turn out are one or two finished products, heads, or tailings, or both. Besides, they generally yield one unfinished product, which may be either the heads, the middlings or the tailings, according to the method of running.

The products of a jig are designated as follows: (1) Tailings, which form the top layer and are either skimmed off the top intermittently by hand, or are carried over the tail continuously by the carrying current. (2) Coarse concentrates, which form the heavy or lower layer upon the sieve, and are composed of grains too coarse to go through. These may be removed intermittently by skimming, or continuously by devices called discharges. (3) Hutch product; the "Hutch" or fine concentrates, is the part which goes through the sieve. It is discharged intermittently by shoveling or by a gate, or continuously by a running spigot, or by an elevator. A jig may be run to make any two or all three of the preceding products.

Movable sieve jigs are divided into movable sieve hand jigs; continuous movable sieve power jigs; intermittent movable sieve power jigs.

The simplest form of movable sieve jig is the hand jig, which costs about $20 and requires little repairs; but as this is used for testing purposes or where small deposits only are worked, it need not be described in detail here. The capacity of these jigs is given by Rittinger at from 3 to 4 cu. ft. of ore per hour for each sq. ft. of sieve surface. In practice the walls of the jig box may or may not project above the water during all parts of the stroke. When it projects above then its suction is equal to the pulsion, that is to say, just as much water will go down through the jig bed per stroke as rises up through it. When, however, the box is immersed, according to the amount of immersion suction will be more or less diminished, leaving pulsion as much as before and giving a much softer and more open whole bed and one which will complete

the separation into layers in much shorter time. This is because of the lift pump action of the jig which allows the water an easier passage up than down in the jig bed. In this case the water overflows the sides when the jig is immersed. The latter method would be preferable for closely sized material, the former for mixed sizes.

All continuous movable sieve power jigs have in common a jigging tank, a jigging screen and frame, and in most cases some special connecting joint between the tank and frame; some mechanism for giving the sieve its vertical oscillations; a feeder for bringing ore at a constant speed; and hydraulic water supply to the hutch. They also have devices for removing the tailings, the coarse concentrates and the hutch product, and for elevating the tailings water, in general returning it into the hutch of the machine. These devices enable them to have a high capacity and a low consumption of water. As generally run they have strong suction, owing to the fact that most of the water that passes up through the bed has to pass down again. Jigs of this type are used at Przibram. The Bradford eccentric jig, Conklin and Robinson jigs are American types. The Schranz rectangular jig, the Bilharz oblong and circular sieve jigs have some Continental use. The Handcock and the Hooper jigs are types in which the sieve has a horizontal as well as an up and down motion.

Intermittent, movable sieve power jigs differ from the continuous in that the feeder, the hydraulic water, the discharges of coarse concentrates, hutches and tailings, are all left out. They are practically hand jigs adapted to power.

In the fixed sieve jig machines the screen is stationary and the water is forced to rise and fall through it by the action of a piston or plunger which is generally placed in an adjacent compartment, connected with the hutch or space below the screen. These machines are almost always driven by power and are the forms in general use to-day. There are two classes of these jigs: (1) The Harz type, where the plunger receives its up and down motion from an eccentric revolving at uniform rate; (2) the accelerated jigs, in which some form of mechanism is adopted to give the plunger more rapid motion during pulsion than during suction.

The term Harz jig has been used loosely in the literature of the subject. The author has adopted the above definition, which is the one commonly accepted in the United States. This machine has found far more favor than any other jigging machine. It is used successfully for coarse and fine ores, for higher and lower specific gravity minerals. By it two, three and even four mineral separations can be made.

It consists of a jigging tank with vertical longitudinal partition, on one side of which is the screen upon which the ore rests; on the other side is the plunger for creating the currents. As the partition does not reach the bottom, there is free passage for the water from the plunger to the sieve compartment and return.

The jig may have one or more jigging compartments, each with its own sieve and plunger separated by cross partitions; two, three and four are the most common number of these compartments, although as many as seven have been used in southwest Missouri.

The Collom jig is an accelerated jig, formerly made of wood but now generally of cast iron. The advantages of this jig are: (1) that the speed being reduced during suction the blinding of the sieves and the formation of hard banks is prevented; (2) the ready adjustability of the throw; (3) the accessibility of the whole machine. These advantages are maintained, however, by a sacrifice of plunger area, and by a restriction of the opening between the plunger and sieve compartments.

Where a comparison was made between the Collom and Harz jig the following conclusions were arrived at: The Collom jig uses more water than the Harz and has less capacity. Four Harz jigs are thought to do the work of eight Collom jigs. The Collom jig also has higher running costs. At Lake Superior there is a tendency to design new jigs which will have the advantage of the Collom, but the more positive motion of the Harz jig.

The Evans jig differs from the Collom jig as the sieve receives symmetrical action of the plunger throughout its entire length. The Parsons and Fisher jig belongs to the Harz type. This is economical in space and handy to work. The disadvantages are: the troublesome wear of the piston and stuffing box, both of which have to work in gritty water, and the inaccessible piston.

The Bilharz circular plunger jig is a slime jig using direct eccentric motion. It has a circular jigging tank with an annular sieve 2·35 m. outside diameter, and a central plunger 0·880 m. in diameter, making 180 strokes per minute. The hutch is cylindrical outside while in its center is a truncated cone whose lower base is equal to the base of the cylinder. The ore is fed by a central distributor all round the inner circumference of the sieve, and the tailings discharged at the outer circumference. It treats 12 to 14 tons (dry weight) in 10 hours. The advantage of this form of jig is in its compactness. But a great disadvantage is that a series of sieves for making a series of products is impossible.

In under piston jigs the whole bed is moved only during pulsion. Rittinger's jigging pump or setzpumpe is an under-piston jig having a tight fitting piston with valves in it to let water up through and valves below to prevent a backward flow, making it a pulsion jig with no suction whatever.

The siphon separator used at Mechernich* has a remarkably high capacity at low cost, and on material free from included grains, does work equal to a pulsion jig, that is to say a jig which has no suction. It has not yet found its way into mills in the United States.

There are many other forms of jigs of more or less utility. The limits of space preclude their description in this article.

Method of Running Jigs.—The work of the jig is on one of three lines: (*a*) The jig makes coarse concentrates and tailings, with a small amount of hutch incidentally from attrition of the large grains; (*b*) the jig makes coarse concentrates, hutch and tailings; (*c*) the jig makes only hutch and tailings.

The following table shows that of 300 jigs there are 76 in Class A, 106 in Class B, 116 in Class C, 1 doubtful (either A or B), and 1 with first sieve C and second sieve B.

* *Engineering and Mining Journal*, Jan. 1, 1887.

In the Freiberg district in 1893, out of 126 jigs, 94 were in Class A.

The following summary shows the jigs divided into groups according to the material which is treated and each group subdivided into the classes:

	Class A.	Class B.	Class C.	
Jigs treating sized products from screens	76	12	26	A or B1
Jigs treating first spigot products from classifiers	0	33	21	B & C 1
Jigs treating later spigot products from classifiers	0	32	55	.....
Jigs treating natural or unsized products	0	7	5	.....
Jigs treating hutch products of preceding jigs	0	17	8	.....
Jigs treating tailings from preceding jigs	0	2	1	.....
Jigs treating both sized products from screens and hutch products from preceding jigs	0	3	0	.....

Class A.—This includes jigs which make coarse concentrates with incidental hutch.

Class B.—This includes jigs which make coarse concentrates, hutch and tailings. This method of jigging is used on more jigs than Class A.

Class C.—This includes jigs which make only hutch and tailings. This method is used upon all kinds of fine products and more jigs are run this way than by either of the two other methods.

Size of Sieve Hole.—The larger the hole relatively to the feed the more freely will the fine grains reach the hutch, and the less will the whole bed be clogged by their presence. In the jigs of Class A the ratio of the diameter of the hole in the sieve to the diameter of the grains in the feed, is less than one, ranging from 0·09 to 1. These jigs make their own bottom beds, and it is probably best to use a large ratio, as the coarse sieve will last longer and cost less. As the ratio for size of the hole to the minimum size of the feed Linkenbach recommends for this class of jigs 1 to 2 for the grains above 5 mm.; 1 to 1·5 for 3 mm. and 1 to 1·33 for 2 mm. grains. Below 2 mm. he recommends that the jig treating size products be put into Class B.

Jigs of Class B make their own bottom beds and the ratios for the maximum grains of the feed to the diameter of hole, range from 0·08 to 1·3 and for the minimum grains from 0·93 to 1·4 on sized products and to infinity on other products. For consistency, the ratio for the maximum grain must be below 1·0 and that of the minimum above 1·0 and the inconsistencies observed in these ratios are probably due to inaccuracy in designating screens. It would seem best to have the ratio for maximum grains nearly equal to 1·0, as this secures large interstices and as free passage as possible, in which the suction may act. This jig needs a thicker bottom bed than a Class A jig, because its interstices are so small.

The jigs of Class C have the ratio greater than 1·0 on all products except the later spigot products of a classifier and the tailings of jigs, in order to let the concentrates through the sieve. On these two, however, it may be less than 1·0, owing to the fact that the concentrates are all in the finer part of the feed. No ratios are given on the later spigot products from the classifiers, as the data on the size of grain is too uncertain. On other products the ratio ranges for the maximum grains from 0·6 to 2·3 and for the minimum grains from 1·6 up to 3·1 for sized products and up to infinity for other products. Any

inconsistency is probably due to irregularities of sieve holes. The ratio should never be small enough to allow the bottom bed material to pass through, probably never less than 0·29 ($\frac{1}{3 \cdot 5}$).

Jigs of Classes B and C in many mills use a larger hole in the first sieve than in the later ones, because the first sieve is called upon to make so much more hutch work than the others and, therefore, needs to work freely.

Material of Bottom Bed.—The remarks in general under this head apply not only to the bottom beds which are put upon jigs, but to the bottom beds which naturally form on them. The bottom bed should be as nearly as possible of the same specific gravity as that of the concentrates; in fact, the same mineral should be used. If much heavier it requires excessive power to lift it and in that case causes excessive boiling of the top layer, and if it is not lifted, the quicksand effect of the liquid bottom bed is lost and it simply acts under hindered settling conditions in many small tubes. These remarks do not apply when a few shot are mixed with the ore. If the bottom bed is much lighter than the feed it lets gangue into the hutch.

From the above statement of principles it would seem that it is suitable to supply the earlier sieves of a jig with a bottom bed of the purest mineral and the later sieves, which are used to remove the last and most elusive particles, with one of middlings composed of included grains of gangue and concentrates. This is quite general.

If the bottom bed is of low specific gravity, for example, middlings or even blende, it refuses to remain level; it forms much deeper at the tail than at the head and is in danger of working off over the tail. This difficulty is met by sloping the sieve up 1 in. in its length, so that it becomes approximately parallel to the surface of the blende bottom bed. In this way a bottom bed of even thickness is obtained and it may be made of less depth, or in some mills from two to three cross partitions are used to prevent the crawling of a pyrite bottom bed.

The bottom bed should not be of too soft mineral, as excessive abrasion will take place, causing loss and frequent renewal. The loss, however, is partly recovered in the hutch except where a mineral like chalcopyrite is used which is both soft and of low specific gravity, and its fine abraded particles are lost in the tailings. As an exception to the rule forbidding the use of a material of higher specific gravity for the bottom bed, the replacement of galena by iron shot or iron punchings has been successfully done at Ammeberg. Pyrite has likewise been used to replace chalcopyrite or blende on account of its superior hardness. Davies mentions the use of shot made of an alloy of iron and aluminum which may be made of desired specific gravity by mixing the metals in suitable proportions. Lead shot either alone or with coarse ore has been used in some mills for jigging pyrite or galena. In South Africa lead shot are used as bottom beds on jigs for separating diamonds. Magnetite may be used for a bottom bed in jigging iron ores, and feldspar in jigging coal.

Feldspar is favored as a bed material both on account of its weight and the fact that it breaks into elongated and flattish fragments which are considered favorable to jigging.

The thicker the bottom bed the less freely concentrates pass through; therefore, the cleaner the hutch work will be. With a thin bottom bed the opposite is true. If the bottom bed is too thick a portion of what should go to the hutch fails to do so and may be lost in the tailings. For an ore with a large percentage of concentrates we need a thin bottom bed and for the reverse a thick bottom bed. This might be construed as an argument for placing a thicker bottom bed on a later sieve, though an argument may be given for the exact reverse; namely, in order to guarantee that the tailings are completely robbed of values, it may be better to let a little quartz go into the last hutch with the line concentrates. If so, then the bottom bed on the last sieve will be made thinner. The mill man will have to decide between these two arguments, which to follow. If he is trying to keep quartz out of the concentrates, then the former will be the method; if to extract the last possible grain of value from the tailings, then the latter should be employed.

The size of grain on the bottom bed effects the quality of the hutch product more than almost any other factor. If coarse, the bottom bed will have large interstices and will discharge hutch freely; if fine, the reverse will be the case. According to the author's measurements, the particles of ore will freely move down in the interstices of the bottom bed when the diameter of the grains of the latter is 3·5 or more times that of the former. We can, therefore, regulate the freedom of discharge as readily by the size of the grains as by the thickness of the bottom bed, or both. These adjustments may be used at the same time. A thinner, finer bottom bed may have the same rate of discharging hutch as a thicker, coarser bottom bed, but the latter will have a thinner top layer, which is more advantageous when jigging very fine material.

When the bottom bed is extremely coarse, the interstices will be so large that it does not move during pulsion, and the interstices act like so many tubes, the specific gravity of the grains having no effect upon the operation. Under these conditions the jig simply acts by the law of hindered settling and gangue is prevented from entering the hutch only by the application of sufficient hydraulic water. When, however, the interstices are smaller and the bottom bed rises, becoming liquefied (quicksand) during pulsion, then all the gangue is pushed above it, and if the bottom bed is thick enough, or the suction not too great, no gangue can come into the hutch.

The removal of coarse concentrates may be effected either by skimming or by automatic discharges running continuously or intermittently. Skimming is not used nearly so much in American mills as automatic discharges, though it is the rule in the native copper mills in Lake Superior, where there are very few cases of automatic discharge.

Skimming has advantages over automatic discharge where there is a small proportion of concentrates, as it would be impossible under these conditions to regulate a continuous discharge. It may do better work than the discharge, because the attendant selects the quality of products desired; and again the sieve is cleaned and the bottom bed carefully readjusted at frequent intervals.

The disadvantages are: the amount of labor required if there is a large proportion of concentrates, the time lost, the derangement of the mill work, and

finally, the depth of the bottom bed is too variable for the best work, beginning at the smallest allowable thickness and increasing to the maximum.

Automatic discharges may be divided into two groups: gate and dam discharge, and other forms. Many forms of gate and dam discharges have been designed which may be divided according to their position: side discharges, tail discharges and center discharges.

The number of sieve compartments in a jig depends in general upon three things: (1) The purpose for which the jig is used; (2) the capacity of the jig; (3) the ease of separation. The practice of the United States for Harz jigs where two-mineral separation only is required favors one sieve for feed which has a maximum grain lying between 64 to 32 mm.; two sieves for grains 32 to 16 mm.; three sieves for grains 16 to 8 mm.; three, four or five sieves for grains 8 to 0 mm. Where three-mineral separation is required from 3 to 6 sieves on a jig are usually employed in this country. In Europe four compartments are usually used for this separation, though three sieves are successfully used at Ems. The length of sieve is necessarily limited, although the effect of length is to increase capacity within certain limits; the best practice is to use shorter lengths for fine than for coarse stuff. In this country, however, many mills have sieves of uniform length regardless of the size of the material.

The wider the sieve the greater its capacity, and it would therefore seem advisable to make all jigs as wide as possible, so long as no difficulties of construction in consequence are encountered. The summary shows that there is a regular decrease in width from coarse to fine. The cause of this is that the finer the grain the more necessary for the pulsion to be evenly distributed over the whole width of the sieve, a thing which offers less difficulty the narrower the sieve. The following table shows a summary of results of American practice in mills visited by the author:

SUMMARY OF LENGTHS AND WIDTHS OF JIG SIEVES IN THE MILLS, EXCEPT COLLOM JIGS.

Jigs in which the maximum size fed is known.

Maximum Grain of Feed lies between these diameters. Mm.	Number of Jigs Considered.	Length.			Width.		
		Maximum. Inches.	Minimum. Inches.	Average. Inches.	Maximum. Inches.	Minimum. Inches.	Average. Inches.
64–32	5	48	28·0	35·9	24·0	16·5	20·6
32–16	11	46	26·0	36·0	24·0	16·0	19·4
16– 8	38	40	24·0	31·8	24·0	15·0	18·3
8– 4	38	38·75	22·0	31·7	24·0	15·0	18·3
4– 2	35	38·75	22·0	32·0	24·0	14·0	15·2
2– 1	13	36·0	22·0	31·4	24·0	13·0	18·2
1– 0	4	32·50	22·0	29·1	16·0	15·5	15·8

Jigs in which the maximum size fed is unknown, being fed by later spigots of classifiers. The size given is that fed to the classifier.

Maximum Grain of Feed lies between these diameters. Mm.	Number of Jigs Considered.	Length.			Width.		
		Maximum. Inches.	Minimum. Inches.	Average. Inches.	Maximum. Inches.	Minimum. Inches.	Average. Inches.
16–8	3·0	34·5	34·5	34·5	22·5	22·5	22·5
8–4	4·0	36·0	31·0	32·3	24·0	15·0	17·3
4–2	36·0	38·75	22·5	32·7	24·0	12·0	17·8
2–1	12·0	36·0	22·0	30·3	24·0	14·0	17·4
1–0	2·0	30·0	27·0	28·5	21·5	16·0	18·8

Speaking generally, the capacity of a jig increases with the number of strokes, but the speed is limited by the time necessary to develop the full effect of the stroke. The best American and European practices agree in increasing the speed rate from the coarse jigs toward the fine but with moderate speed throughout. Quite a number of mills have adopted the uniform speed throughout, while several mills increase speed toward the fine jigs, but use high speed throughout. Two out of three of these mills use lead shot on the sieves.

As it is customary to decide upon the speed of a jig when the mill is designed it is important to decide upon the best rate of pulsation.

The following table is a summary of American practice in the mills visited by the author. It gives not only the average pulsations per minute but the ranges for different sizes. It does not include, however, the Collom jig. The averages appear to the author to be well suited for adoption in practice.

European practice as recommended by authorities or observed in mills, shows, in the majority of cases, rates below the average rates of American practice, especially on the finer jigs. The pulsions of the Collom jigs are slower than that of others as the mechanism requires longer to get through a cycle of action.

Jigs in which the maximum size fed is known.

The maximum Grain of Feed lies between these diameters Mm.	Number of Jigs Considered.	Number of Strokes per minute.		
		Lowest.	Highest.	Average.
64–32	6	95	175	129
32–16	12	100	175	131
16– 8	38	80	250	144
8– 4	30	115	268	176
4– 2	31	130	350	225
2– 1	14	135	400	250
1– 0	8	210	384	281

Jigs in which the maximum size fed is unknown, being fed by later spigots of classifiers. The size given is that fed to the classifier.

The maximum Grain of Feed lies between these diameters Mm.	Number of Jigs Considered.	Number of Strokes per minute.		
		Lowest.	Highest.	Average.
16–8	8	120	160	147
8–4	4	160	210	197
4–2	35	140	400	237
2–1	12	141	315	213
1–0	2	250	400	325

The length of stroke or the amount of throw is adjusted according to some arbitrary rule when the jigs are set up. The manager varies the throw from time to time until the jigs are doing their best work. It is not usual to make any further change after this condition has been once established, unless a radical change is made in the work. To help the mill men judge the amount of throw required by the different size of feed, computations have been made showing the relation of the throw and the size of feed as found in the mills visited by the author. The result of these computations is given in the following table:

Jigs in which the maximum size fed is known.

The maximum Grain of Feed lies between these diameters. Mm.	Number of Jigs Considered. (1st sieve only.)	Diameter of maximum Grain in Feed.			Length of Stroke on First Sieve.			Ratio of average stroke to average size of Grain.
		Highest. Mm.	Lowest. Mm.	Average. Mm.	Highest. Mm.	Lowest. Mm.	Average. Mm.	
64 to 32	5	54·0	38·1	41·67	101·6	38·1	67·94	1·63
32 to 16	11	25·4	18·0	21·56	89·1	25·4	49·95	2·32
16 to 8	33	16·0	8·3	11·75	69·8	12·7	36·48	3·10
8 to 4	26	8·0	4·4	5·81	43·4	9·5	23·47	4·05
4 to 2	26	4·0	2·1	3·03	41·3	1·59	14·34	4·73
2 to 1	13	2·0	1·22	1·71	19·1	3·97	12·27	7·18
1 to 0	3	0.91	0·64	0·73	6·35	4·76	4·76	6·52

Jigs in which the maximum size fed is unknown, being fed by later spigots of classifiers. The size given is that fed to the classifier.

The maximum Grain of Feed lies between these diameters. Mm.	Number of Jigs Considered, (1st sieve only.)	Diameter of maximum Grain in Feed.			Length of Stroke on First Sieve.			Ratio of average stroke to average size of Grain.
		Highest. Mm.	Lowest. Mm.	Average. Mm.	Highest. Mm.	Lowest. Mm.	Average. Mm.	
16 to 8	3	11·1	11·1	11·10	38·1	19·1	27·51	2·48
8 to 4	3	4·5	4·5	4·50	25·4	15·9	20·12	4·47
4 to 2	33	4·0	2·3	2·91	38·1	0·79	10·15	3·49
2 to 1	12	2·0	1·22	1·89	12·7	3·17	7·41	3·92
1 to 0	2	0·91	0·64	0·77	6·35	1·59	3·97	5·15

In the upper part of the table the last column shows that for the larger sizes the throw can only be a little more (1·6 times) than the diameter of the grain, but that for the fine sizes it has to be much larger (6·5 times). The intermediate sizes are graded from the smaller toward the larger. The lower part of this table shows the same thing, but the ratios are based on the maximum sizes fed to the classifier and not on those fed to the jigs. Foreign practice regarding the length of the stroke appears in every case to fall within the ranges given by the author for American mills. The averages, however, seem to be a little higher, especially on coarse jigs. In regard to the amount of throw used on the several sieve, of a multi-sieve jig, the majority of mills prefer less throw on the later sieves owing to the lighter bottom bed.

Quantity of Water Used.—Exact rules for this cannot be given because the quantity will depend upon the area of the sieve, the number of sieves, the quantity and quality of the ore fed, the number and length of the strokes and the height of the tail board. The variation of quantity of hydraulic water is more used for regulating the jigging work from hour to hour than any of the other frequently used adjustments, *viz.:* rates of concentrates discharged; thickness of discharge, and in some cases rate of speed. The final regulation must be according to the appearance and feeling of the whole bed. The following facts are noteworthy: The amount varies from 10,500 gal. per 24 hours on a three-sieve jig in one mill to 400,000 gal. in 10 hours on a 5-sieve jig of another. The average of all is about 65,000 gal. per jig per 24 hours. With the exception of the highest one the average of mills visited by the author was about 34,000 gal. in 24 hours. The amount per sq. ft. of sieve area varies from 0·528 gal. per minute in one mill to 22·22 gal. in another. The average of all mills is 3·635 gal. The hydraulic water used per ton of ore varies from 9·76 tons to 54·98 tons in various mills. The average of all is 19·85 tons. In

European practice the average of water used per ton of ore appears to be somewhat lower.

Capacity of Jigs.—It is impossible to give the exact capacity of the sieves for jigging. There are so many ways of increasing capacity, some of which are employed here and others there. There are so many grades of difficulty which arise in the problem, some very easy and others very difficult of solution; and there are so many grades of good and bad work; one of which is called standard in one place, another in a second place.

In this country it ranges from about 0·1 ton per hour to 12 tons per hour in the case of an exceptionally large jig, and the capacity per square foot of sieve ranges from 0·15 tons to 9·6 tons in 24 hours. The average, however, is between 0·50 and 2 tons.

Power Consumed in Jigging.—It may be said that the power required to drive a jig depends on the area of the sieve; the height of the tail board; the specific gravity of the jigging stuff; the length of stroke and the number of strokes per minute. The consumption varies between 0·5 and 2 h. p. as a rule. Fraser and Chalmers, in computing new work allow 1·5 h. p. on a 1-sieve jig, 2 h. p. for a 2-sieve jig, 2·5 for a 3-sieve jig, 3 h. p. for a 4-sieve jig. To this they add 15% for friction of shafting, slip of belting and other mechanical losses of power.

The average cost of a wooden jig with wooden frames is slightly above $200; its life is long with few repairs.

The labor consumed in running a jig depends upon the conditions under which it is operated. If it is fed and discharged automatically it does not need so much labor. In the case of an exceptionally large jig, one man per shift is required to attend to each jig, but with a normal sized jig, one man attends to as many as 23; but the common practice for fixed sieve jigs would seem to be about 12 to 14 jigs to the man. Linkenbach allows one man for three 4-sieve jigs, in Continental practice.

The laws of jigging will be thoroughly discussed in Chapter XV. of the author's work, *Ore Dressing*.

Vanners.—The essential characteristic of a vanner is that it does its work upon the upper surface of an endless belt, which is shaken rapidly either sidewise or endwise, and which has a continuous slow motion uphill. The agitation makes the ore bed so loose that minerals of higher specific gravity can settle to the lower layer, while those of lower specific gravity rise to the upper layer. The travel of the belt draws the heavy minerals to the head end, while the water washes the light minerals down to the tail.

It appears to be good practice to feed vanners with unclassified pulp which has passed through round holes 0·75 to 0·40 mm. (0·03 to 0·016 in.) in diameter in the screens of the gravity stamps or of such machines as the Huntington, Bryan or Griffin mills. In such cases, however, some of the finest slimes will be lost in the tailings; and if these are of enough value they may be removed for special treatment, either before going to the vanners or from the tailings. Slime table middlings are also a suitable feed for vanners. When 1 mm. or coarser pulp is treated, that portion of the quartz which is larger than 0·75

mm., with its associated rich mineral, may be taken out by a classifier and sent to a corrugated belt vanner or other machine, the overflow of the classifier going to an ordinary vanner. It is probable that with an ore containing a large amount of clay it may be better to classify, sending the spigot product to the vanner and the overflow, if of sufficient value, to canvas tables.

It is well known to mill men that the tailings carry away a portion of the finest grains; but the author believes that instead of there being simply a little accidental pyrite of different sizes in the tailings (as is often stated), there is a diameter below which a large percentage must go into the tailings. This raises the question as to whether the vanner does not involve a principle that deserves to be recognized quite as much as free settling or hindered settling. The term *agitation* would seem to be a proper name for this; and the ratio between the maximum size of quartz and the maximum size of pyrite, for example, that go off together in the tailings may be called the *agitation ratio.* While this is much larger than the hindered settling (jigging) ratio, the author believes it is not as large as is generally supposed. To defend this statement he has only to refer to the gate canvas plant which every 24 hours treats about 100 tons of tailings from the well run vanners of the Kennedy mill, Amador County, Cal., by means of a hydraulic classifier and canvas tables. The spigot product from the classifier is poor enough to be thrown away, and carries all the coarser sizes of quartz, with practically no pyrite. The overflow goes to canvas tables, followed by a finishing vanner, and yields a final product amounting to about 1,000 lb. of clean pyrite per day, while the Kennedy vanners yield about 2·75 tons of clean pyrite per day. That is to say, more than 15% of the pyrite fed to the Kennedy vanners is too fine to be saved with the coarser portion. This is not a criticism of the Kennedy mill, but simply shows a scientific principle. The vanners probably save all they can, and the canvas plant completes the work. As a preliminary determination of the agitation ratio the author made sizing tests of the above classifier spigot product and of the final canvas plant concentrates, showing the sizes of quartz and pyrite in the Kennedy vanner tailings. This showed that, disregarding a few large, flat grains, the maximum size of quartz was 0·427 mm., and of pyrite 0·04 mm., giving an agitation ratio of 10·7:1. This ratio will be much affected by the way the vanners are run, and will be a maximum only when the pulp bed is thin and all the adjustments are correctly made. The determination of this ratio in any mill is a good test as to whether or not the vanners are well run.

The slope of the vanner and the speed of the belt will generally be found to vary together in any mill; other conditions remaining the same, steep slope needs rapid travel, and conversely, gentle slope needs slow travel. If, however, the other conditions do not remain the same, the slope and travel may not vary together. The chief qualities of the pulp that call for variation of slope and travel are: specific gravity of the heads and of the tailings; relative quantities of heads and tailings; and the size of the grains. To be driven at its greatest capacity a machine will need rapid travel and steep slope, but when driven beyond its natural maximum some concentrates will be lost in the tailings for lack of time for proper treatment.

The proper quantity of water in the pulp varies inversely with the slope, and directly with the travel. Clayey talcose and calcareous ores all form glutinous pulp, and should therefore be more dilute than quartz ores.

The bed of pulp on the belt, when a vanner is running properly, will be graded in richness from the head to the tail end, the heads being nearly free from quartz, and the tailings nearly free from concentrates. The author is of the opinion that in ordinary practice the best thickness is from 0·15 to 0·20 in. over the little rollers, and he believes it should never be allowed to exceed 0·25 in. If the bed is too thick it will form a peculiar hard cake, which prevents the separation of the minerals. The thickness found on 43 vanners in 16 mills ranged from 0·10 to 0·43 in., with an average of 0·21 in.

There is a general feeling among California mill men that the side-shake vanner is better for coarse slimes, and the end-shake for fine slimes. On the side-shake machine the settled concentrates are considerably disturbed by the falling streams of pulp as they pass up under the pulp distributor, and the fine grains so disturbed are carried down the belt to settle again, and again try to run the gauntlet of the pulp distributor, though the coarse grains easily pass up. The motion on the end-shake machine is such that the concentrates are less disturbed in passing the pulp distributor; and on the Woodbury machine this advantage is increased by placing the feed holes in several single files parallel to the length of the belt, with quite wide spaces between the files, instead of in a single row across the belt with but little space between the holes. Another disadvantage of the side-shake machine is the formation of banks and gutters at the edges of the belt, in which the separation cannot take place as well as in the middle; but this may be largely overcome by running the vanner with a thin bed. The end-shake vanner avoids almost entirely the formation of side banks. Both practical experience and theoretical considerations point clearly to the conclusion that the end-shake principle is better than the side-shake for very fine pulp; and the author can find no evidence that the end-shake is not as good for coarse pulp also.

Bumping Tables.—Bumping tables use mechanical agitation to bring the heavy and light minerals into their respective layers on a surface; and to utilize the jerk or bump to convey the heavy grains to one side or end of the machine, while the current of surface water conveys the light grains to the other side or end. The side-bump tables, which have the bump at right angles to the flow of water, can make three or four products graded according to their specific gravities; but the end-bump tables, which bump in the opposite direction to the flow of water, can make two products only, the heavy and the light. The Rittinger continuous side-bump table was the first of the former class; and it is not only still retained in some mills, but with some radical improvements in construction and mechanism, and under new names, the principles involved bid fair to produce the most efficient fine concentrators that have yet appeared. The Wilfley table and others that have followed its lead are of this class. The principal improvements are: (1) Making the dimension in the direction of the bump greater than in the direction in which the water flows; (2) using riffles at right

angles to the flow of water, and tapering toward the head end of the table; and (3) substituting for the cam, spring and bumping post, a link motion which gives an accelerated motion to the table during the forward stroke, with a quick reversal and a retarded motion during the return stroke. The following results were obtained on Wilfley tables in the course of regular mill work by the Summit Mining and Smelting Co., on a quartz-pyrite ore, assaying 0·07 oz. gold and 11·2 oz. silver per ton. In 23 hours and 15 minutes 64,000 lb. (32 tons) of ore, which had been crushed by rolls to pass through an 8-mesh screen, were treated on a single table, yielding about 28 tons of tailings, which were reground to pass through a 30-mesh screen and were re-treated in the same time on two other tables. The total amount of concentrates was 12,552 lb. assaying 0·34 oz. gold, 53·7 oz. silver, 36% iron, 8% zinc, 3% lead, and 4·2% silica. About 69% of the concentrates were obtained from the first table. The final tailings carried a trace of gold and 0·8 oz. of silver per ton. Thus 95% of the gold and 94% of the silver in the ore were concentrated into a clean product weighing 20% of the original ore. At the Butte Reduction Works, at Butte, Mont., a test was made on 1,085 lb. of 3 mm. jig tailings, assaying 1·3% Cu, 1·2 oz. silver and 87·2% SiO_2, after grinding to pass through a 20-mesh screen. The concentrates weighed 80 lb., assaying 8% Cu, 8·4 oz. silver and 16·4% SiO_2; the tailings assayed 0·5% Cu, and 0·6 oz. silver. The concentrates contained 45% of the copper and 52% of the silver in the feed. Since the tables in this mill have been put in regular running order they have treated 30 to 40 tons each in 24 hours, and in a little over three months there were obtained, from the formerly waste product, 436 tons of concentrates containing 61,627 pounds of copper and 3,543 ounces of silver.

Film Sizing Tables (*Slime Tables*).—Film sizing tables use the relative transporting power of a film of water flowing on a quiet surface, which may be either smooth or rough, to act upon classified products. The smaller grains, of higher specific gravity, are moved down the slope slowly or not at all by the slow under current; while the larger grains, of lower specific gravity are moved rapidly down the slope by the quick upper current. The commonest type of these tables is the convex revolving slime table, which has the form of a very much flattened cone supported upon a revolving vertical shaft. For the saving of very fine concentrates, fixed rectangular tables covered with cotton duck have come into extensive use during the past few years. The excellent results obtained with them depend largely on the little pits in the duck surface, which act as riffles to catch and hold the minute particles of the concentrates.

Although the principle of film sizing appears at first to be very simple, it seems more and more intricate as it is further studied; and an intelligent understanding of the various considerations involved is necessary to its successful application. These considerations are: the size, shape and specific gravity of the minerals; the preliminary sorting; slope of the table; quantity of water and density of pulp; thickness and velocity of the water film; rate of feeding; quality of the table surface; and the speed at which the table moves. Here we may simply say that the author's experiments clearly indicate that coarse

pulp is best treated with much water on a gentle slope, and fine pulp with little water on a steep slope.

Amalgamation.—The properties of mercury which enable it to form alloys with gold and silver are made use of in milling for the extraction of those metals from their ores.

Amalgamation may be done by any of the following devices: (1) Stamp mill with or without inside plates; (2) Arrastra, Huntington or other crushing mills with or without inside plates; (3) amalgamating pan; (4) clean-up pan; (5) clean-up barrel; (6) outside plates; (7) mercury wells; (8) mechanical amalgamators which do not combine crushing with amalgamating.

While both gold and silver are caught by all these, the amalgamating pan is used where silver predominates in the ore and the others where gold. The first five combine crushing with amalgamation, while the last three are pure amalgamating devices.

The details of amalgamated plates have been given more or less fully by other authorities in previous volumes of THE MINERAL INDUSTRY, and hence will be only briefly considered here. The total plate area per battery varies from 9·75 sq. ft. in a cement gravel stamp mill treating coarse gold, to 102 sq. ft. in the Utica mill at Angel's Camp, Cal. The average of 26 mills is about 55 sq. ft.

Mercury wells are practically obsolete in this country, although there are a few instances of their use. They are still used to a considerable extent in Australia, almost invariably in connection with amalgamated plates.

Mechanical amalgamators are used but little in this country since the work of brightening and amalgamating the gold is well done by the stamp mill. A great many designs have been invented, but for the most part they have been discarded either from their complication, difficulty of cleaning, or some other defect. The Atwood amalgamator and the Eureka rubber are two of these devices which have found some favor in this country in the past. The Lazzlo amalgamator, which is used to a considerable extent in Transylvania, is a pan-shaped device containing a revolving muller.

Magnetic Separation.—Magnetic separation has been confined in practice almost entirely to the separation of iron ores from gangue or from other heavy minerals. Magnetite has been separated directly, but limonite, hematite and siderite have almost always been given a preliminary roasting to convert them into the magnetic oxide. The separation of blende from pyrite, which can be only imperfectly done in the wet way, is readily done magnetically after a preliminary roasting which converts the pyrite to magnetic sulphide or oxide.

Recent advances in the design of magnetic separators have led to their use directly on ores containing iron or manganese which are only weakly magnetic. In regard to the separation of minerals other than iron ores, although the author knows of no plants in operation, yet small scale tests have made a separation on many minerals which lie so near one another in specific gravity as to preclude a separation in the wet way.

In designing magnetic separators it is necessary to have a complete knowledge

of the principles and theory of electromagnets, a thing which has not always been fully appreciated.

Magnetic separators may be divided into three classes: (1) Those with the ore on revolving cylindrical rolls or drums within which are magnets. Examples are the Ball-Norton, the Sautter, the Siemens and Halske, the Heberli, the Payne, the Ferraris, the Wenstrom, and the Buchanan. (2) Those with ore on conveying belts or pans which either traverse magnets or are traversed by magnets. The Wetherill, Conkling, Chase, Lovett-Finney, Hoffman and Kessler magnetic separators belong to this class. (3) Those in which the ore falls in front of magnets. This class is represented by the Edison machine.

Pneumatic Concentration.—The conditions which point to the use of air for concentration are: (1) The removal of dust from natural or pulverized material by a blast of air; (2) the treatment of placer gold, or ores in general, in districts where there is a scarcity of water. There are three chief classes of apparatus for separating ores that depend upon air as a medium: (1) Those using a continuous blast of air which acts on the particles and yields graded products. (2) Those which project the ore particles through air by a force other than air blast, the heaviest grains going farthest, the other grains being graded according to their momentum. (3) Air jigs, or those which subject the ore in a bed to a series of upward intermittent pulsations of air, the lighter particles rising to the top layer, the heavier settling beneath in the same manner as in the hydraulic jig when run without suction.

As examples of the first class there are Edison's dusting chambers, designed to blow out the apatite from magnetite, and the Hochstedt apparatus used at the Rheinpreussen colliery at Homberg on the Rhine for blowing dust out of coal. The second class is represented by the Pape-Henneberg and the Clarkson-Stanfield centrifugal machines. Of the third class, or air jigs, there are two types which have been used to a greater or less extent—the Paddock jig and the Krom jig. The former has developed into the Hooper-Paddock jig, which is used for jigging graphite at Ticonderoga.

In addition to the foregoing there are methods of winnowing, dry panning, combined with hand picking and blowing with the mouth, which are used in dry placer gold fields.

In the book on *Ore Dressing* will appear outlines of more than a hundred mills of different types in the United States, varying from the simple log washer or hand jig plant to the complex silver-lead or copper sulphide mills of Montana and Idaho. A few of the leading types of these mills, together with an outline of the method of testing for process, were given in Vol. VI. of THE MINERAL INDUSTRY, and need not be repeated here.

THE MINING STOCK EXCHANGES IN 1899.

In the United States speculation has been centered in Boston, where the

FLUCTUATIONS OF MINING STOCKS AT BOSTON DURING 1899.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Copper:													
Adventure Con. (*b*)	$25	10·25	8·50	18·13	9·63	16·00	13·00	15·50	13·00	14·00	10·00	11·00	9·00
Allouez (*b*)	25	10·25	6·50	12·88	9·50	12·50	8·00	12·00	9·50	12·63	9·00	9·25	8·00
Anaconda (*a*)	25	.....	.....	.....	.....	47·25	46·25	69·50	60·13	67·75	54·13	55·50	49·63
Arcadian (*b*)	25	78·00	64·25	88·50	63·00	20·00	14·00	71·50	65·00	69·00	50·00	56·50	43·00
Arnold (*b*)	25	14·00	8·88	13·50	8·00	11·50	9·00	10·50	9·50	14·88	8·00	16·00	10·50
Ash Bed (*b*)	25	2·00	1·50	2·50	1·63	2·00	1·50	2·00	1·50	2·50	1·50	2·50	2·00
Atlantic (*b*)	25	45·00	33·00	44·50	33·00	39·00	29·50	44·00	33·25	36·00	30·00	33·00	28·00
Baltic (*b*)	25	35·50	30·00	34·50	27·00	31·50	25·50	36·75	30·00	32·00	27·00	28·50	27·00
Bingham (*h*)	10	.....	.....	.....	.....	.....	.....	16·25	14·50	15·00	10·25	12·50	10·75
Boston & Montana (*a*)	25	358·00	270·00	377·00	343·00	380·00	360·00	390·00	350·00	388·00	340·00	350·00	290·00
British Columbia, Ltd. (*i*)	5	.....	.....	.....	.....	.....	.....	15·00	10·25	14·75	12·50	11·88	11·00
Butte & Boston (*a*)	10	93·00	77·00	96·00	84·00	95·50	78·00	106·00	86·50	104·00	80·00	80·25	71·00
Calumet & Hecla (*b*)	25	755·00	645·00	840·00	750·00	775·00	705·00	860·00	778·00	850·00	790·00	805·00	770·00
Centennial (*b*)	25	35·50	30·75	54·50	34·50	59·00	39·50	49·00	42·00	42·00	34·00	37·00	30·00
Copper Range (*b*)	10	.....	.....	.....	.....	.....	.....	51·00	47·00	49·00	49·00	47·50	40·00
Franklin (*b*)	25	30·00	21·00	30·00	21·00	27·00	21·00	28·00	26·00	27·50	19·50	20·00	17·50
Humboldt (*b*)	25	3·00	2·50	3·25	2·25	4·00	2·00	3·00	1·50	3·00	1·50	3·00	2·00
Isle Royal (*b*)	25	58·50	39·50	66·00	52·00	73·50	61·00	75·00	65·00	70·00	48·00	47·50	40·00
Mass. Con. (*b*)	25	.....	.....	.....	.....	.....	.....	16·25	13·00	15·75	12·50	13·00	10·50
Mayflower (*b*)	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Michigan (*b*)	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	14·00	13·00
Mohawk (*b*)	25	26·50	21·00	34·00	27·00	32·50	24·00	.....	.....	35·50	26·00	28·25	25·50
Old Colony (*b*)	25	19·25	15·25	21·00	16·00	20·00	15·00	18·00	15·00	16·00	12·00	11·75	9·88
Old Dominion (*c*)	25	39·50	35·25	44·25	36·25	47·50	36·25	52·50	42·00	50·00	39·00	40·00	36·00
Osceola (*b*)	25	104·25	80·00	108·50	87·00	94·00	86·00	99·00	89·00	94·50	82·00	86·00	77·00
Parrot (*a*)	10	40·00	37·00	56·25	38·50	49·00	39·00	72·50	50·50	68·00	56·00	59·50	48·00
Quincy (*b*)	25	189·00	146·00	185·00	170·00	175·00	155·00	175·00	165·00	168·00	155·00	165·00	150·00
Rhode Island (*b*)	25	.....	....	15·25	9·25	15·00	11·50	15·00	12·75	13·00	9·00	9·50	5·50
Santa Fé (*k*)	10	.....	.....	22·00	17·95	19·00	13·50	18·50	15·88	16·50	12·00	13·00	11·00
Tamarack (*b*)	25	251·00	160·00	250·00	225·00	320·00	220·00	248·00	220·00	235·00	220·00	218·00	187·50
Tecumseh (*b*)	25	7·50	6·00	10·00	6·88	10·00	7·25	8·50	7·00	8·00	5·50	6·25	4·50
Tri-Mountain (*b*)	25	.....	.....	.....	.....	.....	.....	14·50	13·25	13·00	9·50	11·75	9·50
Union (*b*)	25	.....	.....	12·00	7·50	9·00	6·75	11·00	7·25	11·00	8·00	8·50	6·50
United States (*b*)	25	.....	.....	.....	.....	.....	.....	.....	.....	25·00	20·00	24·00	21·00
Utah Con. (*h*)	5	30·00	23·63	45·00	27·75	45·50	38·50	52·50	40·00	48·75	39·50	43·50	37·88
Victoria (*b*)	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	6·00	5·00
Washington (*b*)	25	3·25	1·00	4·00	2·00	3·00	2·75	3·00	2·50	2·50	2·00	2·75	1·75
White Knob (*j*)	100	.....	.....	.....	.....	21·00	16·00	22·50	17·50	22·50	19·00	14·00	8·00
Winona (*b*)	25	16·00	12·00	22·00	15·00	19·00	16·00	19·00	16·00	.....	.....	14·75	13·50
Wolverine (*b*)	25	50·00	36·50	50·00	41·00	49·00	41·00	49·50	44·75	47·50	43·00	47·00	42·00
Wyandotte (*b*)	25	.....	.....	.....	.....	.....	.....	.....	.....	9·75	6·25	7·00	6·00
Gold:													
Boston & C. C. (*d*)	2	·32	·25	·25	·24	.....	.....	·30	·25	·28	·20	·25	·20
Breece (*d*)	25	1·00	.....	2·38	1·13	2·00	1·88	.....	.....	2·25	1·88	.....	.....
Cochiti (*k*)	10	11·75	9·50	12·00	9·50	15·25	9·25	16·00	13·25	15·25	12·00	14·00	12·00
Gold Coin (Gilpin) (*g*)	5	·75	·50	·99	·35	.....	.....	1·00	.....	1·00	·75	.....	.....
Guanjua to Con. (*k*)	5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	37·00	36·00
Guanjuato, pref.	5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	4·00	.....
Melones (*e*)	10	.....	.....	.....	.....	.....	.....	.....	.....	4·00	.....	4·25	3·50
Merced (*e*)	15	9·00	6·00	12·00	8·00	10·00	9·00	10·25	8·00	16·00	6·00	.....	.....
North Amer. Dreg. (*a*)	10	28·50	23·00	41·00	28·50	39·50	34·00	40·00	35·00	39·50	33·75	39·00	34·00
Pioneer (*e*)	10	7·75	5·63	8·25	6·25	.....	.....	.....	.....	3·00	2·50	2·63	2·50
San Ysabel (*e*)	5	15·50	12·50	17·00	9·25	.....	.....	18·00	12·00	15·50	12·00	15·00	12·00
Victor (*d*)	5	4·50	.....	4·63	3·00	5·38	3·13	4·00	3·50	4·00	2·00	3·75	3·50
Zinc:													
Amer. Zinc Lead & Sm. (*g*)	25	.....	.....	.....	.....	.....	.....	.....	.....	43·50	37·00	44·00	37·00
C. Z. L. Mg. Sm. (*g*)	10	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Miscellaneous:													
Ætna (*e*)	5	.....	.....	.....	.....	4·00	.....	4·50	4·00	5·00	4·00	5·00	4·00
Bonanza (*d*)	10	1·98	1·25	2·43	1·25	2·63	.....	4·50	2·00	4·00	2·50	6·25	1·50
Boston (*e*)	10	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	7·00	.....
Catalpa (*d*)	10	·25	.....	·60	·30	·40	·30	·35	·20	·39	·30	·30	.....
Central Oil	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Crescent (*d*)	10	.....	.....	·50	·25	·40	·25	·25	·20	·25	·20	·25	·24
Dominion Coal (*f*)	100	35·13	33·00	44·75	33·00	56·50	35·00	63·00	53·00	58·00	47·00	58·00	50·00
Dominion Coal, pref.	100	116·00	114·50	121·00	115·00	122·00	120·00	121·00	118·00	120·00	117·00	120·00	115·00
Dunkin (*d*)	25	.....	.....	·45	·30	.....	.....	.....	.....	·35	.....	·20	.....
Federal Steel	100	.....	.....	59·50	46·50	63·63	49·50	74·75	58·25	70·75	52·50	68·75	53·00
Federal Steel, pref.	100	.....	.....	89·50	84·00	90·00	86·00	93·38	83·00	87·50	74·00	84·50	78·25
Mont. C. & C. (*a*)	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Napa (*e*)	7	8·50	8·25	9·00	8·50	9·00	8·50	.....	.....	9·75	9·13	9·25	.....
New Idria (*e*)	5	.....	.....	11·00	10·00	11·50	11·00	.....	.....	12·00	11·00	12·00	.....
United States Oil	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

shares of copper mining companies in Arizona, Michigan and Montana, zinc and lead companies in Missouri and Kansas, and of gold mining companies in

FLUCTUATIONS OF MINING STOCKS AT BOSTON DURING 1899.—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Copper:													
Adventure Con. (*b*)	10·00	9·38	10·00	9·00	9·00	7·00	7·13	5·25	7·90	5·50	6·50	5·00	$83,154
Allouez (*b*)	9·00	7·75	7·75	6·25	9·00	5·00	6·00	3·75	4·50	3·25	3·50	2·50	119,066
Anaconda (*a*)	57·50	54·50	58·50	55·00	55·38	51·88	50·13	41·25	48·13	43·50	45·00	39·00	15,948
Arcadian (*b*)	71·50	59·75	80·00	68·50	74·00	47·50	53·00	35·00	39·38	28·00	32·50	18·00	263,287
Arnold (*b*)	16·00	13·00	15·63	14·00	14·50	10·00	12·00	8·00	9·00	7·75	8·00	4·00	128,228
Ash Bed (*b*)	2·00	1·00	2·00	1·75	2·00	1·75	1·25	·98	9·09	.....	1·00	·75	21,205
Atlantic (*b*)	32·00	30·00	30·00	27·00	31·00	23·00	30·00	25·00	28.00	25·00	26·50	20·00	61,927
Baltic (*b*)	35·00	31·00	31·75	25·38	27·00	21·50	25·38	23·00	25·50	23·00	24·75	18·00	131,961
Bingham (*h*)	13·50	11·50	18·25	13·00	17·00	14·00	17·75	13·75	16·00	14·50	15·73	9·00	101,864
Boston & Montana (*a*)	373·00	348·00	363·00	350·00	353·00	345·00	365·00	321·00	326·00	300·00	322·00	250·00	68,537
British Columbia, Ltd. (*i*)	11·75	9·50	13·75	9·25	12·38	10·00	12·00	10·50	15·50	9·88	11·00	10·00	42,663
Butte & Boston (*a*)	88·00	76·00	79·75	72·00	70·00	60·00	81·00	65·00	74·00	55·00	74·50	38·00	144,439
Calumet & Hecla (*b*)	820·00	780·00	845·00	830·00	840·00	771·00	800·00	740·00	778·00	745·00	775·00	710·00	21,881
Centennial (*b*)	36·75	33·00	37·13	33·00	38·63	32·50	34·00	22·00	26·50	21·50	28·75	14·00	477,663
Copper Range (*b*)	46·00	43·00	44·00	42·25	42·00	35·00	37·00	34·00	41·50	35·00	35·00	22·38	37,971
Franklin (*b*)	21·25	18·00	21·25	19·50	21·00	17·00	19·00	14·50	19·00	14·50	18·25	11·00	155,349
Humboldt (*b*)	3·00	2·00	3·00	2·00	2·75	1·50	2·00	1·50	.....	.....	2·00	1·00	13,724
Isle Royal (*b*)	48·00	44·00	54·00	46·50	49·75	41·00	46·00	36·00	38·50	32·50	32·50	20·00	228,389
Mass. Con. (*b*)	14·38	12·00	13·00	12·00	13·00	9·00	12·50	5·25	11·00	9·00	10·75	9·00	25,894
Mayflower (*b*)	.....	.....	.....	.....	5·75	4·00	4·75	3·00	4·00	3·00	3·00	3·00	7,670
Michigan (*b*)	14·50	13·00	14·00	12·50	13·00	10·00	11·00	9·00	10·00	7·25	9·00	7·00	16,342
Mohawk (*b*)	27·00	26·00	26·75	20·00	25·25	21·00	22·50	12·25	22·00	17·50	20·50	14·00	167,900
Old Colony (*b*)	12·00	9·00	10·50	9·00	10·00	7·50	9·75	6·50	7·00	5·75	6·00	4·00	145,704
Old Dominion (*c*)	39·25	37·00	38·50	35·00	35·50	30·00	34·25	26·00	32·00	28·50	29·50	18·00	421,078
Osceola (*b*)	91·50	87·00	92·00	88·00	88·25	82·00	87·50	79·75	88·50	80·75	81·25	61·00	142,967
Parrot (*a*)	58·00	52·00	55·00	50·50	52·00	42·50	49·50	40·00	53·00	42·00	44·00	31·50	359,545
Quincy (*b*)	172·00	159·00	168·00	155·00	160·00	152·00	160·00	150·00	160·00	152·00	153·00	125·00	20,892
Rhode Island (*b*)	8·50	7·25	8·00	7·00	7·25	5·00	7·25	5·00	6·00	4·13	6·00	4·50	108,052
Santa Fé (*k*)	14·75	12·50	16·75	13·13	16·00	11·75	12·75	7·00	9·88	5·50	8·00	4·50	291,200
Tamarack (*b*)	227·00	215·00	230·00	222·00	228·00	205·00	227·00	210·00	225·00	210·00	216·00	175·00	19,864
Tecumseh (*b*)	7·00	5·00	6·00	5·00	6·00	5·00	5.50	4·00	5·50	4·50	4·50	3·50	47,087
Tri-Mountain (*b*)	11·50	10·00	11·50	9·75	11·75	8·75	10·00	7·00	10·50	8·75	9·38	6·00	33,113
Union (*b*)	9·00	8·00	8·00	7·50	7·13	5·50	6·00	3·63	4·00	3·25	5·00	2·50	48,473
United States (*b*)	24·00	21·00	41·25	23·00	41·25	30·25	43·00	29·50	45·50	29·88	33·00	24·00	440,201
Utah Con. (*h*)	55·00	42·00	45·90	43·00	43·75	35·00	41·00	34·00	39·00	35·00	37·00	21·00	404,594
Victoria (*b*)	6·00	4·00	7·50	3·00	6·50	4·75	5·50	4·00	5·00	4·50	4·50	3·00	26,952
Washington (*b*)	2·50	1·50	2.13	1·50	2·75	1·88	2·38	1·50	2·13	1·63	1·63	1·00	107,270
White Knob (*j*)	10·50	8·00	11·13	7·00	16·00	12·00	16·00	11·00	15·50	13·00	14·00	8·00	24,482
Winona (*b*)	14·00	12·75	12·75	11·25	11·50	7·50	10·00	6·00	9·00	4·50	8·00	4·00	105,435
Wolverine (*b*)	47·50	44·50	48·00	46·00	47·00	40·00	42·50	39·00	45·00	41·00	42·00	34·50	101,578
Wyandotte (*b*)	7·00	5·50	6·88	5·00	5·50	4·00	4·50	2.00	3·00	2·50	2·00	2·00	14,576
Gold:													
Boston & C. C. (*d*)	.....	.....	·40	·15	·35	·25	·53	·35	·40	·35	·25	·25	105,115
Breece (*d*)	2·00	.....	2·50	2·00	2·50	.....	2·50	.....	.....	.....	2·50	2·50	18,650
Cochiti (*k*)	15·00	12·00	20·00	13·00	21·38	17·00	18·25	15·50	17·50	15·75	16·00	10·00	319,082
Gold Coin (Gilpin) (*g*)	·50	.....	.....	.....	·50	·20	.....	.....	·50	·45	.....	.....	20,812
Guanju to Con. (*k*)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	100
Guanjuato, pref.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	100
Melones (*e*)	3·75	3·50	5·00	3·75	4·00	2·50	2·75	1·88	2·00	1·50	1·75	1·50	10,685
Merced (*e*)	12·00	.....	13·75	11·13	14·50	12·00	11·00	10·50	12·75	10·00	16·65	8·00	30,470
North Amer. Dreg. (*a*)	35·00	34·00	37·00	30·50	33·00	25·00	26·00	21·00	21·75	18·00	14·00	2·00	75,904
Pioneer (*e*)	2·75	2·00	3·25	2·00	3·50	2·75	3·00	2·50	2·50	2·00	2·88	1·75	103,819
San Ysabel (*e*)	13·25	12·50	14·50	12·75	16·00	12·00	13·00	11·00	11·00	8·63	10·00	6·00	177,257
Victor (*d*)	3·50	3·25	3·50	3·00	6·50	3·00	.....	.....	3·50	2·00	3·00	2·00	7,625
Zinc:													
Amer. Z. L. & Sm. (*g*)	49·75	43·50	67·00	57·00	.....	.....	50·00	43·00	41·00	34·00	35·00	13·00	41,642
C. Z. L. Mg. Sm. (*g*)	.....	.....	.....	.....	.....	.....	.....	.....	6·25	5·00	5·00	4·50	4,660
Miscellaneous:	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
Ætna (*e*)	4·50	.....	4·75	4·00	4·75	4·25	4·75	4·25	4·63	4·50	4·50	4·25	16,215
Bonanza (*d*)	.....	.....	3·13	1·50	2·25	1·00	4·50	1·00	1·88	1·00	1·25	1·00	458,553
Boston (*e*)	.....	.....	7·00	6·25	7·00	6·50	6·50	6·00	6·00	.....	6·00	6·30	2,725
Catalpa (*d*)	·30	.....	·45	·30	.....	.....	·40	·35	·38	·30	.....	.....	45,760
Central Oil	.....	.....	.....	.....	24·75	23·38	24·50	23·63	23·00	21·75	22·50	17·50	31,982
Crescent (*d*)	·20	.....	·25	·20	·25	·20	·23	·21	·25	·23	·20	·20	42,470
Dominion Coal (*f*)	55·50	50·25	56·50	51·75	55·00	50·00	51·00	41·00	53·50	45·00	53·00	41·00	463,741
Dominion Coal, pref.	118·00	116·00	118·00	117·00	118·00	116·00	117·00	116·00	119·00	117·00	118·00	118·30	1,917
Dunkin (*d*)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,250
Federal Steel	61·13	57·25	62·00	57·75	62·75	51·50	59·25	51·75	62·50	55·00	63·13	40·00	714,718
Federal Steel, pref.	82·13	80·00	83·00	80·50	83·00	76·00	.....	.....	82·00	78·00	82·00	70·50	106,915
Mont. C. & C. (*a*)	.....	.....	.....	.....	.....	.....	.....	.....	10·00	9·00	9·75	7·25	8,725
Napa (*e*)	10·00	9·25	10·50	9·75	11·00	10·50	11·00	10·88	11·25	11·00	10·75	10·50	6,605
New Idria (*e*)	13·00	.....	12·50	.....	.....	.....	12·75	12·00	13·00	12·75	10·00	10·00	6,590
United States Oil	.....	.....	.....	.....	47·00	45·00	47·00	42·00	46·50	40·50	43·00	24·00	87,615
Total sales	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8,102,457

(*a*) Montana. (*b*) Michigan. (*c*) Arizona. (*d*) Colorado. (*e*) California. (*f*) Nova Scotia. (*g*) Missouri. (*h*) Utah. (*i*) British Columbia. (*j*) North Carolin . (*k*) New Mexico.

California are dealt in extensively. In Colorado Springs and Denver the exchanges dealt in shares of various Colorado mining companies, while Salt Lake City has a market chiefly for local stocks which have been less in demand than in the previous year. In San Francisco, where Comstock shares are dealt in, business has waned.

In New York business has been more active in mining shares, and dealings in coal and iron companies have improved materially. In Philadelphia the shares of industrial companies which are engaged in the mining industry either as miners, smelters or consumers of the first mineral products, were moderately speculated in. During the year the shares of many new industrial trusts or combinations have been bought both speculatively and for investment, and transactions have been large. Among the new industrial companies promoted this year the Republic Iron and Steel Co., the National Steel Co., the Union Steel and Chain Co., the International Steam Pump Co., and the American Steel Hoop Co. may be mentioned.

THE BOSTON MINING STOCK MARKET IN 1899.

The year opened with a pronounced boom in the copper shares, and prices

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS, COLO., DURING 1899.

Name of Company.	Par Value	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alamo	$1·00	·06	·05	·07	·06	·08	·06	·08	·07	·08	·07	·09	·07
Anaconda	5·00	·57	·33	·62	·52	·58	·51	·55	·50	·55	·50	·53	·45
Argentum-Juniata	2·00	·40	·34	·40	·27	·35	·30	·37	·25	·37	·30	·30	·20
Cadillac	1·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cripple Creek Con	1·00	·12	·08	·12	·10	·10	·09	·11	·09	·12	·10	·10	·07
Currency	1·00	·06	·04	·06	·05	·06	·03	·05	.....	·08	·05	·08	·07
Dante	1·00	.....	.....	·14	·12	·14	·12	·14	·11	·13	·11	·12	·10
Elkton Con	1·00	·90	·70	1·18	1·00	1·08	·87	·95	·89	·98	·88	1·00	·88
El Paso	1·00	·20	·10	·22	·18	·33	·18	·22	·19	·32	·18	·31	·19
Favorite	1·00	·07	·04	·07	·05	·05	·04	·05	·04	·05	·04	·05	·04
Findley	1·00	·25	·16	·26	·19	·19	·14	·16	·14	·15	·13	·18	·10
Golden Fleece	1·00	·40	·30	·62	·35	·40	·30	·36	·35	·40	·30	·40	·35
Gold Hill	1·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Ingham Con	1·00	·12	·07	·12	·09	·10	·07	·08	·07	·11	·08	·09	·08
Isabella	1·00	1·58	·37	1·58	1·01	1·11	·77	1·00	·85	·96	·77	·90	·35
Jack Pot	1·00	·48	·39	·42	·38	·42	·35	·45	·37	·48	·40	·42	·39
Lexington	1·00	·07	·06	·08	·06	·07	·02	·10	·07	·16	·09	·12	·09
Magnet Rock	1·00	·04	·02	·05	·03	·04	·02	·04	·03	·05	·04	·05	·04
Matoa	1·00	·35	·29	·46	·39	·44	·35	·41	·39	·41	·31	·36	·24
Midway	1·00	.....	.....	.....	.....	.....	.....	.....	.....	·08	·07	·07	·05
Mobile	1·00	·04	·03	·05	·04	·05	·04	·06	·04	·08	·05	·07	·05
Mollie Gibson	5·00	·30	·24	·32	·26	·30	·25	·31	·25	·30	·27	·30	·24
Montreal	1·00	.....	.....	.....	.....	.....	.....	·08	·07	·09	·08	·07	·06
Moon-Anchor	1·00	1·35	1·21	1·34	1·12	1·16	·99	1·20	1·12	1·15	1·03	1·19	·90
Mt. Rosa	1·00	·27	·26	·35	·27	.....	.....	.....	.....	·19	·17	·27	·19
New Haven	1·00	·03	·02	·04	·03	·03	·02	·03	·02	·04	·02	·03	·03
Oriole	1·00	·03	·01	·03	·02	·03	·02	·03	·02	·05	·03	·07	·04
Pilgrim Con	1·00	·09	·04	·09	·06	·06	·04	·06	·05	·07	·06	·06	·05
Pinnacle	1·00	·18	·05	·12	·07	·12	·08	·17	·10	·18	·14	·19	·12
Portland	1·00	1·89	1·46	2·00	1·82	1·89	1·70	2·00	1·79	2·00	1·97	1·99	1·65
Prince Albert	1·00	·05	·02	·06	·04	·05	·04	·05	·04	05	·04	·04	·03
Princess	1·00	·04	·03	·05	·03	·04	.....	·05	·04	·05	·04	·05	·04
Pythias	1·00	.....	.....	.....	.....	.....	.....	·04	·03	·06	·04	·05	·03
Specimen	1·00	·09	·06	·14	·09	·11	·08	·09	·08	·10	·08	·09	·07
Tornado	1·00	.....	.....	.....	.....	.....	.....	.....	.....	·27	.....	·28	.....
Trachyte	1·00	·06	·04	·06	·04	·06	·05	·05	·04	·05	.....	·06	·05
Uncle Sam	1·00	.....	.....	.....	.....	·04	·03	·05	·03	.....	.....	·05	·04
Union	1·00	.....	.....	.....	.....	·23	·20	·25	·20	·23	·21	·29	·19
Vindicator Con	1·00	1·00	·85	·99	·97	·95	.....	·95	·94	·98	·96	·97	·85
Work	1·00	·24	·15	·24	·18	·21	·18	·20	·18	·29	·18	·27	·17

rose rapidly. This speculative spurt brought out many new companies heavily capitalized and representing, in some cases, only prospect holes and abandoned mines in the Lake Superior country. The public subscribed to an enormous amount for these shares, as well as those mines of merely prospective value in Utah, Wyoming, New Mexico, Colorado and Canada. The zinc-lead companies of Missouri and Kansas had then their turn. Some of these are now paying regular monthly dividends.

In the regular list the old mining stocks, notably the Calumet & Hecla, Boston & Montana, Quincy, Tamarack, Osceola, Wolverine and Anaconda, show good trading at higher prices than last year, and have been steady dividend payers.

Quicksilver stocks were in fair request, and shares in the Boston Consolidated of California, the Ætna, Napa, and New Idria mines have been well maintained.

The gold mining shares which are still absent from the dividend list, were dealt in to a limited extent. Fluctuations in price were slight.

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS, COLO., DURING 1899.—*Continued*

Name of Company.	July.		August.		September.		October.		November.		Decemb'r.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Alamo	·12	·09	·11	·09	·11	·10	·12	·10	·13	·11	·20	·12	$922,150
Anaconda	·54	·46	·54	·48	·52	·47	·49	·43	·61	·42	·46	·36	565,128
Argentum-Juniata	·29	·25	·34	·25	·31	·26	·27	·26	·30	·20	·20	·17	180,796
Cadillac	·05	·03	·03	.....	·03	.....	·04	·03	·03	·02	·03	·02	292,100
Cripple Creek Con	·10	·09	·14	·11	·19	·15	·20	·16	·20	·16	·18	·15	963,087
Currency	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	311,100
Dante	·13	·12	·18	·14	·30	·18	·22	·20	·23	·19	·20	·17	1,207,066
Elkton Con	1·04	·91	1·10	·98	1·28	1·10	1·20	1·08	1·30	1·16	1·20	1·13	1,070,798
El Paso	·41	·29	·50	·40	·50	·35	·43	·39	·40	·33	·35	·24	1,227,070
Favorite	·05	·04	·06	·05	·07	·05	·06	·05	·07	·05	·07	·06	1,703,350
Findley	·21	·16	·20	·18	·19	·17	·18	·15	·16	·14	·15	·13	1,460,908
Golden Fleece	·40	·23	·30	·25	·50	·25	·30	·26	·35	·26	·35	·25	18,809
Gold Hill	·03	....	·04	·03	·04	·03	·04	·03	·04	·03	·09	·02	481,500
Ingham Con	·09	·08	·10	·08	·11	·10	·12	·10	·15	·11	·15	·13	495,471
Isabella	·90	·78	·99	·84	1·00	·90	1·24	·92	1·50	1·20	1·44	1·34	5,418,087
Jack Pot	·46	·36	·69	·45	·74	·65	·67	·59	·65	·61	·64	·51	3,448,225
Lexington	·13	·11	·15	·12	·21	·15	·27	·21	·32	·25	·30	·25	2,388,090
Magnet Rock	·05	·04	·06	·05	·06	·05	·06	.....	·06	·05	·05	·04	1,850,816
Matoa	·34	·30	·35	·32	·39	·34	·37	·32	·34	·28	·30	·24	665,800
Midway	·07	·06	·08	·06	·08	·07	·08	·07	·08	·06	·07	·05	1,300,500
Mobile	·07	·05	·06	·05	·06	·05	·05	.....	·05	·04	·04	.....	2,711 500
Mollie Gibson	·30	·25	·31	·24	·31	·25	·30	·25	·28	·23	·28	·20	108,475
Montreal	·09	·08	·11	·08	·16	·09	·16	·11	·13	·11	·12	·07	199,900
Moon-Anchor	1·19	1·09	1·13	·95	1·08	·92	·99	·65	·80	·68	·69	·50	175,980
Mt. Rosa	·28	·24	·40	·25	·37	·31	·46	·39	·50	·40	·56	·42	211,300
New Haven	·03	....	·04	·03	·05	·04	·05	·04	·07	·04	·07	·06	1,102,250
Oriole	·06	....	·09	·06	·08	.....	·08	·06	·08	·07	·07	·05	2,216,621
Pilgrim Con	·06	....	·09	·06	·10	·08	·09	·08	·13	·09	·10	·07	417,500
Pinnacle	·23	·17	·31	·21	·38	·30	·39	·32	·37	·32	·32	·25	4,986,566
Portland	1·98	1·90	2·25	1·95	2·60	2.25	2·50	2·24	2·50	2·40	2·50	2·24	560,895
Prince Albert	·05	·04	·05	·04	·05	·04	·07	·05	·07	·06	·06	·05	2,291,971
Princess	·06	·05	·07	·05	·07	·06	·07	·06	·08	·06	·07	·06	650,700
Pythias	·05	·04	·07	·04	·07	·04	·08	·07	·09	·08	·08	·07	803,500
Specimen	·13	·09	·13	·11	·14	·12	·14	·13	·15	·12	·13	·10	1,024,900
Tornado	·28	·26	·45	·27	·49	·45	·50	·46	·54	·45	·50	·45	243,300
Trachyte	·07	·05	·09	·06	·10	·08	·10	·06	·09	·07	·08	·07	1,080,050
Uncle Sam	·05	....	·07	·05	·07	·06	·07	·05	·07	·06	·07	·05	505,500
Union	·26	·23	.....	.....	·26	·22	·29	·26	·28	·22	·26	·24	2,019,928
Vindicator Con	1·04	·90	1·50	1·00	1·50	1·40	1·95	1·43	1·51	1·40	1·75	1·50	73,400
Work	·28	·21	·30	·26	·36	·31	·34	·30	·34	·31	·31	·26	4,405,161
Total sa'es	....	....	.....	.....	.....	.....	.....	.....	.....	.....	....	.....	51,700,999

The Colorado Mining Stock Market in 1899.

At Colorado Springs there was a considerable increase in mining stock speculation, the total sales amounting to 238,319,221 shares, valued at $34,527,469, as against 65,576,029 shares, valued at $10,287,546 in 1898, showing an increase of 172,743,192 shares, valued at $24,239,923. The Colorado Springs Mining Stock Association has adopted a good policy in weeding out the "wild cat" business on its board, and it is therefore gratifying to record that the year's trading has been in shares having a basis of value behind them—either of working mines or at least good claims showing enough to warrant development. The Denver Mining Stock Exchange and the small exchange at Cripple Creek have

FLUCTUATIONS OF MINING STOCKS AT DENVER DURING 1899.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.
Acacia	$1	.....	.....	.....	.....	·12	·08	·11	·10	·11	·09	·14	·06
Alamo	1	.....	.....	.....	.....	·08	·06	·09	·07	·08	.....	·08	·07
Anaconda	5	·58	·55	·68	·52	·61	·50	·58	·48	·60	·50	·58	·45
Aola	1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Arcadian Con	1	·05	·04	·08	·05	·05	·04	.....	.....	.....	.....	.....	.....
Argentum Juniata	2	·33	·45	·38	·33	·27	·28	·29	·28	·29	·29	·35	·20
Bankers	1	·10	·07	·13	·09	·12	·08	·10	·08	·18	·08	·10	·08
Battle Mountain	1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	·35	·26
Ben Hur	1	·07	·04	·07	·04	·06	·04	·05	·04	·08	·04	·09	·06
Big Six	1	·16	·05	·20	·10	.....	.....	.....	.....	.....	.....	.....	.....
Buckhorn	1	.....	.....	.....	.....	·04	·03	·05	·02	·05	·03	·04	·03
Colorado C. & M	1	.....	.....	.....	.....	·06	·05	·07	·05	·12	·07	·11	·06
Cripple Creek Con	1	·13	·08	·13	·10	·11	·08	·12	·09	·12	·10	·10	·08
Della S	1	.....	.....	.....	.....	.....	.....	·03	·02	·04	·03	·04	·03
Des Moines	1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Elkton Con	1	·95	·67	1·17	·98	1·04	·90	·94	·90	·98	·88	·99	·80
El Paso	1	·20	·09	·20	·18	.....	.....	·21	·19	·34	·19	·31	·25
Enterprise	1	·12	·04	.....	.....	.....	.....	·07	·06	·10	·07	·09	·04
Fanny Rawlings	1	·42	·35	.....	.....	.....	.....	·50	·41	·48	·41	·45	·40
Favorite	1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	·06	·04
Findley	1	·25	·16	·27	·18	·19	·13	·16	·13	·16	·12	·15	·12
Franklin	1	.....	.....	.....	.....	·02	·01	·03	·02	·04	·03	·03	.....
Garfield Con	1	·16	·13	·16	·13	·14	·12	·14	·12	·15	·12	·14	·10
Gilpin & C. C	1	·41	·39	·45	·39	·45	·35	.....	.....	.....	.....	.....	.....
Gold Coin, Victor	1	.....	.....	.....	.....	3·00	1·60	2·50	1·90	2·00	1·95	2·00	1·75
Gold Eagle	1	·40	·28	·55	·33	.....	.....	·40	·32	·50	·34	·50	·30
Gold Fleece	1	·34	·27	·60	·25	·42	·39	·41	·30	·35	·33	·33	·32
Independence	1	·69	·50	·64	·55	·68	·51	·70	·58	·67	·62	·63	·45
Ironclad	1	·04	·03	·04	·03	·04	·03	·04	·03	·05	·03	·07	·03
Isabella	1	1·75	·38	1·59	1·08	1·04	·81	1·02	·85	·97	·77	·94	·65
Jack Pot	1	·49	·36	·52	·49	·41	·34	·45	·38	·45	·41	·40	·29
Jefferson	1	·14	·04	·10	·06	·06	·04	·07	·04	·07	·06	·08	·06
Keystone	1	·12	·06	·12	·08	·11	·08	·11	·08	·16	·10	·10	·09
Los Angeles	1	·10	.....	·14	·10	·12	·09	·13	·09	·13	·09	·10	·07
Magnet Rock	1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	·05	·04
Matoa	1	·32	·28	·51	·30	·44	·38	.....	.....	.....	.....	.....	.....
Mobile	1	.....	.....	.....	.....	·04	·02	·06	·04	·09	·05	.....	.....
Mollie Gibson	5	·33	·20	·31	·24	·30	·25	·35	·24	·31	·26	·30	·24
Moon-Anchor	1	.....	.....	1·30	1·11	1·08	·98	.....	.....	.....	.....	.....	.....
Mt. Beauty	1	.....	.....	.....	.....	·08	·05	·07	·01	·06	·05	·06	·05
New Haven	1	·02	.....	·04	·02	·03	·02	·03	·02	.....	.....	.....	.....
New Zealand	1	·40	·14	·31	·21	·30	·20	·25	·20	·40	·22	·35	·18
Omega	1	.....	.....	.....	.....	·04	·02	·08	·02	·08	·03	·02	·01
Ophir	1	·50	·15	·45	·24	·35	·20	·37	·25	·40	·31	·36	·20
Pharmacist	1	·06	·03	·08	·05	·06	·04	·07	·05	·07	·05	·07	·04
Pinnacle	1	.....	.....	.....	.....	·12	·11	·17	·11	·17	·14	·20	·10
Portland	1	1·90	1·42	2·04	1·80	1·88	1·69	2·01	1·85	2·04	1·96	1·98	1·70
Prince Albert	1	·06	·02	·05	·04	·06	·02	·06	·04	·05	·04	·04	·02
Reno	1	·04	·02	·04	·02	·03	·02	.....	.....	.....	.....	.....	.....
Sacramento	1	·07	·04	·08	·05	·06	·04	·08	·02	·04	·03	·03	·02
Specimen	1	·08	·05	·15	·08	·10	·08	·10	·08	·15	·05	·09	·05
Trachyte	1	.....	.....	.....	.....	·05	·04	·06	·04	·06	·05	·06	·05
Union Gold	1	·27	·18	·27	·22	·23	·19	·25	·19	·24	·21	·20	·18
Virginia Mountain	1	·05	·03	·05	·03	·05	·03	·04	·03	·05	·03	·03	·04
Work	1	·25	·16	·24	·20	·22	·18	·21	·18	·20	·18	·27	·17

done little of note speculatively speaking. The most active stock was Isabella, which advanced from 37c. in January to $1·50 in November, and is paying regular quarterly dividends. Portland, also a steady dividend payer, rose from $1·46 in January to $2·60 in September, and Elkton Consolidated from 70c. in January to $1·30 in November. Moon Anchor, on the other hand, fell from $1·35 in January to 68c. in November. Vindicator, which pays quarterly dividends, gained from 85c. in January to $1·51 in November. Similar advances were made by many of the lower-priced stocks. The largest trading was in gold stocks, while only a few of the silver issues received any attention, foremost among these being Argentum-Juniata and Mollie Gibson.

FLUCTUATIONS OF MINING STOCKS AT DENVER DURING 1899.—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	
Acacia	·14	·10	·19	·15	·24	·16	·47	·25	·53	·36	·40	·36	$228,500
Alamo	·12	·09	·10	·07	·11	·10	·11	·09	·13	·11	·14	·12	273,000
Anaconda	·54	·43	·55	·48	·54	·47	·50	·43	·61	·45	·47	·39	218,400
Aola	·03	·02	·04	·03	·07	·04	·05	·04	·07	·05	·07	·05	.....
Arcadian Con	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	46,500
Argentum-Juniata	·28	·25	·32	·26	·32	·25	·28	·25	·23	·19	·21	·20	32,550
Bankers	·10	·08	·13	·09	·20	·11	·20	·14	·17	·14	·15	·11	130,500
Battle Mountain	·34	·28	·39	·32	·39	·34	·43	·32	·42	·34	·40	·30	16,500
Ben Hur	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	76,000
Big Six	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	131,000
Buckhorn	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	39,500
Colorado C. & M.	·12	·08	·15	·12	·16	·12	·25	·15	·24	·18	·19	·16	219,000
Cripple Creek Con.	·12	·08	·15	·10	·20	·15	·21	·15	·20	·17	·18	·15	146,000
Della S	·04	·03	·03	.....	·03	.....	·06	·02	·05	.....	·02	.....	11,000
Des Moines	·04	·03	·06	·04	·06	·04	·06	·04	·06	·06	·07	·06	51,000
Elkton Con	1·03	·89	1·09	·90	1·30	1·09	1·20	1·08	1·26	1·14	1·19	1·16	17,000
El Paso	·40	·18	·40	.....	.....	.....	.....	.....	.....	.....	.....	.....	1,000
Enterprise	·06	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12,000
Fanny Rawlings	·43	·39	·45	·40	.....	.....	.....	.....	.....	.....	.....	.....	4,700
Favorite	·06	·05	·06	·04	·07	·05	·06	·05	·06	·05	·06	·05	31,000
Findley	·38	·17	·20	·18	·21	·14	·30	·15	·16	·14	·15	·13	37,100
Franklin	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12,000
Garfield Con	·15	·13	·17	·14	·17	·08	·21	·17	·20	·21	·20	·16	135,750
Gilpin & C. C.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24,800
Gold Coin, Victor	2·05	1·90	2·15	2·00	2·10	2·00	2·00	.....	3·00	2·10	3·00	2·15	5,800
Gold Eagle	·50	·13	.....	.....	·60	·43	·60	·48	·60	·48	·02	·54	11,950
Gold Fleece	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5,800
Independence	·63	·57	·61	·40	·56	·47	·50	·45	.50	·48	·51	·49	89,300
Ironclad	·07	·05	·07	·06	·08	·06	·07	·06	·08	·05	·06	·05	285,700
Isabella	·89	·75	1·02	·85	·98	·33	1·30	·92	1·50	1·20	1·49	1·33	510,750
Jack Pot	·50	·31	·64	·44	·74	·64	·68	·56	·67	·63	·61	·60	46,369
Jefferson	·09	·07	·11	·07	·14	·09	·11	·10	·11	·09	·10	·08	15,200
Keystone	·14	·12	·15	·12	·14	·12	·13	·04	·19	·12	·15	·13	82,500
Los Angeles	·10	·08	·11	·09	·10	·08	·11	·09	·10	·08	·09	·07	382,500
Magnet Rock	·05	·04	·06	·05	·07	·05	·06	.....	·06	·05	·05	·04	55,000
Matoa	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9,250
Mobile	·06	·04	·06	·05	·06	·05	.....	.....	·05	·04	·04	·08	68,284
Mollie Gibson	·30	·24	·32	·23	·30	·24	·30	·27	·29	·23	·28	·25	20,200
Moon-Anchor	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,900
Mt. Beauty	·05	.....	·10	·08	·10	·08	·12	·04	·16	·12	·12	·10	334,700
New Haven	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13,000
New Zealand	·32	·25	·46	·28	·70	.40	·60	·20	·53	·40	·54	·40	85,250
Omega	·03	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	26,000
Ophir	·40	·25	·50	·35	·60	·40	·65	·42	·75	·60	·50	.....	48,400
Pharmacist	·05	·04	·06	·04	·06	·04	·09	·07	·20	·08	·16	·15	84,000
Pinnacle	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	23,000
Portland	2·00	1·88	2·06	1·95	3·00	2·00	2·44	2·25	2·52	2·36	2·43	2·30	18,600
Prince Albert	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	185,000
Reno	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	131,000
Sacramento	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10,000
Specimen	·13	·08	·14	·11	·15	·12	·15	·02	·15	·12	·12	·10	45,500
Trachyte	·07	·05	·09	·06	·10	·08	·09	·08	·08	.....	.....	.....	63,000
Union Gold	·36	·27	·36	·33	·40	·34	·39	·30	·40	·35	·37	·34	186,700
Virginia Mountain	·05	·04	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	45,500
Work	·29	·21	·31	·26	·41	·31	·34	·31	·34	·30	·31	·28	149,300
Total sales	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,409,553

FLUCTUATIONS IN THE PRICES OF STOCKS AT NEW YORK DURING 1899.

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Acacia, g., Colo	$1												
Adams Con., Colo	10	.14	.08			.22	.20	.20	.15	.15	.12	.15	.05
Alamo, g., Colo	1	.09	.04	.14	.06	.08	.06	.10	.07	.15	.06	.08	.06
Alice, g., Mont	25	.85	.75	.90	.75	1.05	.65	2.10	.75	2.10	.95	.85	.75
Alliance, g., Colo	1			.11	.08	.10	.08	.15	.08	.15	.10	.15	.10
Amalgamated, c., Mont	100												
Anaconda, c., Mont	25					47.50	43.00	70.00	43.00	68.00	53.00	56.25	49.00
Anaconda, g., Colo	5	.55	.35	1.00	.50	.62	.50	.60	.48	.56	.48	.53	.45
Anchoria Leland, g., Colo	1					1.00	.93	.98	.88	1.00	.92	.98	.73
Andes, s., Nev	3	.40											
Argentum-Jun., g. s. l., Colo	1	.33	.30	.37½	.30	.29	.27	.36	.25	.34	.20		
Belcher, s. g., Nev	3	.15	.06	.30	.15	.32	.15	.50	.29	.35	.20	.40	.20
Best & Belcher, g. s., Nev	3	.45	.24	.32	.40	.60	.45	.71	.48	.66	.45	.70	.35
Breece, l., Colo	25	1.10	.80	2.15	1.00	1.80	1.50	1.80	1.30	2.10	1.25	1.90	1.15
British Col., c., B. C	10												
Brunswick, g., Cal	1	.15	.14	.35	.15			.31	.20	.28	.20	.25	.12
Caledonia, g., S. Dak		.60	.50	.60	.50								
Catalpa, s. l., Colo	10	.15	.08	.22	.15								
Challar, s. g., Nev	3	.35	.10	.44	.20	.47	.22	.40	.18	.30	.22	.34	.10
Chrysolite, s. l., Colo	50	.14	.12	.16	.12	.16	.12	.18	.11	.17	.10	.10	.08
Comstock Tunnel, s. g., Nev	100	.04	.03	.08	.05	.07	.05	.12	.04	.09	.05	.05	.04
Comstock Tunnel Bonds	100	.04		.07	.04	.08	.05	.17	.04	.06	.05	.06	.04
Comstock Tunnel Scrip	100	.04		.06	.04	.06	.05	.07	.04	.11	.06	.12	.04
Con. Cal. & Va., s. g., Nev	2½	1.50	1.10	3.15	1.30	2.10	1.50	2.40	1.55	2.60	.30	2.65	1.25
Con. Imperial, g. s., Nev	1	.01		.02				.04	.02	.10		.02	.01
Creede & Crip. C'k, g., Colo	1	.05	.02	.11	.05	.07	.05	.10	.06	.11	.07	.11	.06
Crescent, s. l., Colo	10	.14	.07	.21	.07	.20	.20	.25	.15	.18	.09	.22	.15
Cripple Creek Con., g., Colo	1	.11	.07	.14	.09	.12	.08	.12	.06	.12	.09	.11	.08
Crown Point, g. s., Nev	3	.15	.08	.30	.12	.25	.15	.30	.15	.35	.15	.35	.22
Damon, g., Colo	1												
Deadwood-Terra, g., S. Dak	25	.50	.40	.60	.45	.65	.40	.65	.50	.60	.45	.80	.25
Dunkin, s., Colo	25												
Elkton Con., g., Colo	1			1.25	1.05	1.05	.85	.96	.90	.94	.84	.99	.83
Enterprise, g., Colo	1	.35	.30	.33	.30	.35	.30	.31	.25	.35	.25	.30	.25
Eureka Con., s., Nev	2½	.35	.20	.40	.35								
Fanny Rawlins, g., Colo	1			.45	.42								
Father de Smet, g., Dak	100	.14	.10	.14	.10			.15	.10	.14	.10	.12	.10
Findley, g., Colo	1			.22	.21								
Garfield Con., g., Colo	1			.15	.12	.14	.12	.14	.11	.14	.12	.14	.10
Gold Coin, g., Colo	1			1.90	1.73	2.00	1.55	2.03	1.90	2.02	1.85	1.98	1.85
Gold Coin Gilp., g., Colo	1			1.00	.80	.75	.40					.75	.25
Golden Age, g., Colo	1												
Golden Fleece, g. s., Colo	1	.40	.25	.65	.30	.41	.30	.40	.35	.37	.25	.40	.27
Gould & Curry, g. s., Nev	3	.30	.20	.65	.12	.50	.33	.53	.25	.38	.26	.37	.18
Hale & Norcross, s. g., Nev	3	.20	.07	.44	.05	.40	.25	.38	.20	.38	.20	.35	.30
Homestake, g., S. Dak	100	60.00	50.00	61.00	50.00	55.00		60.00	50.00	60.00	55.00	74.25	60.00
Horn Silver, s. l., Utah	25	1.30	1.10	1.60	1.25	1.75	1.40	1.80	1.25	1.75	1.40	1.60	1.25
Iron Silver, s. l., Colo	20	.83	.60	.90	.75	.77	.55	.65	.55	.60	.50	.65	.49
Isabella, g., Colo	1	1.20	.87	1.65	1.15	1.20	.80	1.10	.84	.95	.75	.90	.70
Jack Pot, g., Colo	1												
Jefferson, g., Colo	1			.06	.06	.08	.07	.07	.06	.07	.08	.07	.05
Kingston & Pemb'ke, l., Ont	10	.25	.10	.23	.15	.25	.16	.25	.15	.35	.15	.45	.15
Lacrosse, g., Colo	10	.16	.08	.14	.09								
Leadville Con., g. s., Colo	10	.14	.07	.15	.09	.13	.11	.12	.10	.13	.09	.10	.08
Little Chief, s. l., Colo	50	.21	.15	.28	.17	.20	.17	.23	.01	.29	.19	.23	.16
Mexican, g. s., Nev	3	.44	.25	.80	.32	.90	.50	.90	.50	.64	.40	.85	.35
Mollie Gibson, s., Colo	5	.30	.18	.40	.25	.31	.27	.35	.24	.35	.24	.36	.22
Moulton, g., Mont	25	.22	.15	.25	.20								
Mt. Rose, g., Colo	1	.25	.15	.32	.24	.31	.25	.28	.24	.27	.21	.30	.19
Occidental Con., g. s., Nev	3	.50	.20	.30	.20			.30	.20	.30	.20	.30	.12
Ontario, s. l., Utah	100	5.50	5.00	8.50	5.50	7.50	5.00	10.00	5.25	10.00	7.00	9.00	7.50
Ophir, g. s., Nev	3	.58	.40	1.40	.40	1.15	.90	1.40	.50	1.25	.90	2.00	.85
Pharmacist, g., Colo	1	.06	.03	.10	.05	.10	.04	.08	.03	.09	.05	.10	.04
Phœnix Con., g., Ariz	1	.12	.08	.12	.06	.13	.06	.15	.07	.14	.09	.10	.07
Pinnacle, g., Colo	1					.13							
Plymouth, g., Cal	10	.10	.07	.10	.09	.12	.09	.10	.06	.11	.09	.10	.08
Portland, g., Colo	1	1.70	1.40	2.00	1.70	2.00	1.65	2.00	1.65	2.00	1.80	2.00	1.50
Potosi, g. s., Nev	3	.18	.10	.40	.20	.35	.20	.36	.18	.39	.20	.50	.30
Quicksilver, Cal	100	3.00	1.40	3.00	1.25	2.75	1.75	4.13	2.25	3.00	2.00	7.00	1.50
Quicksilver, Cal., pref	100	11.00	3.00	5.00	3.00	10.00	5.00	13.00	6.75	8.50	7.50	8.25	6.50
Savage, g. s., Nev	2½	.10	.05	.39	.18	.40	.21	.33	.20	.25	.20	.30	.15
Sierra Nevada, g. s., Nev	3	.80	.60	1.65	.85	1.50	1.00	1.30	.90	1.00	.70	.92	.55
Small Hopes, s. l., Colo	20	1.40	1.25	1.40	1.20	1.40	1.25	1.35	1.30	1.40	1.25	1.35	1.00
Specimen, g., Colo	1	.04	.02	.08	.04			.10	.05	.07	.06	.07	.05
Standard Con., g. s., Cal	10	2.05	1.95	3.50	2.10	2.50	2.30	2.60	2.25	2.75	2.40	2.75	2.00
Syndicate, g., Cal	1	.08		.14	.06	.10		.10		.10	.08	.08	.06
Tennessee, c., Tenn	25												
Union, g, Colo	1	.24	.17	.30	.22	.23	.18	.25	.18	.23	.20	.22	.18
Union Con., g. s., Nev	2½	.38	.25	.50	.20	.60	.45	.65	.40	.50	.35	.40	.30
Utah Con., s., Nev	1	.15	.05	.35	.15	.28	.20	.20	.10	.18	.10	.22	.10
Work, g., Colo	1	.21	.13	.34	.17	.23	.17	.41	.19	.30	.16	.22	.17
Yellow Jacket, g. s., Nev	3	.33	.07	.45	.13	.35	.18	.44	.27	.50	.30	.56	.30

FLUCTUATIONS IN THE PRICES OF STOCKS AT NEW YORK DURING 1899.—*Continued.*

Name and Location of Company.	July.		August.		Sept.		October.		November		December		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Acacia, g., Colo.	.....	.....	.....	.....	.....	.....	·38	·36	·45	·38	.....	.....	$3,400
Adams Con., Colo.	·08	·05	·15	·06	·10	·07	·15	·10	·18	·10	·15	·08	23,000
Alamo, g., Colo.	·10	·07	·13	·08	·11	·09	·12	·08	·15	·10	·15	·11	100,500
Alice, g., Mont.	1·20	·65	1·05	·55	·80	·60	·70	·60	·51	.....	·50	·33	42,538
Alliance, g., Colo.	·27	·12	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9,250
Amalgamated, c., Mont.	.....	.....	99·00	92·00	97·50	88·63	91·88	84·25	87·50	85·00	87·00	78.00	.....
Anaconda, c., Mont.	60·00	58·75	58·75	55·00	56·13	50·75	50·00	42·50	48·00	43·50	45·13	31·50	719,283
Anaconda, g., Colo.	·51	·37	·57	·35	·56	·45	·46	.40	·55	·42	·47	·35	72,100
Anchoria-Leland, g., Colo.	84	·70	·75	·70	·75	·70	·78	.68	·85	·78	·88	·70	19,700
Andes, s., Nev.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	500
Argentum-Jun., g. s.l., Colo.	·27	·23	·33	·19	·30	·25	·26	·20	·24	·18	·21	·15	14,100
Belcher, s. g., Nev.	·30	·20	·37	·22	·56	·22	·31	·20	·40	·20	·28	·15	6,015
Best & Belcher, g. s., Nev.	·60	·50	·70	·50	·61	·40	·48	·25	·50	·25	.55	·22	17,000
Breese, l., Colo.	2·10	1·30	2·00	1·70	1·90	1·45	1·70	1·50	2·50	1·50	2·50	1·60	32,140
British Col., c., B. C.	.....	.....	13·00	9·50	11·75	10·50	12·00	9·50	11·79	9·50	11·50	9·25	.....
Brunswick, g., Cal.	·18	·15	·23	·15	·28	·20	·35	·20	·30	·18	·25	·14	27,350
Caledonia, g., S. Dak.	.....	.....	·70	·50	·60	·50	.....	.....	.....	.....	.....	.....	.....
Catalpa, s. l., Colo.	·30	·20	·22	·20	·25	·20	·30	.20	·38	·15	·15	.....	100
Chollar, s. g., Nev.	·38	·20	·37	·20	.....	.....	·40	.22	.....	.....	·21	·10	5,700
Chrysolite, s. l., Colo.	.....	.....	.....	.....	.....	.....	·05	.03	.....	.....	.....	.....	4,200
Comstock Tunnel, s. g., Nev.	·07	·04	·05	·04	·12	·04	·04	.....	·05	·03	·04	·03	122,600
Comstock Tunnel Bonds.	·05	·04	·05	·04	·06	·04	.....	.....	·04	·03	·04	·03	28,500
Comstock Tunnel Scrip.	·07	·04	·06	·04	.....	.....	.....	.....	·04	.....	·07	·04	50
Con. Cal. & Va., s. g., Nev.	2·55	1·85	2·35	1·65	2·05	1·50	1·85	1·35	1·70	1·35	2·30	1·45	25,240
Con. Imperial, g. s., Nev.	·03	·01	·03	·02	·02	.....	.....	.....	.....	.....	.....	.....	17,850
Creede & Crip. C'k, g., Colo.	·09	·07	·10	·06	.....	.....	·11	·10	·18	·11	·11	·08	68,000
Crescent, s. l., Colo.	·20	·09	·15	.....	.....	.....	·20	·15	·18	·15	·16	·10	2,900
Cripple Cr'k Con., g., Colo.	·12	·09	·14	·09	·11	·09	·20	·15	·21	·10	·20	·14	197,100
Crown Point, g. s., Nev.	·25	·20	·28	·20	·30	·17	·15	·10	·22	·05	·25	·10	5,600
Damon, g., Colo	.....	.....	.....	.....	.....	.....	·27	.....	·28	·26	.....	.....	1,000
Deadwood-Terra, g., S. Dak.	·80	·60	·85	·60	·85	·55	·95	·65	·70	·60	·60	·50	2,300
Dunkin, s., Colo.	.....	.....	·07	.....	·10	·07	.....	.....	·10	·08	·10	·08	300
Elkton Con., g., Colo.	1·05	·90	1·10	·91	1·30	1·05	1·18	1·08	1·25	1·15	1·55	1·15	32,650
Enterprise, g., Colo.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Eureka Con., s., Nev.	·50	·30	·50	·30	·45	·17	·30	.30	·25	·20	·30	·20	600
Fanny Rawlings, g., Colo.	..	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1,000
Father de Smet, g., Dak.	·10	.....	·15	·08	·10	.....	·10	.....	·10	...	·10	·08	1,800
Findley, g., Colo.	.....	.....	.....	.....	.....	.....	.....	.....	·16	·13	·18	·12	6,500
Garfield Con., g., Colo.	·15	·12	·18	·13	·18	·15	·20	·16	·24	·20	·22	·16	97,200
Gold Coin, g., Colo.	.....	.....	.....	.....	2·30	2·05	2·23	2·10	2·10	.....	.....	.....	15,200
Gold Coin Gilp., g., Colo.	·35	·25	·10	.....	·25	.....	.....	.....	.....	.....	.....	.....	.....
Golden Age, g., Colo.	.....	.....	.....	.....	.....	.....	.....	.....	·07	·05	·07	·03	22,200
Golden Fleece, g. s., Colo.	·38	·22	·31	·22	·30	·24	·32	·23	·35	·23	·34	·22	64,900
Gould & Curry, g. s., Nev.	·40	·25	·55	·28	·30	·20	·35	·25	·25	·18	·20	·09	8,000
Hale & Norcross, s. g., Nev.	·40	·30	·33	·25	·31	·25	·35	25	·40	·28	·50	·30	5,487
Homestake, g., S. Dak.	75·00	60·00	78·00	60·00	75·50	65·00	73·00	65·00	76·00	65·00	75·00	62·00	975
Horn Silver, s. l., Utah.	1·50	1·20	1·45	1·25	1·35	1·10	1·35	1·20	1·20	1·10	1·10	1·00	4,775
Iron Silver, s. l., Colo.	·60	·50	·85	·48	·57	·50	·60	·50	·69	·52	·60	·50	12,700
Isabella, g., Colo.	·90	·75	1·00	·80	·96	·90	·96	·90	1·45	1·20	1·45	1·25	98,100
Jack Pot, g., Colo.	.....	.....	·60	·55	·72	·31	·64	·58	·63	·51	·62	·55	8,000
Jefferson, g., Colo.	·09	·04	·15	·07	·10	·08	·13	·08	·12	·11	.....	.....	81,900
Kingston & Pemb'ke, l., Ont.	·40	·20	·20	·15	·18	·15	·50	·18	·95	·65	·60	·35	13,530
Lacrosse, g., Colo.	·16	.....	·20	·14	·16	·14	·20	·14	·20	·08	·17	·15	2,100
Leadville Con., g. s., Colo.	·10	·08	·09	·06	·11	·09	·11	·08	·10	·08	·09	·06	20,000
Little Chief, s. l., Colo.	·22	·18	·21	·18	·20	·12	·23	·18	·18	·16	·15	·17	7,100
Mexican, g.s., Nev.	·70	·48	·70	·40	·60	·40	·37	·18	·60	·12	·60	·30	27,400
Mollie Gibson, s., Colo.	·29	·22	·28	·20	·30	·24	·31	·23	·30	·23	·25	·18	79,300
Moulton, g., Mont.	·35	·30	·35	·25	·35	·30	·35	·25	·35	·25	·35	·20	2,300
Mt. Rose, g., Colo.	·27	·22	·35	·22	·40	·30	·41	·35	·42	·37	·55	·28	76,900
Occidental Con., g. s., Nev.	·22	·18	·20	·14	·25	·14	·20	·15	·15	·10	·15	·10	1,810
Ontario, s. l., Utah.	9·50	6·00	9·00	6·75	8·25	6·25	8·50	6·75	8·63	8·00	9·50	7·50	3,700
Ophir, g. s., Nev.	1·10	·75	1·20	1·00	1·05	1·00	1·00	·70	1·30	·40	1·10	·35	10,610
Pharmacist, g., Colo.	·06	·03	·06	·04	·11	·03	·08	·06	·09	·07	·12	·06	103,100
Phœnix Con., g., Ariz.	·13	·07	·12	·08	·11	·08	·16	·08	·16	·10	·13	·09	205,800
Pinnacle, g., Colo.	.....	.....	.....	.....	.....	.....	·35	·30	·33	·30	·30	·24	10,700
Plymouth, g., Cal.	·12	·10	·10	·08	.....	.....	·10	·08	·20	·08	·20	·08	300
Portland, g., Colo.	2·00	1·80	2·15	1·80	2·50	2·10	2·35	2·20	2·75	2·30	2·50	2·20	22,150
Potosi, g. s., Nev.	·52	·40	·50	·25	1·00	·30	·61	·33	·36	·20	·35	·18	11,750
Quicksilver, Cal.	2·50	1·90	3·00	1·90	2·00	1·75	2·00	1·50	2·00	1·50	2·25	1·25	8,219
Quicksilver, Cal., pref.	11·75	7·50	10·00	7·50	8·00	7·25	7·50	7·00	8·50	7·00	9·50	7·00	3,482
Savage, g. s., Nev.	·32	·18	·35	·15	·30	·15	·35	·12	·17	·12	·17	·03	15,800
Sierra Nevada, g. s., Nev.	·68	·55	·75	·55	·90	·70	·75	·40	·55	·30	·65	·40	24,550
Small Hopes, s. l., Colo.	1·35	1·00	1·35	1·18	1·35	1·25	1·30	1·00	1·25	·40	1·09	·90	6,340
Specimen, g., Colo.	·07	·05	·11	·06	·11	·09	·11	·09	·12	·09	·10	·06	500
Standard Con., g. s., Cal.	2·60	2·10	3·00	2·40	3·20	2·70	3·25	2·75	2·90	2·50	2·80	2·25	18,560
Syndicate, g., Cal.	·08	·07	.....	.....	.....	.....	·03	.....	·08	.....	·03	·03	13,650
Tennessee, c., Tenn.	.....	.....	20·50	19·00	20·50	16·50	18·00	15·00	16·50	14·00	16·00	10·00	.....
Union, g., Colo.	·30	·24	·35	·30	·36	·22	·38	·24	·40	·34	·44	·28	41,960
Union Con., g. s., Nev.	·40	·24	·35	·18	·40	·20	·35	·28	·35	·20	·34	·22	7,850
Utah Con., s., Nev.	·35	·10	·14	·05	·14	·05	·14	·08	·09	·05	·08	·03	40,400
Work, g., Colo.	·26	·18	·30	·20	·35	·30	·34	·18	·34	·30	·33	·25	102,710
Yellow Jacket, g. s., Nev.	·43	·30	·40	·25	·30	·24	·30	·11	·25	·15	·20	·10	17,980
Total sales.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,950,554

c., copper; g., gold; i., iron; l., lead; q., quicksilver; s., silver.

The New York Mining Stock Market in 1899.

Considerable business has been done on the Consolidated Stock and Petroleum Exchange during the year, while on the New York Stock Exchange trading in the higher-priced stocks was moderate. The new Mining and Industrial Exchange that was formed some months ago has suspended business, owing to a lack of interest in the cheap stocks that it has been offering the public. There have been a number of flotations in this city, especially of zinc-lead and copper mining companies. Among the most important incorporations is the Amalgamated Copper Co., with a capitalization of $75,000,000, supported by people identified with the Standard Oil Co. Though the subscription price was $100, the stock advanced on the curb to $125 in May, but receded to $77 in December. In October a 2% dividend was declared, and quarterly dividends of like amount are promised. The Stratton's Independence, Ltd., of Cripple Creek, is another important flotation, and of the 1,100,000 (par £5) shares, the public were offered 100,000 shares at $12·50 in September by the Venture Corporation, Ltd., of London, which secured control of the mine. The stock was subscribed and dividends have since been paid. It is being regularly dealt in on the London Exchange. The Tennessee Copper Co., with property near Ducktown, Tenn., capitalized at $5,000,000, was incorporated in April by Messrs. Lewisohn, the

PRICES OF INDUSTRIAL AND COAL STOCKS AT NEW YORK AND PHILADELPHIA DURING 1899.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
American Alkali	$50	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
American Alkali, pref	50	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Amer. Smelting & Refining	100	.....	.....	.....	.....	44·50	42·50	57·00	43·50	50·50	32·00	42·00	33·75
Amer. Smelt. & Ref., pf	100	.....	.....	.....	.....	120·00	85·00	92·63	86·50	89·00	80·00	86·00	81·50
Amer. Smelting & W	100	54·00	38·00	64·75	45·00	66·25	59·63	71·50	61·00	71·50	59·25	64·00	52·00
Amer. Smelting & W., pf	100	123·00	108·00	102·25	92·50	105·50	100·00	102·50	92·50	97·75	87·25	97·00	95·00
Bethlehem Iron	50	.....	.....	.....	.....	155·00	82·00	107·00	108·25	105·50	101·00	64·00	59·50
Bethlehem Steel	50												
Cambria Iron	50	46·00	42·50	46·50	45·00	47·50	46·00	46·25	44·75	45·00	44·00	45·13	44·00
Cambria Steel	50	17·25	11·75	20·50	15·75	24·63	20·50	24·75	20·50	22·00	17·63	21·88	19·00
Central of New Jersey	100	106·25	97·00	112·00	98·50	119·25	105·00	124·75	117·00	120·75	110·00	118·50	104·00
Choctaw Transfer Certificates	100	30·50	29·13	36·00	29·63	37·25	33·50	37·00	35·75	36·00	33·50	41·00	34·75
Choctaw, pref	50	45·00	43·88	47·13	44·50	48·75	45·75	48·50	45·75	45·50	40·00	48·75	45·00
Colorado Coal & Iron Dev	100	1·75	1·00	1·00	·50	1·50	.....	1·00	.....	.....	.....	.....	.....
Colorado Fuel & Iron	100	37·00	32·50	35·00	31·13	37·75	32·50	54·75	35·00	53·25	39·50	46·75	41·63
Colo. H. C. & Iron	100	8·50	6·50	8·00	6·00	13·75	6·75	17·00	12·00	17·50	14·00	14·00	11·00
Delaware & Hudson	100	117·38	102·25	116·00	110·00	117·38	111·00	125·00	115·00	122·00	115·00	123·75	116·00
Federal Steel	100	55·63	51·50	53·75	46·38	67·50	49·38	75·00	60·00	70·50	54·13	66·38	57·75
Federal Steel, preferred	100	89·25	82·63	90·00	84·00	90·63	86·00	93·50	82·50	87·75	72·50	84·75	79·25
Lehigh Valley R.R.	50	27·88	24·25	26·75	24·50	30·00	25·25	29·00	26·50	25·75	23·50	28·13	23·25
National Lead	100	39·50	37·13	38·38	35·50	36·25	32·25	37·75	33·63	34·50	28·00	30·50	25·00
National Lead, pref	100	.....	.....	.....	.....	114·25	111·25	114·00	112·75	113·50	111·00	114·75	89·75
National Salt	100	.....	.....	.....	.....	40·00	34·25	68·00	37·00	55·00	49·00	50·00	41·00
National Salt, pref	100	.....	.....	.....	.....	78·00	70·00	80·00	69·00	77·00	69·00	80·00	70·00
New Central Coal	100	46·00	37·00	43·00	38·00	43·00	35·00	42·00	37·00	39·00	37·00	39·00	35·00
New York, Ont. & West	100	25·00	19·25	27·50	20·88	28·75	25·38	29·88	26·63	27·00	23·63	27·50	24·75
Pennsylvania R.R.	50	70·00	61·00	68·25	65·25	66·88	65·50	67·63	66·13	65·38	63·00	67·00	63·00
Pennsylvania Steel	50	35·00	21·90	40·50	26·00	37·50	39·00	90·00	58·00	85·50	74·00	96·00	74·88
Pennsylvania Steel, preferred	50	55·00	40·00	61·00	49·00	81·00	60·00	96·00	81·00	.....	.....	93·00	80·00
Philadelphia & Reading	100	25·75	22·00	23·50	20·50	25·00	21·25	24·88	22·50	22·88	19·50	21·75	19·75
Phila. & Reading 1st, pref	100	66·00	51·75	66·38	57·75	68·00	60·75	68·50	64·00	64·63	53·00	62·00	56·50
Phila. & Reading 2d, pref	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Republic I. & S.	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Republic I. & S., pref	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Standard Oil	100	478·00	446·00	490·00	449·00	484·00	470·00	490·50	470·00	496·00	458·00	470·00	455·00
Susquehanna Iron & Steel	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Tenn. C. I. & R.R.	100	47·75	36·00	46·50	38·50	54·75	43·38	68·00	54·00	66·38	54·00	65·00	58·63
United Gas Imp.	50	147·25	128·00	147·75	141·75	160·00	144·25	172·75	161·00	166·00	150·75	168·00	156·50
Welsbach, Can.	100	2·88	2·00	3·00	2·38	4·00	2·50	4·13	3·00	2·88	2·50	2·75	2·00
Welsbach, Com'l	100	10·00	.....	10·50	.....	14·50	10·00	19·00	14·00	13·00	.....	13·00	12·00
Welsbach, pref	100	62·50	61·50	60·00	.....	68·00	39·00	75·00	64·00	68·50	.....	51·25	.....
Welsbach Light	100	42·63	37·50	39·00	36·50	46·75	37·50	60·25	43·00	55·00	49·00	52·00	38·00

largest shareholders. This stock is being called on curb. The highest price paid was $30·625 in April, and the lowest $10 in December. Trading was moderate. The Union Copper Mining Co., with property near Salisbury, N. C., capitalized at $3,000,000, is controlled by a few people, and has sold on curb, at $36 to $38 in September, and down to $25 in November. Standard C. sold at $2 in February, and at $3·25 in September, and paid dividends of 10c. at intervals. The Syndicate Mining Co. reduced its capital stock from $10,000,000 to $100,000 at $1 par, and sold at 7 to 10c. per share. Quicksilver shares were influenced by the upward tendency of the metal market, and the common stock of the Quicksilver Mining Co. sold up to $3·50 and the preferred at $11.

Of the Colorado stocks the Cripple Creek section was most active. Portland, which pays regular monthly dividends, sold up to $2·45 in November, from $1·40 in January. Isabella has been lively owing to the discovery of new rich pockets in the property. The stock jumped from 38c. to $1·75 in January, and later receded to 75c. Dividends have been resumed. Elkton Consolidated, a regular dividend payer, sold at 80c. in February to $1·15 in December. Anchoria Leland brought 98c. in March and 70c. in September. Pharmacist

PRICES OF INDUSTRIAL AND COAL STOCKS AT NEW YORK AND PHILADELPHIA DURING 1899.

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
American Alkali	.....	.....	4·75	3·63	4·75	3·63	3·50	2·00	3·55	3·00	3·25	1·50	$121,059
Am. Alkali, pref	.....	.....	4·50	2·30	3·88	2·00	2·25	1·88	1·75	1·25	1·50	1·00	44,042
Amer. Smelting & Ref	40·00	35·50	45·38	39·50	43·38	36·25	39·25	34·88	40·75	35·25	40·00	30·00	328,954
Amer. Sm. & Ref., pf	87·00	82·00	92·00	85·00	92·00	86·25	88·38	85·38	91·75	86·00	92·00	83·00	228,683
Amer. Smelt. & W	59·50	53·38	62·50	54·75	58·50	40·75	54·63	47·25	50·00	46·00	50·00	32·00	1,943,645
Amer. Smelt. & W., pf	97·25	94·00	101·00	97·00	99·88	86·25	96·50	93·00	97·75	93·00	96·00	85·00	506,956
Bethlehem Iron	61·50	60·00	63·00	61·75	63·00	61·13	61·00	60·00	60·50	59·50	59·25	53·50	36,562
Bethlehem Steel	.....	.....	22·88	21·75	23·75	21·00	24·00	20·00	20·50	18·50	18·00	15·75	22,480
Cambria Iron	45·38	45·00	45·50	45·00	45·00	44·75	44·75	43·50	44·63	43·50	44·50	43·00	32,912
Cambria Steel	24·13	21·75	23·38	22·00	23·75	22·38	23·75	20·75	22·25	19·75	22·63	16·00	6,400
Central of N. J	119·00	115·63	119·88	116·50	121·25	116·50	125·88	117·50	126·00	120·25	123·00	113·00	487,748
Choctaw Transfer Ctfs	41·50	40·00	27·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	61,327
Choctaw, pref	49·00	43·38	48·63	47·00	49·00	47·75	48·00	46·00	45·75	44·00	46·00	41·50	157,914
Colo. Coal & Iron, Dev.	.....	.....	3·38	1·88	3·13	1·88	2·25	1·88	2·00	1·50	2·00	.....	24,153
Colo. Fuel & Iron	47·38	44·00	54·88	46·50	64·00	51·00	57·00	52·25	57·25	46·00	56·00	35·00	788,692
Colo. H. C. & Iron	15·50	13·00	.....	.....	20·50	17·00	19·00	17·50	20·50	18·25	18·00	13·00	10,901
Delaware & Hudson	124·75	121·50	125·50	121·75	125·50	121·88	124·50	119·00	123·13	119·25	121·00	111·00	360,492
Federal Steel	61·25	57·25	61·75	57·13	62·63	51·00	59·38	51·38	63·00	53·50	63·00	40·00	2,196,646
Federal Steel, pf	82·13	79·75	83·00	81·00	83·13	75·13	80·25	76·00	82·38	78·13	82·00	70·00	1,166,304
Lehigh Valley R.R	27·88	26·00	27·63	26·75	27·13	25·50	27·50	25·25	27·50	25·50	26·75	22·88	268,214
National Lead	31·25	28·50	33·38	30·25	31·50	28·50	31·25	28·00	31·25	28·00	30·00	24·00	177,682
National Lead, pref	113·50	111·50	114·50	111·25	111·75	110·00	114·25	108·50	109·00	106·38	107·00	103·00	15,707
National Salt	50·00	40·00	49·00	38·00	48·00	40·00	48·00	45·00	50·00	42·50	50·00	42·00	101
Nat'l Salt, pf	80·00	77·00	78·00	71·00	79·00	72·50	80·00	73·00	80·00	74·00	80·00	78·00	4,180
New Central Coal	40·00	37·00	39·00	37·00	.....	.....	.....	.....	.....	.....	.....	.....	2,188
New York, Ont. & West.	27·25	25·63	27·88	25·75	27·50	24·25	26·50	23·75	26·63	24·88	26·00	20·00	1,131,608
Pennsylvania R.R	68·88	67·13	68·63	67·63	68·00	65·00	66·75	65·00	68·63	64·50	68·88	64·00	129,990
Pennsylvania Steel	90·00	75·00	90·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	47,556
Pennsylvania Steel, pf	90·06	83·00	89·00	88·00	.....	.....	.....	.....	.....	.....	.....	.....	6,398
Phila. & Reading	22·00	20·25	22·00	21·25	.....	.....	.....	.....	.....	.....	.....	.....	405,677
Phila. & Reading 1st, pf	62·75	59·25	63·50	59·25	62·50	55·50	62·13	56·25	61·63	56·88	60·00	47·00	1,925,491
Phila. & Reading 2d, pf	.....	.....	35·00	32·38	34·25	31·00	33·75	30·50	33·50	29·00	32·00	24·00	165,136
Republic I. & S	.....	.....	29·00	22·00	33·88	25·25	28·00	23·63	26·75	21·88	26·00	17·00	298,326
Republic I. & S. pf	.....	.....	79·00	70·75	79·00	70·50	73·13	69·13	71·75	68·50	71·00	60·00	107,659
Standard Oil	467·00	453·00	474·00	455·00	480·00	461·00	465·00	460·00	475·00	454·50	470·00	450·00	.....
Susq. Iron. & Steel	.....	.....	.....	.....	7·75	6·75	7·13	5·50	5·50	4·38	5·00	4·25	88,485
Tenn. C. I. & R.R	71·25	64·50	100·00	71·88	126·00	96·00	119·75	109·00	119·00	110·00	117·00	64·00	1,812,394
United Gas Imp	171·00	164·63	170·50	165·00	170·25	162·00	165·75	157·00	163·00	157·00	162·00	144·00	296,971
Welsbach, Can	2·25	2·00	2·50	2·25	2·75	2·00	2·13	2·00	2·00	.....	1·88	.....	56,513
Welsbach, Coml.	10·88	9·25	18·25	10·00	11·50	10·00	10·00	.....	.....	.....	.....	.....	6,738
Welsbach, pref	68·00	58·00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	4,358
Welsbach Light	47·00	40·00	55·00	47·50	55·00	46·75	50·88	48·00	..·50	47·00	48·00	35·00	108,590
Total sales	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	15,580,734

and 1,850 levels. Its shares have dropped steadily, however, in the face of encouraging reports from the mine, and a large weekly output of high grade ore.

The record for 1899 has been the worst for many years. The total transactions on the San Francisco Stock Exchange for 1899 show aggregate sales of 6,370,152 shares. A feature outside of the mining share market is the promotion of many California oil companies, who are offering their shares to the public and in some cases are paying regular dividends.

Salt Lake Stock Market in 1899.

The year began in a period of strong advances. Mercur had the first awakening, and was bought in by New England and other outside holders. The Park City list leads, both in dividends and enlarged ore production. Silver King holds first place; the shares having advanced from $35 in January to $55 in December on regular payment of dividends. Daly-West increased its number of shares early in the year from 75,000 to 150,000 to include the Haggin interest and active mining began in the spring. The shares have ranged from

FLUCTUATIONS OF UTAH MINING SHARES IN 1899.

Companies.	Par Value.	January.		February.		March.		April.		May.		June.	
		7	21	4	18	4	18	1	15	6	20	3	17
Anchor	$10	1·50	·55	·60	.....	·50	·75	·60	·65	·75	.....	.....	·75
Ajax	10	1·15	1·15	1·25	1·32½	1·30	1·27½	1·48	1·58	1·52½	1·35	1·05	
Alliance	1	.....	·60	·40	·25	·25	·50	.....	·20	·10	·10	·10	·05
Alice	25	·75	·72½	·72½	·80	·85	·75	·55	·70	.....	·70	·80	·50
Bullion-Beck	10	5·25	4·25	5·00	4·75	4·75	4·50	4·50	5·25	5·90	5·85	5·90	5·80
Buckeye	1	·05½	·05½	·05	·04½	·04½	·03	·04½	·04	·03	·03½	·03½	·02
Centennial Eureka	50	36·00	32·00	32·50	38·50	38·00	44·25	50·00	49·00	52·00	50·00	40·00	47·00
Chloride Point	1	1·20	1·05	1·15	1·07½	·96	·98	·85	·98	·82½	·81½	·75	·83
Daly	20	·98	1·00	1·30	1·15	1·17½	1·21	1·20	1·20	1·45	1·30	1·35½	1·31
Daly-West	20	6·75	9·75	10·75	9·75	9·17½	10·12½	10·32½	9·95	9·92½	9·90	9·90	10·50
Dalton & Lark	1	·04½	·07½	·14	·10½	·07½	·09	·10	·11	·09½	·09½	·09½	·09
Dalton	5	·02½	·03½	·02½	·03	·02½	·01½	·02	·02½	·03½	·02½	·03½	·02
Dexter	5	3·22½	3·20	3·22½	2·85	2·35	2·45	.....	2·25	2·53½	2·40	2·25	2·03
Daisy	3	·62½	·57	·50	·50	·49	·41½	·45½	·28½	·30½	·27½	·21	·18
Eagle	1	·06½	·23	·15½	·10	·14½	·06	·07½	·06½	·07	·08½	·05	·06
Eagle & Blue Bell	1	2·05	2·00	1·85	2·00	1·65	1·75	1·81	1·60	1·55	1·37	1·37½	1·37
Four Aces	1	·59½	·62½	·64	·60½	·33½	·38½	·38½	·40½	·31½	·29½	·21	·22½
Geyser-Marion	5	·80½	·91½	·86	·89	·85	·79½	·77½	·75	·73	·76	·62½	·56
Galena	10	·56	·52½	·54	·50½	·44½	·40	·40	·41	·40	·28	·28	·26½
Grand Central	1	8·00	8·00	6·65	6·60	6·85	8·00	8·30	7·90	7·97½	7·82½	8·00	8·12½
Golden Eagle	1	·08½	·08½	·08½	·08½	·07	·06½	·06½	·06	·06½	·06½	·05½	.....
Horn Silver	25	1·05	1·15	1·40	1·40	1·50	1·40	1·45	1·30	1·50	1·25	1·50	1·50
Homestake	1	·10½	·0½	·10	·09½	·10	·10	.....	·11½	·18½	·14½	·10½	·08
Ingot	50	·50	·57½	.....	.....	.....	.....	.....	·34	·28	·26	·25	·20
Joe Bowers	1	·19½	·23	·18	·19½	·14½	·15½	·18½	·17	·15½	·15	·12½	·12
Joe Bowers Ex	25	.....	·13½	·13½	·09	·07½	·07½	·09½	·08½	·11½	·09½	·08½	·07½
La Reine	1	.....	.....	.....	.....	.....	.....	·67½	·70	·75	·80	·86	·88
Lower Mammoth	1	·70	·79	1·10	·92½	·77½	·68½	·78	·79½	·75½	·68½	·46½	·46
Little Pittsburg	5	·19½	·11½	·11	·13	·09½	·10	·07½	·11	·10	·07½	·08½	·09½
Mammoth	25	1·97½	1·92½	2·10	2·07½	1·82½	1·84	1·72	1·69	1·88	1·87½	2·00	1·96
Mercur	25	7·80	7·45	6·75	6·27½	6·35	6·60	6·98½	7·05	7·15	7·20	7·40	7·45
May Day	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	·65	·60
Northern Light	5	·50	·99	·91	·78	·58½	·54	·58	·69½	·77	·74½	·65	·60½
North Swansea	2	.....	.....	.....	.....	.....	.....	.....	.....	.....	·18½	.....	·11½
Omaha	2	·48	·44	·41	·40	·36½	·42½	·44	·38	·30	·25	·40	·26
Ontario	100	6·25	6·55	7·90	7·25	7·00	6·80	6·50	7·20	8·50	7·50	6·50	7.00
Petro	5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	·71	·70
Richmond-Anaconda	1	·05½	·07½	·07½	·07	·10	·7½	·10½	·10½	·10½	·06	·07	·04
Sunshine	10	·45	·62	·62	·58½	·40	·55½	·87½	·79	·94½	·70	·60	·53
Swansea	5	3·05	3·20	3·50	3·45	3·50	3·80	3·75	3·90	4·00	4·05	3·85	3·85
South Swansea	1	1·17½	1·20	1·20	1·20	1·18½	1·18½	1·22½	1·25	1·37½	1·49½	1·45	1·50
Sunbeam	1	·14	·23	·29	·48	·41½	·59	·58	·60	·52	·44	·35	·63½
Sacramento	5	·47½	·49	·49	·46	·43½	·42½	·46	·47	·46½	·46	·45	·47½
Silver King	20	31·00	36.00	33·00	32·50	32.00	33·00	35·00	35·00	40·00	40·00	40·00	38·00
Star Consolidated	1	1·20	1·20	1·35	1·25	1·19½	1·12	1·06½	1·08	·99½	·99½	·99	·93½
Showers Consolidated	5	.....	·41	·36½	·33	·32	·30	·33½	·41½	·42	·39	·38	·35
Tetro	1	·12	·12½	·11½	·08½	·08½	·06	·06	·07	·05	·08½	·05	·08½
Utah	10	·80	1·02½	·80	·80	·75	·80	·60	·50	·70	1·00	·93	·75
Valeo	1	1·30	1·45	1·45	1·40	1·20	1·25	1·32½	1·24	1·05	1·13	1·09½	1·00
West Mountain Placer	1	·75	·60	·40	·60	·60	·45	.....	·63½	·40	·47	·25	.....
Yankee Consolidated	10	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

$13 to $12 and dividends were paid at intervals during the year. Ontario has steadily strengthened. Tintic stocks have supplied the bulk of the trading and furnished a number of sensations, pleasant and otherwise. Bullion-Beck maintained a regular $10,000 dividend each month until December. Centennial-Eureka, the Utah corporation, closed a glorious career, and Centennial-Eureka, the Maine corporation, now does business on the Boston Exchange. The original company had 30,000 shares; the new company 200,000—100,000 issued and 100,000 set apart for treasury needs. Joe Bowers will increase its shares from 400,000 to 700,000, the increase going to the Burnham heirs; this adjusts the title difficulty out of court. Grand Central's dividend career came to a halt in September, when a trespass suit was brought against the Mammoth. Mammoth is no less erratic than before. Since buying the control in June last, $260,000 in dividends have been paid.

FLUCTUATIONS OF UTAH MINING SHARES IN 1899.—*Continued.*

Companies.	July.		August.		September.		October.		November.		December.	
	1	15	5	19	2	16	7	21	4	18	2	16
Anchor	·40	·20	·50	....	·50	.....	·50	·50	·50	.....	.....	.....
Ajax	1·25	·90	1·25	1·35	1·00	1·00	·75	·98	·90	·90	·60	·45
Alliance	·05	.....	·25	·35	·41	.....	.....	.....	.....	·25	·25	·20
Alice	·60	·60	·67	·62½	·60	·50	·60	·40	·45	·45	·35½	·30
Bullion-Beck	5·75	5·60	5·40	5·55	5·05	5·00	3·65	3·85	3·70	3·80	3·50	3·10
Buckeye	·08	·08½	·08	.....	.....	.....	·01½	.....	·02½	·02½	·02½	·02½
Centennial Eureka	49·00	50·37½	60·00	60·00	68·50	68·00	68·00	8·00	.....	.....	.....	.....
Chloride Point	·78½	·75	·67½	·60	·44	·38	·18	·25	·23½	·18½	·17½	·12
Daly	1·38	1·55	1·48	1·46	1·38	1·37	1·46	1·41	1·66½	1·68	1·55	1·70
Daly-West	11·85	11·90	11·81	12·30	13·01	12·71	13·34	12·73	12·94	12·45	12·17½	11·85
Dalton & Lark	·09½	·08½	·08½	·09½	·08½	·10	·08	·03	·04	·05	·05	·04½
Dalton	·02	·01½	·01½	.....	·02½	.....	·01	·01	·01½	.....	·02½	·02
Dexter	1·90	1·85	2·00	2·16	1·95	2·20	2·58	2·45	2·25	2·24	2·33½	2·12
Daisy	·26	·28	·35½	·28½	·25½	·21½	·19	·19½	·21	·09½	·09½	·18
Eagle	·05	·04½	·06½	·06½	·05½	·04½	·04½	·03½	·04½	·02	·03	.....
Eagle & Blue Bell	1·33	1·40	1·67½	1·90	1·60	1·70	1·72½	1·45	1·50	1·30	1·40	1·43½
Four Aces	·21½	·21½	·24½	·28	·27½	·25½	·19½	·13½	·09	·06½	·06	·05½
Geyser-Marion	·50	·57	·48½	·45	·39½	·43	·30	·34	·30	·29	·24	·21
Galena	·21	·26½	·34	·35	·39	·36½	·32	·30	·35	·30	·25	·26
Grand Central	8·55	8·90	9·30	7·85	7·57½	5·25	4·97½	5·50	5·92½	5·50	5·79	5·95½
Golden Eagle	·08½	·06½	·05½	·05½	·04½	·04½	·04½	·04½	·05½	·04½	·05½	·04½
Horn Silver	1·35	1·25	1·25	1·00	1·22½	1·20	1·00	1·17½	1·00	1·00	1·05	·90
Homestake	·06½	·08	·11½	·12	·10½	·11½	·07½	·04½	·09	·10	·08½	·08½
Ingot	·20	·10	·15	·16	·15½	·13	·08½	·09	·14½	·09	·09	·08
Joe Bowers	·14½	·17	·18½	·19½	·19½	·17½	·16½	·17	·17½	·17½	·17½	·15½
Joe Bowers Ex	·06	·06½	·08½	·08½	·07	·08	·07½	·05	·07½	·06½	·06½	·06½
La Reine	1·00	·95	·96	·95	·80½	·70	·65	·60	·50	·50	·50	·50
Lower Mammoth	·27½	·32	·37	·40	·27½	·23	·25	·26	·52½	·45	·40	·45½
Little Pittsburg	·06½	·08	·07	·07½	·06	·04½	·02½	·08½	·06½	·05	·05½	·05½
Mammoth	1·85	1·89	2·20	2·09½	1·98½	2·09	2·41	3·48½	3·01	2·95	2·85	2·90
Mercur	7·00	6·75	6·90	6·90½	7·08	7·28	7·00	6·93	6·80	6·75	6·50	6·60
May Day	·61	·58	·62	·60	·76	·70	·70	·61	·60	·62	·55	·68½
Northern Light	·60	·53½	·57	·44½	·28½	·29½	·20½	·14½	·18½	·15½	·16½	·15½
North Swansea	·15½	·15½	·15½	·15½	·14½	·14	.....	.....	.....	·10	·11	....
Omaha	·26½	·26	·26	·28	·24½	·15	·13½	·13	·13	·07	·09	·04
Ontario	7·25	6·90	7·75	8·40	8·25	7·50	8·60	8·20	7·95	7·75	8·00	8·51
Petro	.....	·98½	·98½	·90	·81½	·71½	·60½	·40	·27	·27½	·32½	·30
Richmond-Anaconda	·10	·10	·11	·10	·11	·11	.....	·29	.....	·20	.....	·20
Sunshine	·60	·67½	·58½	·60	·51	·48½	·31	·30	·31	·27	·23½	·16½
Swansea	3·85	3·90	4·08	4·25	4·27	4·25½	3·90	3·82½	3·50	3·73	3·80	3·55
South Swansea	1·51	1·70	1·70	1·60	1·62	1·63	1·65	1·35	1·31½	1·29	1·27½	1·23
Sunbeam	·39	·42½	·51	·47½	·55½	·55½	·57½	·67½	·61½	·74½	·66½	·54½
Sacramento	·47	·46½	·46½	·47	·47½	·47½	·47½	·46	·30½	·31½	·30½	·29½
Silver King	40·25	42·00	45·00	48·50	49·00	49·00	50·00	45·00	51·25	50·00	51·50	52·00
Star Consolidated	·92½	·72	·70½	·66½	·50	·55	·37½	·43	·46½	·42	·82½	·22½
Showers Consolidated	·35	·37	·56½	·52	·45½	·43	·44	·36	·20½	·33	·36	·40
Tetro	·07½	·08½	·09	·12½	·10½	·09	·08½	·06	·05½	·06½	·07	·06
Utah	·65	·70	·72½	·60	·75	·70	·70	·60½	·60	·50	.....	.....
Valeo	·92½	·80	·65	.....	·77	·66	·63½	·85	·75	·81½	·70	·60
West Mountain Placer	·25	·25	·35	·36	·30	·20	·25½	·11	·21	·30	·35	·35
Yankee Consolidated	.....	.....	·11	·14	·15½	·15½	·16	·15½	·19½	·20	·13½	·11½

NOTE.—This table gives the closing bid quotation on the first and third Saturday of each month. As near as may be, this affords the best index of the Salt Lake market and generally is but little under what the shares can be bought for particularly for the active traders.

FLUCTUATIONS OF MINING STOCKS AT LONDON DURING 1899.

Name of Company.	Location.	Shares Issued.	Par Value.	Divid'nds Paid in 1899.	January-March. H.	January-March. L.
			£ s. d.	£ s. d.	£ s. d.	£ s. d.
Alaska-Mexican, g.	Alaska	180,000	1 0 0	1 7½	1 10 0	17 6
Alaska-Treadwell, g.	Alaska	200,000	5 0 0	6 0	4 17 6	4 7 6
Anaconda, c.	Montana	1,200,000	5 0 0	18 3½	9 17 6	6 11 3
Cons. Goldfields.	Mexico	223,985	1 0 0	. . . . .	1 0 0	10 0
De Lamar, g. s.	Idaho	400,000	1 0 0	6	5 6	3 0
Elkhorn Priority, s.	Colorado	87,500	1 0 0	. . . . .	3 9	1 3
Golden Gate, g.	California	80,000	1 0 0	. . . . .	5 0	1 3
Grand Central, g. s.	Mexico	250,000	1 0 0	4 0	1 12 6	1 5 0
Hall Mines, c. s.	British Col.	250,000	1 0 0	. . . . .	1 0 0	5 0
Le Roi, g.	British Col.	200,000	5 0 0	5 0	5 17 6	5 10 0
Lillie, g.	Colorado	250,000	1 0 0	1 10 0	. . . . .	. . . . .
Montana, g. s.	Montana	657,128	1 0 0	6	8 3	6 3
Mountain Copper	California	250,000	5 0 0	14 0	7 10 0	5 15 0
Newfoundland, c.	Newfoundland	250,000	1 0 0	. . . . .	. . . . .	. . . . .
Plumas-Eureka, g.	California	140,625	2 0 0	. . . . .	3 6	2 6
Richmond, g. s. l.	Nevada	54,000	5 0 0	. . . . .	8 9	5 0
Sierra Buttes, g.	California	122,500	2 0 0	. . . . .	2 6	1 6
Stratton's Independence, g.	Colorado	1,000,000	1 0 0	4 0	. . . . .	. . . . .
Colombian Hydraulic., g.	Colombia	75,000	1 0 0	6	15 0	8 9
Copiapo, c.	Chile	100,000	2 0 0	12 3	3 12 6	2 8 9
Frontino & Bolivia, g.	Colombia	128,662	1 0 0	4 6	2 10 0	2 0 0
St. John del Rey, g.	Brazil	425,482	1 0 0	2 3	1 11 3	1 6 0
Tolima A., s. g.	Colombia	14,000	5 0 0	. . . . .	2 0 0	1 15 0
Tolima B., s.g.	Colombia	6,000	5 0 0	. . . . .	1 5 0	7 6
Utah Con., g.	Utah	300,000	1 0 0	. . . . .	10 5 0	4 5 0
Velvet, g.	British Col.	100,000	1 0 0	. . . . .	. . . . .	. . . . .
Ymir, g.	British Col.	125,000	1 0 0	. . . . .	1 6 3	16 3
British Am. Corp	British Col.	1,500,000	1 0 0	. . . . .	18 3	14 3
Linares, l.	Spain	15,000	3 0 0	1 5 6	10 0 0	7 15 0
Mason & Barry, c. sul.	Portugal	185,172	2 0 0	5 0	4 2 6	3 2 6
Rio Tinto, c.	Spain	325,000	5 0 0	1 9 9	40 17 6	31 15 0
Rio Tinto, Pref.	Spain	325,000	5 0 0	5 0	6 10 0	5 18 9
Tharsis, c.	Spain	625,000	2 0 0	11 0	9 2 6	7 6 3
Libiola, c.	Italy	50,400	5 0 0	4 0	3 0 0	1 17 6
Assoc. Gold Mines	W. Australia	450,000	1 0 0	4 0	6 16 3	5 16 3
Broken Hill Prop., s.	N. S. Wales	960,000	8 0	3 6	2 13 9	2 2 6
Great Boulder Prop.	W. Australia	1,750,000	2 0	2 0	1 8 9	1 1 3
Hannan's Brownhill, g.	W. Australia	140,000	1 0 0	1 15 0	8 11 3	7 8 9
Ivanhoe Gold Corp.	W. Australia	200,000	5 0 0	1 0 0	8 6 3	6 12 6
Kalgoorlie, g.	W. Australia	115,500	1 0 0	. . . . .	7 10 0	6 12 6
Lake View Consols, g.	W. Australia	250,000	1 0 0	2 10 0	23 0 0	19 3 9
Mt. Lyell M. & R., c.	Tasmania	275,000	8 0 0	12 0	9 17 6	6 5 0
Mt. Morgan, g.	Queensland	1,000,000	1 0 0	6 9	5 17 6	5 5 0
Waihi, g.	New Zealand	320,000	1 0 0	8 0	6 1 3	4 17 6
W. Austr. Joint Stock, L. & F.	W. Australia	920,000	1 0 0	. . . . .	12 6	8 0
Champion Reef, g.	Colar Fields	440,000	10 0	16 6	5 1 3	4 6 3
Mysore Gold.	Colar Fields	500,000	10 0	14 3	5 12 6	5 0 0
Nundydroog, g.	Colar Fields	242,000	1 0 0	5 6	3 16 3	3 0 0
Ooregum, g.	Colar Fields	145,000	1 0 0	3 6	3 18 9	3 6 3
Ooregum, pfd, g.	Colar Fields	108,191	1 0 0	5 6	4 11 3	3 17 6
Angelo, g.	Transvaal	275,000	1 0 0	5 0	8 12 6	6 16 3
Bonanza, g.	Transvaal	200,000	1 0 0	11 0	4 11 3	4 2 6
British S. Af. Chartered.	South Africa	4,373,559	1 0 0	. . . . .	3 16 3	2 17 6
Cape Copper, c.	Cape Colony	300,000	2 0 0	12 6	5 7 6	3 17 6
Cape Copper, pfd.	Cape Colony	75,000	2 0 0	12 6	5 5 0	3 10 0
City & Suburban, g.	Transvaal	340,000	4 0 0	14 0	6 15 0	5 2 6
Cons. Deep Level, g.	Transvaal	187,250	1 0 0	10 0	3 12 6	2 17 6
Crown Reef, g.	Transvaal	120,000	1 0 0	1 18 0	18 5 0	14 0 0
De Beers Cons., d.	Cape Colony	790,000	5 0 0	2 0 0	30 12 6	26 6 3
Durban Roodepoort, g.	Transvaal	125,000	1 0 0	19 0	6 15 0	5 15 0
Ferreira, g.	Transvaal	90,000	1 0 0	3 0 0	26 0 0	23 0 0
Geldenhuis Deep, g.	Transvaal	300,000	1 0 0	17 0	12 15 0	10 10 0
Geldenhuis Est., g.	Transvaal	200,000	1 0 0	17 6	8 7 6	7 7 6
Ginsberg, g.	Transvaal	160,000	1 0 0	9 0	4 0 0	3 2 6
Henry Nourse, g.	Transvaal	125,000	1 0 0	1 0 0	9 15 0	8 12 6
Heriots, g.	Transvaal	111,864	1 0 0	15 0	8 0 0	6 10 0
Jagersfontein, d.	Orange Fr. St.	200,000	5 0 0	9 0	13 7 6	9 11 3
Johannesburg Con. Invest.	South Africa	2,682,396	1 0 0	. . . . .	2 6 3	1 3 9
Jubilee, g.	Transvaal	50,000	1 0 0	15 0	7 0 0	5 2 6
Jumpers, g.	Transvaal	100,000	1 0 0	19 0	6 15 0	5 0 0
Kleinfontein, g.	Transvaal	231,250	1 0 0	. . . . .	3 17 6	2 11 3
Langlaagte Est., g.	Transvaal	470,000	1 0 0	6 0	4 7 6	3 12 6
May Con., g.	Transvaal	288,750	1 0 0	9 0	5 2 6	4 2 6
Meyer & Charlton, g.	Transvaal	100,000	1 0 0	15 0	6 17 6	5 0 0
Namaqua, c.	Cape Colony	94,331	1 0 0	6 6	4 0 0	3 3 9
Primrose, g.	Transvaal	300,000	1 0 0	12 0	5 17 6	4 7 6
Rand Mines, g.	South Africa	337,136	1 0 0	1 15 0	45 2 6	33 12 6
Robinson, g.	Transvaal	550,000	5 0 0	17 0	11 2 6	8 17 6
Sheba, g.	Transvaal	1,075,000	1 0 0	. . . . .	1 18 9	1 6 3
Simmer & Jack Prop., g.	Transvaal	940,000	5 0 0	7 6	6 12 6	5 0 0
Wemmer, g.	Transvaal	80,000	1 0 0	1 10 0	13 5 0	11 10 0
Wolhuter, g.	Transvaal	215,000	4 0 0	2 0	6 15 0	4 10 3
Worcester, g.	Transvaal	95,772	1 0 0	6 0	3 7 6	2 11 0

FLUCTUATIONS OF MINING STOCKS AT LONDON DURING 1899.—*Continued.*

Name of Company.	April–June.		July–September.		October–December.		Year.	
	H.	L.	H.	L.	H.	L.	H.	L.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
Alaska-Mexican, g.	1 2 6	17 6	1 2 6	17 6	1 2 6	17 6	1 2 6	1 7 6
Alaska-Treadwell, g.	4 17 6	4 10 0	5 7 6	4 10 0	5 5 0	50 0	5 7 6	4 7 6
Anaconda, c. s.	10 2 6	8 16 3	11 8 9	11 7 6	10 0 0	9 7 6	11 8 9	6 11 3
Cons. Goldfields.	1 5 0	1 0 0	1 0 0	17 6	1 0 0	11 3	1 5 0	11 3
De Lamar, g. s.	5 6	4 0	5 0	3 0	5 6	4 0	5 6	3 0
Elkhorn Priority, s.	3 9	1 3	3 9	1 3	3 9	1 3	3 9	1 3
Golden Gate, g.	3 9	1 0	2 0	1 0	2 0	1 0	5 0	1 0
Grand Central, g. s.	1 6 3	1 0 0	1 6 3	16 3	1 11 0	1 1 3	1 12 6	16 3
Hall Mines, c. s.	15 0	10 0	13 9	8 9	11 3	6 3	1 0 0	5 0
Le Roi, g.	7 10 0	5 12 6	6 15 0	6 2 6	6 12 6	6 7 6	7 10 0	5 10 0
Lillie, g.					1 0 0	15 0	1 0 0	15 0
Montana, g. s.	7 6	5 6	5 0	3 9	4 9	3 0	8 3	3 0
Mountain Copper.	7 5 0	5 17 6	8 12 6	7 11 3	8 0 0	7 2 6	8 0 0	5 15 0
Newfoundland, c.					12 6	5 0	12 6	5 0
Plumas-Eureka, g.	3 6	2 6	2 6	1 3	5 0	2 0	5 0	1 3
Richmond, g. s. l.	7 6	3 9	6 3	1 3	6 3	3 9	8 9	1 3
Sierra Buttes, g.	2 6	6	1 0	6	3 9	2 6	3 9	6
Stratton's Independence, g.					3 1 3	2 10 0	3 1 3	2 10 0
Columbian Hydraulic, g.	11 3	6 3	8 9	3 9	7 6	3 9	11 3	3 9
Copiapo, c.	3 17 6	3 0 0	3 17 6	3 15 0	4 2 6	3 12 6	4 2 6	2 8 9
Frontino & Bolivia, g.	2 5 0	1 17 6	2 6 3	2 1 3	2 6 3	1 18 9	2 10 0	1 17 6
St. John del Rey, g.	1 12 6	1 7 6	1 10 0	1 7 6	1 10 6	1 9 6	1 12 6	1 6 0
Tolima A., s. g.	2 0 0	15 0	3 0 0	1 15 0	3 0 0	2 10 0	3 0 0	1 15 0
Tolima B., s. g.	1 10 0	15 0	2 0 0	1 0 0	2 0 0	1 10 0	2 0 0	15 0
Utah Con., g.	9 5 0	8 0 0	9 0 0	8 0 0	8 0 0	7 7 6	10 5 0	4 5 0
Velvet, g.					1 12 6	1 3 9	1 12 6	1 3 9
Ymir, g.	1 1 3	18 9	1 5 0	1 1 3	1 10 0	1 3 9	1 10 0	16 3
British Am. Corp.	1 14 3	1 1 3	1 2 0	1 0 0	1 1 6	1 0 9	1 14 3	14 3
Linares, l.	9 0 0	8 0 0	8 10 0	7 10 0	10 0 0	7 10 0	10 0 0	7 10 0
Mason & Barry, c. sul.	4 15 0	3 15 0	4 0 0	3 12 6	4 7 6	3 15 0	4 15 0	3 2 6
Rio Tinto, c.	46 2 6	41 16 3	47 2 6	45 10 0	48 7 6	46 12 6	48 7 6	31 15 0
Rio Tinto, Pref.	6 6 3	6 2 6	6 7 6	6 2 6	6 2 6	5 17 6	6 10 0	5 17 6
Tharsis, c.	8 12 6	8 2 6	8 5 0	7 15 0	8 15 0	8 5 0	9 2 6	7 6 3
Libiola, c.	2 5 0	1 17 6	2 0 0	1 12 6	1 17 6	1 12 6	2 0 0	1 12 6
Assoc. Gold Mines.	9 16 3	6 15 0	12 12 6	9 6 3	12 15 0	10 15 0	12 15 0	5 16 3
Broken Hill Prop., s.	2 6 3	2 3 9	2 5 0	2 2 0	2 5 6	2 2 6	2 18 9	2 2 0
Great Boulder Prop.	1 10 0	1 8 6	1 19 0	1 0 0	2 0 0	1 18 0	2 0 0	1 1 3
Hannan's Brownhill, g.	11 10 0	8 15 0	12 17 6	11 2 6	12 7 6	11 15 0	12 17 6	7 8 9
Ivanhoe Gold Corp.	11 10 0	9 5 0	17 0 0	11 10 0	17 2 6	16 10 0	17 2 6	6 12 6
Kalgoorlie, g.	10 7 6	8 7 6	12 0 0	8 17 6	10 15 0	10 2 6	10 15 0	6 12 6
Lake View Consols, g.	21 12 6	15 0 0	23 0 0	21 10 0	21 16 3	20 12 6	23 0 0	15 0 0
Mt. Lyell, M. & R., c.	9 0 0	8 0 0	11 2 6	9 2 6	11 2 6	10 12 6	11 2 6	6 5 0
Mt. Morgan, g.	5 10 0	5 5 0	5 8 9	5 5 0	5 5 0	5 2 6	5 17 6	5 2 6
Waihi, g.	7 5 0	5 15 0	7 15 0	7 0 0	10 10 0	8 6 3	10 10 0	4 17 6
W. Austr. Joint Stock, L. & F.	7 0	6 6	8 3	6 3	4 6	3 6	12 6	3 6
Champion Reef, g.	5 2 6	4 16 3	5 6 3	5 1 3	5 6 3	5 2 9	5 6 3	4 6 3
Mysore Gold.	5 11 3	5 7 6	5 11 3	5 10 0	5 16 3	5 11 3	5 16 3	5 0 0
Nundydroog g.	3 11 3	3 0 0	3 3 9	3 0 0	3 10 0	2 6 3	3 16 3	2 6 3
Ooregum, g.	3 16 3	3 3 9	3 16 3	3 13 9	3 15 0	3 3 9	3 18 9	2 3 9
Ooregum, pfd., g.	4 11 3	4 2 6	4 15 0	4 3 9	4 16 3	4 3 9	4 16 3	3 17 6
Angelo, g.	7 12 6	7 5 0	7 5 0	7 0 0	7 10 0	6 17 6	8 12 6	6 16 3
Bonanza, g.	4 15 0	4 7 6	4 10 0	4 5 0	4 10 0	4 2 6	4 15 0	4 2 6
British S. Af. Chartered.	3 17 6	3 0 0	3 1 3	2 16 3	4 6 3	3 0 0	3 17 6	2 16 3
Cape Copper, c.	4 11 3	4 0 0	4 5 0	4 0 0	5 16 3	4 15 0	5 16 3	3 17 6
Cape Copper, pfd.	4 2 6	3 17 6	4 5 0	4 0 0	5 7 6	4 15 0	5 7 6	3 10 0
City & Suburban, g.	6 5 0	5 12 6	5 16 3	5 7 6	5 13 9	5 2 6	6 15 0	5 2 6
Cons. Deep Level, g.	4 0 0	3 5 0	1 12 6	1 5 0	2 0 0	1 0 0	3 12 6	1 0 0
Crown Reef, g.	20 0 0	17 0 0	17 0 0	15 15 0	16 0 0	15 0 0	20 0 0	14 0 0
De Beers Cons., d.	28 16 3	27 2 6	28 10 0	26 7 6	29 0 0	26 2 6	30 12 6	26 2 6
Durban Roodepoort, g.	6 10 0	6 0 0	6 5 0	5 17 6	6 2 6	5 2 6	6 15 0	5 2 6
Ferreira, g.	24 10 0	23 10 0	24 5 0	22 0 0	22 0 0	20 10 0	26 0 0	20 10 0
Geldenhuis Deep, g.	11 7 6	10 12 6	11 0 0	9 5 0	16 2 6	9 2 6	12 15 0	9 2 6
Geldenhuis Est., g.	8 17 6	8 6 3	7 16 3	6 5 0	6 17 6	6 5 0	8 17 6	6 5 0
Ginsberg, g.	3 13 9	3 8 9	3 12 6	3 5 0	3 5 0	3 0 0	4 0 0	3 0 0
Henry Nourse, g.	9 12 6	8 10 0	8 12 6	7 17 6	8 12 6	8 0 0	9 15 0	7 17 6
Heriots, g.	7 7 6	6 17 6	7 2 6	6 12 6	6 10 0	6 5 0	8 0 0	6 5 0
Jagersfontein, d.	14 3 3	13 12 6	14 0 0	13 10 0	15 0 0	13 0 0	15 0 0	13 0 0
Johannesburg Con. Invest.	1 18 9	1 8 9	1 15 0	1 10 0	2 1 3	1 13 9	2 6 3	1 3 9
Jubilee, g.	7 15 0	6 5 0	7 2 6	6 0 0	6 15 0	5 10 0	7 15 0	5 2 6
Jumpers, g.	6 17 6	5 18 0	5 10 0	5 0 0	5 7 6	4 17 6	6 17 6	4 17 6
Kleinfontein, g.	3 2 6	2 15 0	2 15 0	2 12 6	2 18 0	2 10 0	3 17 6	2 11 3
Langlaagte Est., g.	3 16 3	3 7 6	3 7 6	3 2 6	3 10 0	3 5 0	4 7 6	3 2 6
May Con., g.	5 11 3	5 7 6	5 7 6	4 12 6	4 16 3	4 2 6	5 11 3	4 2 6
Meyer & Charlton, g.	6 13 9	6 3 9	6 17 6	5 17 6	5 15 0	4 12 6	6 17 6	4 12 6
Namaqua, c.	4 7 6	3 10 0	4 7 6	4 0 0	4 18 9	4 8 9	4 18 9	3 8 9
Primrose, g.	5 0 0	4 6 3	4 10 0	4 1 3	4 10 0	4 2 6	5 17 6	4 1 3
Rand Mines, g.	41 10 0	39 15 0	39 7 6	36 5 0	43 12 6	37 5 0	45 2 6	33 12 6
Robinson, g.	11 0 0	10 12 6	10 10 0	9 7 6	9 12 6	9 0 0	11 2 6	8 17 6
Sheba, g.	1 12 6	1 3 9	1 6 3	1 0 0	1 6 3	1 0 0	1 18 9	1 0 0
Simmer & Jac'k, Prop., g.	6 10 0	6 2 6	6 8 9	5 17 6	6 5 0	5 12 6	6 12 6	5 0 0
Wemmer, g.	14 0 0	13 2 6	13 10 0	12 15 0	12 10 0	11 15 0	14 0 0	11 10 0
Wolhuter, g.	5 16 3	4 15 0	4 17 6	4 7 6	4 18 9	4 15 0	6 15 0	4 7 6
Worcester.	3 10 0	3 0 0	3 2 6	2 10 0	2 17 6	2 5 0	3 10 6	2 10 0

NOTE.—c., copper; d., diamonds; g., gold; l., lead; s., silver; sul., sulphur.

DIVIDENDS PAID BY AMERICAN MINES $1=$1,000; total, full amount).

Name and Location of Company.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Paid.
Adams, s.l., Colo.	8	.....	.....	6	.....	.....	.....	.....	$603,500
Ætna Con., q., Cal.	.....	.....	20	20	40	40	40	60	210,000
Alamo, Utah	.....	.....	.....	.....	.....	.....	.....	3	2,500
Alaska-Mexican, g., Alaska	.....	.....	120	90	90	80	72	72	429,031
Alaska-Treadwell, g., Alaska (a)	300	375	375	400	350	300	300	300	4,220,000
Alice, s.g., Mont.	.....	.....	.....	.....	.....	80	20	.....	1,075,000
Alliance, g., Colo.	.....	.....	.....	.....	.....	.....	.....	32	31,500
Alliance, s.l., Utah	.....	.....	.....	.....	.....	5	.....	.....	5,000
Amalgamated, c., Mont.	.....	.....	.....	.....	.....	.....	.....	3000	3,000,000
Amazon, g., Colo.	.....	.....	.....	.....	.....	.....	.....	20	20,000
American Coal, Md.	.....	.....	98	105	120	120	135	150	727,500
Am. Dev. & Mg., g.s.c., Mont.	.....	.....	52	24	.....	.....	.....	.....	76,422
American, g.s.l., Colo.	.....	.....	.....	.....	.....	36	54	60	446,000
American, z., Kans.	.....	.....	.....	.....	.....	.....	.....	20	20,000
Am. Sm. & Ref., pref.	.....	.....	.....	.....	.....	.....	.....	1137	1,137,500
American Steel & Wire.	.....	.....	.....	.....	.....	.....	.....	2800	2,800,000
American Zinc, Kan.	.....	.....	.....	.....	.....	.....	.....	20	20,000
Am. Zinc-Lead Sm., Mo.	.....	.....	.....	.....	.....	.....	.....	120	120,000
Anaconda, g., Colo.	13	.....	.....	.....	.....	.....	.....	.....	12,500
Anaconda, c., Mont.	.....	.....	.....	.....	2250	3000	3000	3900	12,150,000
Anchoria-Leland, g., Colo.	.....	.....	.....	.....	30	72	60	36	198,000
Apollo Con., g., Alaska.	.....	.....	.....	.....	.....	100	.....	40	140,000
Appie Eilen, g., Colo.	.....	.....	.....	.....	.....	.....	25	.....	25,000
April Fool, g., Nev.	.....	.....	.....	.....	.....	.....	.....	16	16,000
Argentum-Juniata, s., Colo.	.....	.....	.....	156	.....	.....	.....	.....	156,000
Argonaut, g., Cal.	.....	.....	.....	.....	.....	.....	180	240	420,000
Arizona, c., Ariz.	.....	.....	.....	.....	.....	.....	.....	421153	888,412
Aspen, s.l., Colo.	100	100	40	.....	.....	.....	.....	.....	900,000
Associated, g., Colo.	.....	.....	.....	.....	.....	.....	50	12	84,000
Atlantic, c., Mich.	.....	.....	.....	.....	.....	40	40	.....	780,000
Aurora, l., Mich.	100	200	.....	.....	50	.....	50	50	890,000
Bald Butte, g., Mont.	85	49	172	127	88	88	105	120	769,648
Bangkok-Cora Belle, s., Colo.	.....	.....	.....	54	6	.....	.....	.....	106,000
Belden, F. E., m., N. H.	45	60	60	48	4	.....	.....	.....	217,000
Big Seven, Cal.	.....	.....	.....	.....	.....	3	3	.....	6,000
Big Six, g.s., Colo.	.....	.....	.....	.....	3	5	7	.....	15,000
Bimetallic, s.g., Mont.	200	190	.....	.....	.....	.....	.....	.....	1,630,000
Bonanza Dev., N. Mex.	.....	.....	.....	.....	.....	.....	.....	1050	1,050,000
Bonanza, Wash.	.....	.....	.....	.....	.....	.....	.....	8	7,500
Boreel, s., Colo.	.....	60	28	.....	.....	.....	.....	.....	105,000
Boston-Aurora, pref., z., Mo.	.....	.....	.....	.....	.....	.....	.....	32	31,908
Boston, q., Cal.	.....	.....	.....	.....	.....	.....	.....	20	20,000
Boston & California, Cal.	.....	.....	.....	.....	.....	.....	.....	72	72,000
Boston & Colo. Sm., Colo.	.....	.....	.....	.....	.....	.....	.....	56	281,250
Boston & Montana, c.s., Mont.	.....	.....	275	1050	1500	1800	1950	5375	14,500,000
Boston-Duenweg, z., Mo.	.....	.....	.....	.....	.....	.....	.....	32	32,000
Boston Get There, z., Mo.	.....	.....	.....	.....	.....	.....	.....	11	11,250
Boston-Little Circle, z., Mo.	.....	.....	.....	.....	.....	.....	.....	80	80,000
Boston-Providence, z., Mo.	.....	.....	.....	.....	.....	.....	.....	8	7,500
Boston Sunflower, z., Mo.	.....	.....	.....	.....	.....	.....	.....	5	4,500
Breece, l., Colo.	.....	.....	.....	.....	.....	.....	10	40	70,000
Brotherton, i., Mich.	40	80	.....	.....	.....	.....	.....	.....	120,000
Bullion-Beck & Champion, Utah.	.....	.....	425	395	290	170	90	120	2,438,400
Bunker Hill & Sull., s.l., Ida.	.....	.....	.....	.....	.....	102	238	201	801,000
Calumet & Hecla, c., Mich.	2000	2000	1500	2000	2500	5000	5000	10,000	66,350,000
Cariboo-McKinney, g., B. C.	.....	.....	.....	49	76	48	67	72	311,965
Centennial-Eureka, s.g.l., Utah.	90	188	195	510	390	98	15	120	2,150,000
Central Lead, l., Mo.	.....	.....	.....	.....	.....	28	54	60	142,000
Champion, g., Cal.	41	41	41	27	.....	51	26	26	321,700
Charleston, p., S. C.	.....	140	.....	.....	.....	10	30	20	200,000
Church, g., Cal.	.....	.....	.....	5	.....	.....	.....	.....	5,000
Cleopatra, g.s., So. Dak.	.....	450	.....	.....	.....	.....	.....	.....	450,000
Cloverdale, z., Mo.	.....	.....	.....	.....	.....	.....	.....	4	4,000
C. O. D., g., Colo.	.....	.....	.....	20	5	.....	.....	.....	25,000
Colonial, l., Mo.	.....	.....	.....	.....	.....	.....	.....	10	10,000
Colorado Central, s., Colo.	55	28	.....	.....	.....	.....	.....	.....	502,661
Colorado Fuel & Iron, Colo.	.....	1203	160	.....	160	80	160	.....	1,762,500
Colorado Smelting, Mont.	.....	.....	.....	.....	.....	.....	250	100	1,945,000
Commodore, Colo.	.....	.....	.....	.....	.....	25	20	48	432,000
Commonwealth, z., Mo.	.....	.....	.....	.....	.....	.....	.....	20	20,000
Consolidation Coal, Md.	205	205	205	205	205	205	205	205	5,716,650
Consolidated Gold Mines, Colo.	.....	.....	.....	.....	.....	.....	.....	10	10,000
Con. Tiger & Poorman, Ida (b).	.....	.....	.....	.....	.....	.....	20	.....	20,000
Copper Queen, c.s., Ariz.	140	300	200	150	.....	.....	.....	.....	1,910,000
Coptic, s., Nev. (d).	.....	.....	.....	1	.....	.....	.....	.....	77,000
Cortez, s., Nev.	95	45	.....	.....	.....	.....	.....	.....	735,000
Crœsus, Cal.	.....	.....	.....	.....	.....	.....	.....	19	49,000
Crowned King, g.s.l., Ariz.	.....	.....	.....	.....	.....	.....	96	35	242,760
Dalton & Lark, g.s.l., Utah.	.....	.....	.....	.....	88	.....	.....	.....	87,500
Daly, s.l., Utah.	450	188	.....	.....	37	38	.....	.....	2,925,000
Daly-West, Utah.	.....	.....	.....	.....	.....	.....	.....	120	120,000
Deadwood-Terra, g., So. Dak. (e).	100	.....	.....	.....	100	80	30	.....	1,350,000
De Lamar, s.g., Idaho.	200	450	500	450	500	.....	48	48	2,346,000
Della S., Colo.	.....	.....	50	.....	.....	10	.....	.....	60,000

DIVIDENDS PAID BY AMERICAN MINES.—*Continued.*

Name and Location of Company.	1892.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Paid.
Delta, l.z., Mo	.....	.....	.....	.....	.....	.....	.....	5	$4,995
Doe Run, l., Mo	.....	.....	.....	.....	.....	30	30	30	90,000
Dutch, Cal	.....	.....	.....	.....	.....	8	17	.....	39,000
Eldorado, g., Cal	.....	.....	.....	.....	.....	.....	.....	10	10,000
Elkhorn, s.l., Colo	303	225	142	50	20	.....	40	.....	1,325,000
Elkton Con., g., Colo	.....	.....	60	.....	90	260	220	68	720,711
El Paso, Colo	.....	.....	.....	.....	.....	5	7	.....	12,398
Empire State, Idaho	.....	.....	.....	.....	.....	.....	67	251	318,088
Enterprise, s.g., Colo. (*k*)	450	125	.....	.....	.....	.....	50	.....	900,000
Fanny Rawlings, g., Colo	.....	.....	.....	.....	.....	.....	.....	20	20,000
Federal Steel	.....	.....	.....	.....	.....	.....	.....	3196	3,195,660
Fern, g., B. C	.....	.....	.....	.....	.....	.....	10,000	.....	10,000
Ferris-Haggerty, c.g.s., Wyo	.....	.....	.....	.....	.....	.....	.....	5	5,000
Florence, s., Mont	.....	.....	.....	35	54	54	25	44	230,000
Forepaugh, Colo	.....	.....	.....	16	.....	.....	.....	.....	16,000
Franklin, c., Mich	160	120	80	.....	.....	.....	.....	.....	1,240,000
Galena, s.l.g., Utah	.....	.....	.....	6	49	5	.....	.....	71,000
Garfield Con., g., Colo	.....	.....	.....	.....	.....	.....	.....	12	84,000
Geyser-Marion, g., Utah	.....	.....	.....	.....	.....	63	33	.....	96,000
Gold Coin of Victor, Colo	.....	.....	.....	.....	.....	120	120	210	360,000
Golden Cycle, g., Colo	.....	.....	.....	.....	.....	60	60	105	258,500
Golden Eagle, g., Colo	.....	.....	.....	.....	10	.....	.....	10	20,000
Golden Fleece, g., Colo	.....	.....	.....	192	162	6	.....	.....	569,480
Golden M. & Ext., Ont	.....	.....	.....	.....	.....	.....	.....	10	10,000
Golden Reward, g., So. Dak	60	60	.....	.....	.....	.....	30	.....	155,000
Golden Star, g., Ont	.....	.....	.....	.....	.....	.....	.....	46	45,500
Gold & Globe, g., Colo	.....	.....	.....	.....	11	25	4	.....	51,625
Gold King, g., Colo	.....	.....	.....	.....	.....	.....	.....	90	90,000
Grafton, g., Colo	.....	.....	.....	.....	.....	.....	.....	10	10,000
Grand Central, g., Utah	.....	.....	.....	.....	.....	.....	219	348	666,250
Granite Mountain, s.g., Mont	520	.....	.....	.....	.....	.....	.....	.....	12,120,000
Grass Valley Expl., Cal	.....	.....	.....	.....	.....	.....	.....	30	30,000
Gwin, g., Cal	.....	.....	.....	.....	.....	16	35	35	86,500
Hall Mines, Ltd., g. (ord.), B. C	.....	.....	.....	.....	.....	.....	60	60	120,000
Hecla Con., s.l., Mont	90	60	120	90	45	30	.....	15	2,190,000
Helena-Frisco, l.s., Idaho	20	.....	15	10	50	.....	.....	165	920,000
Hidden Treasure, g., Cal	.....	.....	.....	.....	.....	.....	.....	4	3,600
Holy Terror, g., So. Dak	.....	.....	.....	.....	.....	36	81	50	167,000
Homestake, g., So. Dak	150	150	256	344	375	375	636	963	8,143,750
Horn Silver, s.l., Utah	200	230	150	50	50	.....	80	20	5,250,000
Idaho, g., B. C	.....	.....	.....	.....	.....	.....	.....	28	292,000
Idaho, g., Cal	42	105	.....	.....	.....	.....	.....	.....	5,489,000
International, z., Mo	.....	.....	.....	.....	.....	.....	.....	3	3,000
Iowa, g.s.l., Colo	.....	.....	.....	.....	50	25	10	.....	90,500
Iron Mountain, s., Mont	.....	30	50	105	82	5	20	.....	507,500
Isabella, g., Colo	.....	.....	.....	23	180	68	.....	270	540,000
Jackson, g.s., Nev	.....	.....	.....	2	.....	.....	.....	.....	80,000
Jack Pot, g., Colo	.....	.....	.....	.....	.....	.....	.....	175	175,000
Jamison, g., Cal	.....	.....	.....	.....	.....	.....	.....	12	50,700
Jay Hawk, s.g., Mont. (*h*)	33	.....	.....	.....	.....	.....	.....	.....	33,375
Kennedy, g., Cal	500	480	540	184	.....	.....	.....	.....	1,796,000
Keystone, g., Cal	.....	.....	.....	.....	10	.....	.....	.....	10,000
Lake Superior, i., Mich	400	.....	.....	84	.....	.....	252	84	2,132,000
Last Chance, B. C	.....	.....	.....	.....	.....	.....	20	25	45,000
Last Chance, s., Colo	650	.....	.....	.....	.....	.....	.....	.....	650,000
Last Dollar, g., Colo	.....	.....	.....	.....	.....	.....	.....	30	30,000
Leadville, s.l., Colo	12	12	.....	.....	.....	.....	.....	.....	316,000
Le Roi, g., B. C	.....	.....	.....	.....	.....	.....	.....	240	1,305,000
Lexington, g., Colo	36	.....	.....	.....	.....	.....	.....	.....	36,000
Lillie, g., Colo. (*s*)	.....	.....	.....	.....	.....	.....	50	136	304,066
Little Tiger, g., Cal	.....	.....	.....	.....	.....	.....	.....	33	32,501
Maid of Erin, s.l.c., Colo. (*j*)	.....	.....	.....	.....	.....	.....	6	.....	740,000
Mammoth, g.s.c., Utah	.....	.....	.....	.....	60	.....	200	260	1,610,000
Marion Con., Colo	.....	.....	.....	.....	.....	.....	.....	5	300,000
Mary McKinney, g., Colo	.....	.....	.....	.....	.....	.....	.....	30	30,000
Maryland Coal, Md	.....	47	66	85	94	75	85	94	546,609
Matoa, g., Colo	.....	.....	.....	.....	.....	.....	25	.....	25,000
Mercur, g., Utah	.....	60	175	200	200	250	355	125	1,366,000
Merrimac, g., Cal	.....	.....	.....	.....	.....	9	.....	.....	9,000
Missouri Zinc Fields, pref., Mo	.....	.....	.....	.....	.....	.....	.....	145	145,000
Modoc, g., Colo	.....	.....	.....	.....	.....	10	90	45	145,000
Mollie Gibson, s., Colo. (*g*)	1700	1280	100	50	.....	.....	.....	.....	4,080,000
Montana, Ltd., g.s., Mont	.....	.....	.....	205	.....	.....	36	99	453,700
Montana Ore Purchasing, Mont	.....	.....	.....	160	320	160	160	560	1,360,000
Monument, g., Colo	.....	.....	.....	.....	.....	.....	13	6	18,124
Moon-Anchor, g., Colo	.....	.....	.....	.....	24	54	105	.....	216,000
Moose, g., Colo	.....	.....	108	72	6	.....	.....	.....	186,000
Morning Star Drift, g., Cal	83	72	106	154	132	142	82	158	847,200
Morse, g., Colo	.....	.....	.....	.....	.....	.....	.....	160	215,650
Moulton, s., Mont	30	.....	20	30	.....	.....	.....	40	500,0 0
Mount Rosa, g., Colo	.....	.....	.....	.....	5	10	20	40	75,000
Mount Shasta, Cal	.....	.....	.....	.....	.....	.....	.....	6	6,000
Mountain Copper, Cal	.....	.....	.....	.....	.....	31	62	1,080	1,173,750
Napa Con., q., Cal	70	70	50	80	70	80	80	110	1,060,000

DIVIDENDS PAID BY AMERICAN MINES.—*Continued.*

Name and Location of Company.	1892	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Paid.
National Lead	1043	1341	1490	1192	1043	1043	1192	1192	$9,536,672
National Salt	.....	.....	.....	.....	.....	.....	.....	126	126 000
New Central Coal, Md	50	50	50	.....	.....	.....	20	40	470,000
New Elkhorn, g., Colo	.....	.....	.....	.....	72	.....	.....	.....	72.000
New Idria, q., Cal	.....	.....	.....	.....	.....	10	70	110	190.000
N.Y. & Hond. Rosario, s.g., C.A	.....	.....	.....	.....	.....	165	165	180	1,160,000
New York, z., Mo	.....	.....	.....	.....	.....	.....	.....	21	21.000
North Star Mines of N. J., g., Cal	.....	.....	.....	.....	.....	.....	.....	50	50.000
Nugget, g., Colo	.....	.....	5	5	.....	.....	25	.....	35,000
Okanogan, g., Wash	.....	.....	.....	.....	.....	.....	.....	3	3.125
Olive, g., Ont	.....	.....	.....	.....	.....	.....	.....	12	12,000
Omaha, g., Cal	7	43	43	.....	.....	.....	.....	.....	106.100
Ontario, s., Utah	750	.....	.....	.....	180	203	.....	15	13,572,500
Original-Empire, g., Cal	.....	.....	.....	.....	.....	.....	.....	100	500.000
Orphan Belle, g., Colo	.....	.....	.....	.....	.....	.....	.....	83	197,899
Osceola, c., Mich	150	100	.....	100	125	150	300	558	3,080,500
Painlico, g., Nev	12	.....	.....	.....	.....	.....	.....	.....	189.000
Pandora, g., Mont	3	.....	.....	.....	.....	.....	.....	.....	6.000
Parrot, c.s., Mont	216	138	67	.....	.....	.....	483	1,242	3,380,868
Pennsylvania Coal, Pa	800	800	1,000	800	1,050	800	800	800	4,050,000
Pennsylvania Con., g., Cal. (*i*)	.....	.....	.....	.....	.....	18	28	68	136,475
Petro, s., Utah	.....	.....	.....	.....	.....	.....	.....	15	32.000
Pharmacist, g., Colo	36	44	.....	.....	.....	.....	.....	.....	80,000
Pioneer, g., Cal	.....	.....	.....	.....	.....	.....	50	13	62,500
Pleasant Valley, c., Utah	.....	20	.....	.....	.....	.....	.....	.....	593,056
Plumas Eureka, g., Cal	51	51	51	51	25	25	.....	.....	2,713,169
Portland, g., Colo	.....	.....	67	556	240	330	570	730	2,557,080
Princess, g., Colo	.....	.....	.....	.....	.....	45	.....	.....	45,000
Queen Bess Propr., s.l., B. C	.....	.....	.....	.....	.....	.....	.....	25	25,000
Quicksilver, q., Cal. (*l*)	.....	.....	.....	.....	.....	.....	.....	93	2,497,092
Quincy, c., Mich	350	300	400	600	1000	800	650	950	11,070,000
Rambler-Cariboo Con., s.l., B. C	.....	.....	.....	.....	.....	.....	40	20	60,000
Raven, g., Colo	.....	.....	.....	.....	.....	.....	20	50	69.500
Reco, g., B. C	.....	.....	.....	.....	.....	250	28	.....	297.500
Red Cloud, s.l., Idaho	80	70	10	.....	.....	.....	.....	.....	180,000
Republic Con., g., Wash	.....	.....	.....	.....	.....	.....	120	158	277,500
Rescue, g., N. M	12	.....	.....	.....	.....	.....	.....	.....	12,000
Reward, g., Cal	.....	.....	.....	.....	.....	.....	.....	20	20,000
Richmond Con., s.l., Nev	.....	14	.....	.....	.....	.....	.....	.....	4,386,780
Rico-Aspen, Colo	.....	50	300	.....	.....	.....	.....	.....	350.000
Royal Con., B. C	.....	.....	.....	.....	.....	.....	.....	50	1,050,000
Running Lode, g., Colo	6	1	.....	.....	.....	.....	.....	.....	27,000
Russell Irwin, z., Mo	.....	.....	.....	.....	.....	.....	.....	15	15,000
Sacramento, g.s.l., Utah	.....	.....	.....	.....	15	15	60	50	128,000
Saint Joseph, l., Mo	150	150	150	150	.....	144	113	113	2,897,500
Santa Rosalia, g., Cal	.....	.....	.....	.....	.....	.....	.....	5	130,000
Seventy-Six, g., Colo	.....	.....	.....	.....	.....	.....	2	.....	2,950
Sierra Buttes, g., Cal	27	27	27	20	7	.....	.....	.....	1,598,383
Silver King, g.s.l., Utah. (*p*)	.....	.....	.....	263	375	450	450	625	2,450,000
Small Hopes, s., Colo	38	25	.....	.....	25	.....	25	25	3,225,000
Smuggler, s.l.z., Colo	.....	.....	.....	.....	.....	.....	120	125	1,185,000
Smuggler Union, g.s., Colo. (*m*)	.....	.....	.....	.....	150	.....	.....	.....	150,000
South Eureka, g., Cal	.....	.....	.....	.....	.....	.....	12	.....	12,000
South Swansea, s.l., Utah	.....	.....	.....	.....	.....	68	38	53	165,000
Squaw Mountain, g., Colo	.....	.....	.....	.....	.....	.....	.....	10	10,000
Standard Con., g., Cal. (*n*)	40	20	40	20	.....	40	27	60	3,899,226
Standard, Idaho	.....	.....	.....	.....	.....	.....	.....	20	1,745,000
Stratton's Independence, Colo	.....	.....	.....	.....	.....	.....	.....	960	960,000
Strong, g., Colo	.....	.....	.....	.....	.....	250	300	300	850,000
Swansea, Utah	.....	.....	.....	.....	20	50	65	65	201,500
Tamarack, c., Mich	600	600	400	400	300	360	440	600	6,270,000
Temonj, g., Colo	.....	.....	.....	10	.....	.....	.....	.....	10.000
Tomooy, g., Colo	.....	.....	.....	300	300	.....	50	152	812,000
Trinity River, g., Colo	.....	15	.....	.....	.....	.....	.....	.....	15,000
Union, g., Colo	.....	.....	27	18	28	.....	.....	.....	82,744
United Verde, c.s., Ariz	20	435	.....	.....	.....	.....	.....	300	862.500
United, z.l., Mo	.....	.....	.....	.....	.....	.....	.....	40	40,000
Utah, Utah	15	5	.....	17	22	4	.....	2	179.000
Victor, g., Colo	.....	120	105	240	240	90	350	.....	1,155,000
Victor, L. & M., Colo	.....	.....	.....	24	12	.....	.....	.....	42,000
Vindicator, g., Colo	.....	.....	.....	.....	.....	.....	138	178	304,500
War Eagle, B. C	.....	.....	.....	.....	.....	.....	177	315	492.750
Weatherly-Bonanza, Wash	.....	.....	.....	.....	.....	.....	.....	2	1,562
What Cheer, z., Mo	.....	.....	.....	.....	.....	.....	.....	4	4,500
Wolverine, c., Mich. (*q*)	.....	.....	.....	.....	.....	.....	60	210	270,000
Yellow Aster, g., Cal	.....	.....	.....	.....	.....	40	129	155	323,780
Yosemite, s., Utah	.....	.....	.....	.....	.....	5	.....	.....	350,000

c., copper; g., gold; i., iron; l., lead; m., mica; p., phosphate rock; q., quicksilver; s., silver; z., zinc.

(*a*) Formerly the Alaska Mining and Milling Company, reorganized in 1891 as the Alaska-Treadwell Gold Mining Company; the dividends credited for 1891 and 1892 are the payments of the latter company. The Alaska Mining and Milling Company paid $700,000 previously.

(*b*) Consolidation of Cœur d'Alene and Tiger formed in 1895. The Cœur d'Alene Silver Lead Mining Company paid $375,000 in dividends; and the Tiger about $250,000.

(*d*) Formerly the Young America South Mining Company, reorganized as the Coptis in 1891.
(*e*) Previous to consolidation the Deadwood paid $275,000 and the Terra $75,000.
(*g*) Mollie Gibson levied assessments to 1892 amounting to $20,000.
(*h*) Jay Hawk and Lone Pine Consolidated Mining Company, Limited.
(*i*) Pennsylvania Con. levied assessments to 1892 amounting to $50,051.
(*j*) Maid of Erin Silver Mines, Limited, formerly Henriette & Maid Consolidated Mining Company. The dividends for 1887, 1888, 1889 and 1890 were paid by the old company, and those for 1891 and 1892 by the new company.
(*k*) Enterprise Mining Company, of Rico, Colo.; there is also an Enterprise Mining Company in Aspen and one in Leadville.
(*l*) Including dividends paid on preferred stock and common stock.
(*m*) Previous to consolidation in 1896 the Smuggler Mining Company had paid $1,140,000 in dividends.
(*n*) Bodie, Bulwer, and Mono transferred to Standard Con., January, 1897. Previous to consolidation Bodie paid $1,677,572, Bulwer paid $100,000, and Mono $12,500, while Standard Con. levied assessments to 1890 amounting to $99,887.
(*p*) Silver King levied assessments to 1897 amounting to $3,000.
(*q*) Wolverine levied assessments to 1895 amounting to $180,000.
(*s*) Lillie transferred to an English company in October, 1898; old company paid $134,110 in dividends to September 1, 1898.

ASSESSMENTS LEVIED BY MINING COMPANIES.

Name and Location of Company.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Levied.
Abbie, Cal.	.....	.....	.....	.....	.....	$3,500	.....	$3,500
Acorn, Utah.	.....	.....	.....	.....	.....	.....	$250	250
Ada Con., Utah.	$330	.....	$3,000	.....	.....	.....	.....	3,333
Ætna, Utah	.....	.....	.....	.....	.....	2,500	500	3,000
Alamo, Utah.	.....	.....	.....	.....	$625	313	.....	937
Alaska, Utah.	.....	.....	.....	.....	.....	.....	50,000	50,007
Alexandria, S. Dak.	.....	.....	.....	.....	.....	.....	500	500
Alhambra, Nev.	.....	.....	.....	.....	.....	100,000	.....	100,000
Alliance, Utah.	.....	.....	50,000	.....	.....	.....	15,000	215,000
Allouez, Mich.	.....	$16,000	.....	.....	.....	80,000	.....	1,520,980
Alpha, Nev.	10,000	6,000	15,750	$15,750	15,750	8,400	3,150	319,200
Alta, Nev.	25,200	50,400	20,160	32,400	25,200	16,200	16,200	3,681,110
Alta, Utah.	.....	.....	.....	.....	.....	.....	938	938
American Quartz, Cal.	.....	.....	.....	.....	1,000	.....	.....	1,000
Anchor, Utah.	90,000	.....	.....	.....	.....	.....	.....	560,000
Andes, Nev.	.....	25,000	15,000	15,000	20,000	10,000	15,000	1,210,000
Arnold, Mich.	.....	.....	.....	.....	.....	180,000	.....	180,000
Arrastraville, Cal.	.....	.....	.....	.....	.....	10,000	20,000	30,000
Badger Hill & Cherokee, Cal.	.....	.....	.....	.....	.....	.....	60,000	60,000
Baliol, Cal.	.....	.....	.....	.....	25,000	30,000	.....	55,000
Banner, Idaho.	.....	.....	.....	.....	10,000	.....	.....	10,000
Beaver Creek, Mont.	.....	.....	.....	.....	4,000	.....	.....	4,000
Belcher, Nev.	.....	50,200	50,200	50,200	57,200	20,800	36,400	3,551,200
Bellefontaine, Cal.	.....	.....	.....	.....	.....	10,000	10,000	$20,000
Belle Isle, Nev.	20,000	.....	.....	10,00	.....	.....	.....	240,271
Ben Butler, Utah.	.....	.....	.....	0	.....	.....	2,000	2,000
Best & Belcher, Nev.	75,600	50,400	50,400	25,20	75,600	30,240	40,320	2,631,408
Bingham Placer, Utah.	.....	.....	.....	.....	.....	.....	27,500	27,500
Blue Bird, Utah.	.....	.....	.....	0	.....	.....	500	500
Bogan Silver, Utah.	.....	.....	.....	.....	8,125	.....	.....	26,875
Boston & Cripple Creek, g., Colo.	.....	.....	.....	.....	.....	20	.....	20,000
Boulder, g., Cal.	.....	.....	.....	.....	.....	10,000	3,000	13,000
Brunswick Con., Cal.	20,000	10,000	20,000	15,000	300,000	.....	15,00	175,000
Buchanan, Cal.	.....	.....	.....	.....	.....	25,000	.....	25,000
Buckeye, Utah.	.....	.....	.....	3,750	5,000	6,250	.....	15,000
Buffalo Hump Dev., Wash.	.....	.....	.....	.....	.....	.....	200	200
Bullion, Nev.	25,000	30,000	40,000	20,000	20,000	15,000	10,000	3,130,000
Bunker Hill, Utah.	.....	.....	.....	.....	.....	.....	1,500	1,500
Burlington, Cal.	.....	.....	.....	3,000	.....	.....	.....	3,000
Butte & Boston, Mont.	.....	.....	1,500	1,500	.....	.....	.....	3,000
Butte & Phila., Mont.	.....	.....	.....	.....	1,000	.....	.....	1,000
Caledonia, s., Nev.	.....	50,000	.....	5,000	.....	.....	.....	3,240,000
California Borax, Cal.	.....	.....	.....	.....	.....	.....	4,000	4,000
California, Cal.	.....	.....	4,500	.....	1,000	.....	2,000	19,500
Cedar Creek, Cal.	.....	.....	.....	.....	.....	2,000	.....	2,000
Centennial, Mich.	.....	.....	.....	.....	120,000	300,000	.....	460,000
Central Eureka, Cal.	.....	.....	.....	.....	20,000	22,000	34,000	135,942
Central, g.s., Cal.	.....	.....	.....	.....	2,250	.....	.....	7,500
Central North Star, Cal.	10,000	.....	.....	.....	.....	.....	.....	10,000
Challenge Con., Nev.	15,000	50,000	5,000	7,500	5,000	10,000	12,500	443,000
Channel Bend, Cal.	.....	.....	.....	.....	.....	2,000	.....	2,000
Chicago & Mercur, Utah.	.....	.....	.....	.....	.....	.....	1,000	1,000
Chollar, Nev.	61,600	56,000	56,000	28,000	44,800	50,400	28,000	2,075,600
Christmas, Utah.	.....	.....	.....	.....	.....	.....	1,500	1,500
Church, g., Cal.	.....	.....	.....	.....	.....	3,000	.....	3,000
Cinnabar King, Cal.	.....	.....	.....	.....	.....	1,000	.....	1,000
Cleveland, g., Utah.	.....	.....	.....	.....	.....	2,500	.....	2,500
Columbia, Utah.	.....	.....	.....	.....	.....	750	.....	750
Comstock, Utah.	.....	.....	.....	25,000	.....	.....	.....	25,000
Confidence, Nev.	.....	6,240	14,976	7,488	14,976	13,728	8,736	548,832
Con. St. Gothard, Cal.	.....	.....	.....	.....	.....	35,000	15,000	50,000
Con. Cal. & Va., Nev. (*a*)	6,220	108,000	54,000	118,800	216,000	108,000	108,000	786,000
Con. Imperial, Nev.	.....	.....	5,000	500	10,000	10,000	10,000	2,256,000

ASSESSMENTS LEVIED BY MINING COMPANIES.—*Continued.*

Name and Location of Company.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Levied.
Con. New York, Nev.	$25,000	$5,000	$10,000	$5,000	$60,000	$30,000	$3,000	$136,500
Crown Point, Nev.	80,000	65,000	25,000	70,000	.....	25,000	20,000	2,990,000
Dalton, Utah	3,750	5,000	5,000	5,000	2,500	7,500	5,000	58,750
Derbec Blue Gravel, Cal.	5,000	.....	.....	.....	.....	.....	.....	15,000
Dexter, Nev. (*b*)	.....	.....	.....	.....	.....	30,000	.....	38,000
Diamond Con., Utah	.....	.....	.....	.....	.....	.....	8,000	8,000
Eagle, Cal.	.....	.....	.....	.....	5,000	.....	.....	5,000
Eagle, Oregon	.....	.....	.....	.....	3,000	3,000	.....	6,000
East Best & Belcher, Nev.	.....	20,000	.....	.....	.....	.....	.....	85,000
East Golden Gate, Utah	.....	.....	.....	.....	3,000	.....	.....	3,000
East Sierra, Nev.	.....	5,000	.....	.....	.....	.....	.....	25,000
Ely, Utah	.....	.....	.....	.....	.....	.....	2,000	2,000
Emerald, g., Utah	.....	.....	.....	.....	.....	3,000	.....	3,000
Eureka Con. Drift, Cal.	.....	.....	.....	.....	75,000	5,000	15,000	185,500
Eureka Con., Nev. (*c*)	.....	.....	12,500	.....	.....	10,000	5,000	590,000
Eureka Con., Utah	.....	.....	.....	5,000	.....	.....	.....	5,000
Excelsior, Cal.	.....	.....	.....	.....	.....	667	5,000	5,667
Exchequer, Nev.	15,000	10,000	5,000	5,000	5,000	.....	.....	730,000
Fall River, Cal.	.....	.....	.....	.....	.....	.....	2,500	7,500
Far West, g.s., S. Dak.	.....	.....	.....	.....	750	.....	.....	42,125
Florence, Utah	.....	.....	.....	.....	.....	.....	1,000	1,000
Ford, Cal.	.....	.....	.....	.....	.....	.....	5,000	5,000
Forlorn Hope, Cal.	.....	.....	.....	.....	.....	20,000	.....	25,000
Four Aces, Utah	.....	.....	.....	.....	2,500	2,500	2,500	7,500
Frisco, Utah	.....	.....	.....	.....	.....	.....	1,000	1,000
Galata, Cal.	.....	.....	.....	.....	.....	10,000	.....	10,000
Galena, g., Utah	.....	.....	.....	.....	.....	10,000	.....	10,000
Galena Treasure, S. Dak.	.....	.....	.....	.....	.....	.....	200	200
Genevieve, Utah	.....	.....	.....	.....	.....	.....	5,000	5,000
Gerrymander, Cal.	.....	.....	.....	.....	.....	.....	10,000	10,000
Geyser, s. l., Colo.	.....	.....	.....	.....	.....	50,000	200,000	1,275,000
Gold Belt, Utah.	1,345	.....	.....	1,667	.....	.....	.....	3,012
Gold Coin, Gilpin Co., Colo.	.....	.....	.....	.....	.....	.....	10,000	10,000
Golden Chan. Gravel Placer, Cal.	.....	.....	.....	.....	.....	.....	3,000	3,000
Golden Eagle, Nev.	.....	.....	.....	.....	.....	.....	4,000	4,000
Golden Fleece, Cal.	.....	.....	.....	.....	260	.....	.....	56,260
Golden King, Cal.	.....	.....	.....	.....	.....	3,500	.....	3,500
Golden Star, Cal.	.....	.....	.....	.....	.....	.....	2,000	2,000
Gold Flat, Cal.	20,000	.....	.....	.....	.....	.....	.....	13,000
Gold Leaf, Wash.	.....	.....	.....	.....	.....	.....	100	100
Gold Ridge, Oregon	.....	.....	.....	.....	4,000	.....	.....	4,000
Gold & Silver Carb., Utah	.....	.....	.....	.....	.....	.....	2,500	2,500
Good Title, Cal.	.....	.....	.....	.....	.....	20,000	.....	20,000
Goodyear, Mont.	4,188	.....	.....	.....	.....	.....	.....	17,188
Gould & Curry, Nev.	80,200	32,400	48,600	48,600	59,400	37,800	49,400	4,678,450
Granite Hill, Cal.	.....	.....	.....	.....	.....	.....	3,000	194,000
Grape Vine Canyon, Cal.	.....	.....	.....	.....	.....	.....	10,000	10,000
Gray Eagle, Cal.	.....	.....	.....	.....	10,000	.....	.....	1,0000
Great Eastern, Utah	.....	.....	.....	.....	.....	1,500	1,500	6,500
Great Western, q., Cal. (*d*)	.....	.....	.....	.....	.....	5,000	20,000	75,520
Grizzly, Cal.	.....	.....	.....	.....	.....	1,500	.....	1,500
Hale & Norcross, Nev. (*e*)	.....	.....	.....	.....	.....	.....	39,200	39,200
Hartery Con., Cal.	.....	2,000	2,000	2,000	.....	.....	.....	33,000
Head Center Con., Ariz.	.....	.....	.....	.....	.....	.....	25,125	25,125
Hidden Treasure, Cal.	1,000	.....	.....	.....	.....	.....	.....	1,000
Hilda, Cal.	.....	.....	.....	.....	.....	2,000	.....	2,000
Hillside, Utah	.....	.....	.....	.....	.....	.....	1,500	1,500
Himalaya, Utah	.....	.....	.....	.....	.....	.....	5,000	15,000
Home, Cal.	.....	.....	.....	.....	.....	3,000	.....	3,000
Homestake, Utah	.....	.....	.....	.....	4,000	4,000	4,000	12,000
Horn Silver Tunnel, Utah	.....	.....	.....	.....	.....	.....	500	500
Horsefly, Cal.	.....	.....	.....	.....	.....	8,000	.....	8,000
Horseshoe Bar Con., Cal.	.....	.....	.....	.....	24,000	6,000	3,000	85,800
Humboldt, Cal.	.....	.....	.....	.....	.....	.....	2,000	2,000
International, Utah	.....	.....	.....	.....	.....	.....	1,500	1,500
Jack Rabbit, Cal.	18,000	5,000	.....	.....	.....	.....	.....	118,000
Jackson, Nev.	10,000	.....	.....	.....	.....	.....	.....	247,500
Jennie Lind, Cal.	.....	.....	.....	.....	.....	.....	1,000	1,000
Joe Bowers Ext., Utah	.....	.....	.....	.....	.....	4,000	13,500	17,500
Julia Con., Nev.	5,000	.....	5,500	5,500	5,500	.....	3,300	1,498,500
Jumbo, Utah	.....	.....	.....	.....	.....	500	.....	500
Junction, Cal.	.....	.....	.....	.....	.....	10,500	6,000	16,500
Jupiter Gravel, Cal.	.....	.....	.....	.....	40,000	40,000	.....	80,000
Justice, Nev.	30,000	75,000	20,000	15,750	.....	10,500	21,000	3,667,750
Karan, Cal.	.....	.....	.....	.....	.....	2,500	.....	2,500
Kate Hayes, Cal.	.....	.....	.....	.....	.....	10,000	10,000	70,000
Kentuck, g., Utah	.....	.....	.....	.....	.....	30,000	.....	30,000
Kentuck Con., Nev.	10,500	26,250	10,500	5,250	10,500	5,250	.....	125,300
Lady Washington, Nev.	.....	.....	.....	5,400	.....	.....	5,400	163,200
La Grange, Cal.	.....	.....	.....	.....	.....	.....	15,000	15,000
Laird, Cal.	.....	.....	.....	.....	.....	10,000	.....	10,000
Larkin, Cal.	.....	.....	.....	.....	.....	2,000	2,000	4,000
La Suerte, Cal.	.....	.....	.....	.....	.....	7,500	7,500	15,000
Leo, Mont.	.....	.....	.....	10,000	.....	.....	.....	52,500

ASSESSMENTS LEVIED BY MINING COMPANIES.—*Continued.*

Name and Location of Company.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Levied.
Leon, Cal.	.....	.....	.....	.....	.....	1,500	.....	$1.500
Little Chief, Utah.	.....	.....	.....	.....	.....	.....	$2,000	2.000
Little Pittsburg, Utah.	.....	.....	$4,000	$6,000	$4,000	$4,000	5,000	23,000
Live Oak & Minuett, Cal.	.....	.....	.....	.....	.....	20,000	10,000	30.000
Live Yankee, Cal.	.....	.....	.....	.....	.....	5.000	2,000	$7,050
Lone Hill, Cal.	.....	.....	.....	.....	1,125	.....	.....	1,120
Lower Mammoth, Utah.	.....	.....	.....	.....	.....	12,500	15,000	27,500
Lucky Bill, Utah.	.....	.....	.....	7,200	6,600	1,800	2,500	58,900
Luiah Con., Utah.	.....	.....	.....	.....	.....	.....	1,500	1,500
Madge, S. Dak.	.....	.....	.....	.....	.....	.....	200	200
Marguerite, Cal.	.....	.....	.....	.....	20,000	15,000	15,000	85,000
Marina Marsicano, Cal.	.....	.....	.....	.....	16,360	19,000	19,000	54,360
Martha Washington, Utah	.....	.....	.....	.....	.....	1,000	2,000	3,000
Martin White, Nev.	$25,000	$25,000	.....	.....	.....	.....	120.000	1,200,000
Maxfield, Utah.	.....	.....	.....	.....	.....	6,000	9,000	15,000
Mayday, Cal.	.....	.....	.....	.....	.....	5.000	10,000	15.000
Mayday, Utah.	.....	.....	.....	.....	.....	2,000	2,000	4,000
Mayflower Gravel, Cal. (*f*)	.....	.....	.....	.....	.....	6,000	.....	6,000
Merrimac, Cal.	.....	.....	.....	.....	.....	20,000	.....	20,000
Meteor, Utah.	.....	.....	.....	.....	.....	3,734	7,000	10.734
Mexican, Nev.	76,500	75,600	75,600	40,320	20,000	40,320	25,200	2,283,920
Midnight Bowers, Utah.	.....	.....	.....	.....	.....	.....	1,000	1,000
Mineral Hill, Cal.	.....	.....	.....	.....	2,500	.....	.....	2,500
Molly Bawn, Utah.	.....	.....	.....	.....	.....	.....	1,000	1.000
Montecito, Cal.	.....	.....	.....	.....	.....	10,000	.....	10,000
Montreal, Utah.	.....	.....	375	.....	375	375	25,000	30,625
Morgan, Cal.	.....	.....	.....	.....	.....	10,000	.....	10,000
Mount Diablo, Nev. (*g*).	.....	.....	.....	7,500	.....	.....	.....	145,000
Mountaineer, Cal.	.....	.....	.....	.....	.....	15,000	.....	15,000
Murray Hill, Utah.	.....	.....	.....	.....	.....	250	250	500
Nancy Hanks, Cal.	.....	.....	.....	.....	.....	5,000	.....	5,000
National Con., Cal.	.....	.....	.....	.....	.....	15,000	45,000	60,000
Nashville, g., Cal.	.....	.....	.....	.....	.....	2,000	.....	2,000
Navajo, Nev.	20,000	.....	.....	.....	.....	.....	.....	555,521
Nevada Queen, Nev.	25,000	.....	5,000	.....	.....	.....	.....	270,000
New Erie, Utah.	.....	.....	.....	.....	.....	.....	2,000	2,000
New Imperial, Utah.	.....	.....	.....	.....	.....	.....	1,000	1,000
New Klondike, Utah.	.....	.....	.....	.....	.....	.....	2,500	2,500
North Banner, Cal.	.....	.....	.....	10,000	.....	.....	.....	21.794
North Belle Isle, Nev. (*h*).	38,075	.....	.....	10,000	.....	.....	.....	523,075
North Bloomfield, Cal.	.....	.....	.....	.....	.....	25,000	.....	25,000
North Commonwealth, Nev. (*i*).	10,000	.....	.....	.....	.....	.....	.....	120.000
North Eureka, Utah.	.....	.....	.....	1,500	.....	.....	.....	1,500
North Gould & Curry, Nev.	10,000	20,000	.....	10,000	.....	20,000	.....	375,000
North Mercur, Utah.	.....	.....	.....	.....	.....	10,000	.....	10.000
Northern Light, g., Utah.	.....	.....	.....	.....	.....	80,000	.....	80,000
Northern Spy, Utah.	.....	.....	.....	.....	.....	10,000	10,000	20,000
Occidental Con., Nev.	55,000	30,000	30,000	65,000	30,000	20,000	20,000	519,179
Old Bonanza, Cal.	.....	.....	.....	.....	.....	.....	2,500	2,500
Old Home, Cal.	.....	.....	.....	.....	.....	5,000	3,500	8,500
Omaha Con., Cal.	.....	.....	.....	.....	.....	10,000	25,000	35,000
Opohonga, Utah.	.....	.....	.....	.....	.....	1,500	1,000	2,500
Ophir, Nev.	100,000	100,000	100,000	25,200	25,200	43,200	50,400	4,652,968
Orient, Cal.	.....	.....	.....	.....	.....	.....	50,000	50,000
Orleans, Cal.	.....	.....	.....	.....	.....	15,000	12,000	17,000
Oro Quartz, Cal.	.....	.....	.....	.....	.....	.....	4,500	4,500
Osceola Con., Cal.	.....	.....	.....	.....	1,000	1,000	1.000	11,810
Overman, Nev. (*j*).	60,000	34,560	23,040	34,560	17,280	17,280	11,520	4,141.110
Palo Alto, Wash.	.....	.....	.....	.....	.....	100	200	300
Peer, Ariz.	10,000	5,000	.....	.....	.....	.....	.....	215,000
Peerless, Nev.	5,000	5,000	.....	.....	.....	.....	.....	410.000
Phœnix, Utah.	.....	.....	.....	.....	1,000	.....	.....	1,000
Pine Hill, Cal.	3,000	3,000	5,000	5,000	10,000	.....	.....	30,000
Potosi, Nev.	84,000	112,000	56,000	28,000	50,400	50,400	39,200	2,196,400
Powning, Cal.	.....	.....	.....	.....	.....	.....	2,500	2,500
Rainbow, S. Dak.	.....	.....	438	425	100	.....	.....	5,213
Red Cap, Cal.	.....	.....	.....	.....	.....	90,000	.....	90.000
Reddick, Cal.	.....	.....	.....	.....	.....	5,000	5,000	10,000
Red Jacket, Nev.	.....	.....	.....	.....	.....	.....	2,000	2,000
R. G. W., Utah.	.....	.....	.....	.....	.....	.....	2,000	2,000
Rescue Gold, Nev.	.....	.....	.....	.....	3,000	2,000	500	5,500
Reservation, Wash.	.....	.....	.....	.....	.....	.....	100	700
Revenue, Utah.	.....	.....	.....	.....	.....	.....	4.500	4.500
Reward, Cal.	.....	.....	.....	.....	7,040	3.840	3,200	67,080
Richmond, Cal.	.....	.....	.....	.....	.....	10.000	.....	19.000
Ridge Copper, Mich.	.....	.....	.....	.....	50,000	.....	.....	50.000
Rockland, Cal.	.....	.....	.....	.....	.....	3,000	.....	3,000
Ropes, Mich.	20,000	.....	.....	.....	.....	.....	.....	20,000
Rose Creek, Cal.	.....	.....	.....	.....	.....	5,000	5,000	10,000
Sacramento, Utah.	.....	.....	.....	.....	.....	.....	600	600
Sailor Con., Cal.	.....	.....	.....	.....	.....	.....	11,000	11,000
Salmon River., Nev.	.....	.....	.....	.....	.....	.....	1,425	11,970
St. Mary's Copper, Mich.	2,000	.....	2,000	.....	.....	.....	.....	4,000
Salt Lake & Nevada, Utah	.....	.....	.....	.....	.....	1,000	.....	1.000

ASSESSMENTS LEVIED BY MINING COMPANIES.—*Continued.*

Name and Location of Company.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	Total Levied.
Savage, Nev.	112,000	100,805	67,200	67,200	67,200	44,800	22,400	7,309,800
Scorpion, Nev.	.....	5,000	.....	.....	5,000	.....	.....	420,000
Seg. Belch. & Mides, Nev.	35,000	20,000	20,000	10,000	10,000	8,000	10,000	378,000
Sevier, Utah	.....	.....	.....	.....	22,500	.....	.....	50,000
Sheep Rock, Utah	.....	.....	.....	.....	5,250	.....	.....	5,250
Shower Con., Utah	.....	.....	.....	.....	.....	.....	8,000	8,000
Sierra Nevada, Nev.	45,000	50,000	50,000	50,000	400,000	20,000	50,000	6,756,910
Silver Bell, Mont.	.....	.....	.....	.....	.....	5,000	.....	5,000
Silver Cloud, Utah	.....	.....	.....	.....	.....	.....	5,000	5,000
Silver Hill, Nev.	5,400	5,400	.....	.....	5,400	5,400	.....	2,220,200
Silver King, Ariz.	.....	100,000	.....	75,000	50,000	50,000	50,000	490,000
Silver State, Utah	.....	.....	.....	.....	1,000	.....	.....	1,000
Siskiyou Con., Cal.	9,000	14,000	4,000	2,000	.....	3,000	.....	3,000
Snowflake, Utah	.....	.....	.....	.....	.....	2,000	5,000	51,000
Soulsby Con., Utah	.....	.....	.....	.....	18,125	.....	.....	17,500
South Bingham, Utah	.....	.....	.....	.....	.....	.....	2,500	2,500
South Eureka, Cal.	.....	.....	.....	.....	.....	.....	6,000	6,000
South Fork Con., Utah	.....	.....	.....	.....	.....	5,000	.....	5,000
Spanish Bar, Cal.	.....	.....	.....	.....	.....	2,000	5,000	7,000
Star, Utah	.....	.....	.....	.....	.....	1,000	6,000	7,000
Sterling, Cal.	.....	.....	.....	.....	.....	15,000	.....	15,000
Success, Utah	.....	.....	.....	.....	.....	.....	9,750	9,750
Sunbeam Con., Utah	.....	.....	.....	.....	11,250	5,000	50,000	73,125
Tanama, Cal.	.....	.....	.....	.....	.....	.....	10,000	10,000
Tetro, Utah	.....	.....	.....	.....	.....	18,000	15,000	33,000
Texas, Cal.	.....	.....	.....	.....	.....	.....	15,000	15,000
Thorpe, Cal.	.....	.....	.....	.....	.....	5,000	5,000	10,000
Tintic, Utah	.....	.....	.....	.....	.....	.....	4,375	4,375
Tracy, Cal.	.....	.....	.....	.....	.....	.....	5,000	5,000
Trinity, Cal.	.....	.....	.....	.....	.....	1,000	.....	1,000
Triumph, Utah	.....	.....	.....	.....	.....	.....	1,000	1,000
Troy, Alaska	.....	.....	.....	.....	2,000	6,250	6,250	16,750
Tuscarora Chief, Utah	.....	.....	.....	.....	.....	.....	2,000	2,000
Union Con., Nev.	45,000	35,000	20,000	40,000	20,000	30,000	18,000	2,660,000
Utah Con., Nev.	25,000	5,000	5,000	10,000	30,000	20,000	15,000	495,000
Valeo, Utah	.....	.....	.....	.....	.....	.....	10,000	10,000
Vallejo, q., Cal.	.....	.....	.....	.....	.....	2,000	2,000	4,000
Victory, S. Dak.	.....	.....	.....	375	.....	.....	.....	2,625
Waterloo, Cal.	30,000	.....	.....	.....	.....	.....	.....	30,000
Watt Blue Gravel, Cal.	.....	.....	.....	.....	.....	.....	3,000	58,000
Weldon, Ariz.	10,000	.....	.....	.....	.....	.....	.....	65,000
Willow Creek, S. Dak.	.....	.....	.....	.....	.....	.....	1,100	1,100
Yankee Boy, S. Dak.	.....	.....	.....	.....	.....	.....	500	1,500
Yankee Con., Utah	.....	.....	.....	.....	.....	.....	1,000	1,000
Yankee Girl, Utah	.....	.....	.....	.....	.....	.....	5,000	5,000
Yellow Jacket, Nev.	90,000	90,000	90,000	90,000	.....	24,000	36,000	5,826,000
Yuba River, Cal.	.....	.....	.....	.....	.....	2,500	.....	2,500
Zuba, Cal.	.....	.....	.....	.....	.....	5,000	.....	5,000

NOTE—All the Comstock companies reduced their capitalization between September and December, 1898.

(*a*) Previous to the consolidation in August, 1884, the California had paid $31,320,000 in dividends and the Consolidated Virginia $42,390,000. Since then, and to February, 1895, the consolidated company paid $2,894,000 in dividends.

(*b*) Dexter paid $100,000 in dividends to 1898.

(*c*) Eureka Con. paid $5,112,500 in dividends to 1892.

(*d*) Great Western paid $388,366 in dividends to 1898.

(*e*) Hale & Norcross paid $5,776,200 in dividends to 1898.

(*f*) Mayflower paid $166,824 in dividends to December, 1895.

(*g*) Mt. Diablo paid $225,000 in dividends to 1898.

(*h*) North Belle Isle paid $230,000 in dividends to 1898.

(*i*) North Commonwealth paid $25,000 in dividends to 1891.

(*j*) Overman reincorporated in September, 1898.

AUSTRALASIA.

THE most important articles of mineral production in the seven colonies of Australasia are gold, silver, lead, copper, tin, iron and coal. These industries are referred to specifically under the respective captions elsewhere in this volume. The statistics of production, imports and exports as reported in the official statistics are summarized in the following tables:

MINERAL PRODUCTION OF NEW SOUTH WALES. (*a*) (*c*) (IN METRIC TONS AND DOLLARS; £1=$5.)

Year.	Alunite.		Antimony and Ore.		Bismuth.		Chrome Ore		Clay. (Fire.)		Coal.		Coke.		Cobalt Ore.	
1894....	876	$17,240	1,270	$98,720	...	.....	3,084	$61,800	24	$300	3,730,329	$5,777,865	35,009	$166,045	3	$50
1895....	845	16,640	486	36,255	...	.....	4,297	65,240	20	275	3,798,325	5,476,635	28,072	123,415	6	130
1896....	1,393	20,580	135	9,170	41	$2,450	3,913	56,400	35	345	3,972,068	5,627,405	26,778	109,255	...	.....
1897....	736	10,960	172	18,060	3	4,000	3,433	51,345	....	.....	4,453,729	6,150,205	65,229	226,960	...	.....
1898....	2,988	44,115	84	4,580	30	23,075	2,145	31,505	15	160	4,781,551	6,359,163	83,538	294,674	119	2,800

Year.	Copper Ingot.		Copper Ore and Regulus.		Gold—Kg.		Iron. (*b*)		Iron Oxide		Lead, Argentiferous.		
											Ore.	Metal	Value.
1894....	1,582	$305,170	590	$62,235	10,102·2	$5,783,585	2,406	$95,850	440	$3,350	140,018	43,225	$10,976,695
1895....	2,838	596,500	1,075	107,225	11,202·6	6,579,646	2,442	78,100	155	1,740	123,266	30,162	7,805,050
1896....	4,524	1,001,180	15	375	9,221·2	5,366,800	4,798	166,415	381	4,005	271,640	19,886	8,794,665
1897....	6,864	1,499,145	169	4,355	9,088·9	5,442,065	3,391	109,310	234	2,680	275,249	18,395	8,407,640
1898....	5,744	1,400,240	181	4,195	10,590·4	6,223,649	5,283	211,250	398	4,160	394,676	10,270	8,223,895

Year.	Lead, Pig.		Man. Ore.		Opal—Kg.		Platinum. Kg.		Shale, Kerosene.		Silver—Kg.		Stone.	
													Limestone Flux	
1894..	31	$1,300	14	$230	91·2	$28,420	.....	.....	21,510	$158,905	26,339	$470,750	91,420	$346,445
1895..	20	985	3	50	153·3	30,000	.....	.....	60,377	376,095	17,112	409,290	105,861	340,800
1896..	24	1,395	..	.....	630·4	125,000	75·8	$17,395	32,348	171,010	6,307	132,590	90,946	271,305
1897..	32	1,990	..	....	2400·4	675,000	61·2	14,745	34,635	209,060	4,666	88,555	68,671	206,990
1898..	1,745	96,410	1	25	.....	400,000	38·9	10,310	30,164	159,170	16,580	296,390	9,401	28,915

Year.	Tin Ingot.		Tin Ore.		Zinc Ore.		Sundry Minerals
1894....	2,653	$897,225	198	$38,760	.....	.....	$4,460
1895....	2,235	680,400	78	12,715	.....	.....	23,185
1896....	1,737	496,060	98	14,595	.....	.....	4,620
1897....	1,159	350,640	14	2,800	29,303	$118,440	87,160
1898....	908	302,825	1	175	39,564	144,705	10,105

(*a*) From the Annual Report of the Department of Mines and Agriculture, New South Wales, 1897. (*b*) Manufactured from old iron. (*c*) In addition to the above there is a small output of diamonds, which amounted to 56.359 cts. ($156,965) for the ten years preceding 1898.

MINERAL IMPORTS OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Cement—Barrels.		Chrome Ore.		Coal.		Coke.		Copper. In Matte and Regulus.		Copper. Ingot.	
1894....	145,662	$361,115	895	$16,885	379	$1,295	48,347	$183,420	187	$29,225	253	$71,040
1895....	162,621	380,760	3,840	66,705	378	1,870	43,206	184,160	94	19,155	246	50,480
1896....	126,242	289,025	5,537	109,220	807	4,870	43,824	675,535	93	19,880	5	1,065
1897....	184,831	403,210	2,731	49,240	1,755	6,065	33,970	511,980	42	8,680	6	1,585
1898....	183,851	418,580	4,780	93,770	666	2,985	4,064	61,735	.....	.....	3	1,000

Year.	Copper—*Continued.* Ore.		Copper—*Continued.* Rod, Sheet, and Wire.	Gold. Coin.	Gold. Bullion.	Iron. Pig.		Iron. Ore.		Iron and Steel. Certain Manufactures of. (*c*)	Iron and Steel. All Other.
1894....	669	$32,535	$16,070	$1,498,790	$13,133,480	3,744	$59,880	1,409	$7,275	50,987	$2,654,950	$162,660
1895....	379	20,630	11,400	279,225	10,051,195	6,568	98,055	1,522	6,945	55,004	2,642,300	142,085
1896....	315	12,875	27,895	2,407,100	9,943,070	4,195	124,255	108	555	67,978	3,626,075	.....
1897....	1,401	92,440	96,445	5,749,095	10,208,230	16,256	269,470	1,293	8,095	71,469	3,904,105	68,495
1898....	1,013	40,245	61,635	18,323,980	9,566,335	12,081	189,885	2,638	8,525	65,140	3,621,285	136,640

Year.	Lead. Pig.		Lead. Pipe, Sheet and Old.		Nickel Ore.		Quicksilver. (*No. of Flasks.*)		Salt. Rock.		Salt. In Bags.	
1894....	7,016	$334,545	53	$4,445	1,482	$39,690	1,078	$3...535	5,900	$35,245	26,099	$230.04
1895....	4,775	224,330	18	1,955	317	9,865	797	29,595	10,511	70,515	26,956	223.81
1896....	7,440	362,720	988	19,720	3	120	261	9,250	7,375	39,825	25,784	207.56
1897....	5,281	289,590	808	20,235	208	5,000	470	12,255	4,654	34,110	36,722	261.22
1898....	2,879	135,330	49	3,685	788	7,575	469	17,005	8,647	59,290	30,991	217.11

Year.	Saltpeter.		Silver. Ingot—Kg.		Silver. Ore.		Silver. In Matte—Kg.		Silver. Coin.	Slate. Roofing—*Number*		Slate. Slabs. *Number.*	
1894....	131	$12,480	415	$8,350	2,958	$270,140	(*b*)	.....	$50,000	1,077,501	$41,555	2,755	$5,535
1895....	188	15,410	95	1,945	922	63,275	508	$10,110	109,470	954,449	30,695	2,150	7,405
1896....	162	19,905	180	3,900	15,078	672,770	3,958	79,245	49,000	2,219,681	90,595	3,602	11,675
1897....	176	18,455	288	5,060	1,288	38,680	1,928	35,500	131,680	1,908,475	78,990	2,605	8,575
1898....	181	17,170	267	4,980	964	37,625	.....	.....	286,010	1,893,871	90,240	3,482	12,430

Year.	Soda Ash.		Soda. Bicarbonate.		Soda. Caustic.		Soda. Crystals.		Stone, All Kinds, Excepting Marble.	Sulphur.	
1894....	1,057	$30,120	488	$17,070	1,598	$96,550	1,577	$28,445	$11,425	2,067	$43,340
1895....	1,068	27,370	636	24,530	850	46,995	1,164	24,895	69,740	1,728	32,80
1896....	1,749	60,300	611	25,080	1,010	48,465	745	18,305	40,925	1,330	22.33
1897....	1,452	50,583	788	31,395	1,213	66,900	495	7,940	27,065	2,287	33.96
1898....	1,469	38,905	953	36,495	1,241	63,225	468	13,440	30,235	1,845	26.96

Year.	Tin. Ingot.		Tin. Ore.		Tin. Plates.	Yellow Metal.	Zinc. Slabs.		Zinc. Sheet and Manufactures.	
1894....	1,097	$379,760	2,208	$513,480	$299,665	$63,920	688	$57,610	278	$29,060
1895....	946	272,700	1,493	270,980	372,870	80,525	693	53,880	330	28,110
1896....	910	265,505	1,073	169,875	353,345	92,895	1,416	116,115	368	35,500
1897....	693	205,135	662	104,510	200,350	69,290	1,341	122,815	561	58,010
1898....	478	161,245	554	92,665	373,660	102,250	1,220	122,395	377	43,905

(*a*) From the *New South Wales Statistical Register*. (*b*) No returns given. (*c*) The figures for "certain manufactures" of iron and steel in the years 1894, 1895 and 1897 include only those articles the weight of which is given in the official returns. The figures for 1896 include all manufactures of iron and steel. "All other" manufactures comprise the values of those articles for which no weights are given in the reports.

MINERAL EXPORTS OF NEW SOUTH WALES. (*a*) (*g*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Acid, Sulphuric.		Antimony. Ore.		Antimony. Auriferous.		Antimony. Metal.		Bismuth Ore.	Cement. *Barrels.*	Chrome Ore.		Cobalt Ore.	
1894....	140	$20,720	679	$32,240	561	$56,945	30	$4,535	2·0	$50	6,927	$16,060	3,977	$90,855	590	$15,415
1895....	247	19,480	225	9,945	209	19,850	44	6,460	0·6	210	10,279	23,275	8,151	140,535	235	7,265
1896....	100	5,535	87	2,810	25	3,160	23	3,200	45·0	3,350	33,987	74,650	9,465	145,165	284	9,750
1897....	52	3,495	49	2,510	21	1,500	102	14,050	d3·3	4,150	15,272	39,185	5,959	101,330	73	2,500
1898....	24	1,390	57	2,215	13	795	13	1,570	31·5	23,775	16,801	45,490	6,924	122,775	184	4,450

Year.	Coal.		Coke.		Copper.								Rod, Sheet, Wire.
					Ingot.		In Matte.		Regulus.		Ore.		
1894....	2,159,127	$4,023,845	11,858	$53,330	1,983	$373,195	373	$70,640	(c)	.....	301	$7,050	$3,670
1895....	2,200,890	3,869,770	8,948	42,625	2,926	619,730	194	37,710	1,096	$115,926	55	1,730	3,370
1896....	2,514,506	4,501,320	5,531	22,615	4,026	904,540	595	117,595	(c)	.....	15	375	3,000
1897....	2,739,769	4,760,270	32,221	135,225	4,196	964,880	2,778	549,440	(c)	.....	169	4,255	5,770
1898....	2,836,465	4,813,340	31,284	125,330	5,285	1,285,715	460	114,765	(c)	.....	181	4,195	12,950

Year.	Gold.			Iron and Steel.				Lead.					
	Ore.	Coin.	Bullion—Kg.	Pig.		Manufactures. (b)		Pig.		Pipe and Sheet.		Argentiferous.	
1894....	$227,315	$9,877,865	2,848	$1,663,455	400	$6,585	9,646	$587,685	3,886	$165,020	326	$21,060	43,198	$7,716,065
1895....	946,845	13,552,800	3,758	2,174,985	201	3,825	14,504	726,680	2,737	181,400	577	39,945	30,162	4,798,680
1896....	488,425	18,014,280	2,576	1,417,520	1,470	26,350	17,289	1,089,375	3,150	167,425	750	52,535	19,897	3,974,385
1897....	282,040	21,733,285	3,295	2,043,690	1,188	22,495	19,515	1,297,320	2,100	119,395	1,043	77,725	18,895	3,927,385
1898....	227,410	32,645,300	3,326	2,022,995	1,294	31,060	15,454	1,021,015	3,367	191,210	707	57,945	10,271	1,605,680

Year.	Nickel Ore.		Saltpeter.		Salt.				Silver.				
					Rock.		Other, in Bags.		Ore.		In Matte—Kg.		Coin.
1894....	2,148	$53,545	5	$770	199	$1,855	2,660	$34,795	143,200	$3,416,890	7,019	$183,575	$63,890
1895....	343	9,175	17	2,480	855	7,995	3,999	46,085	194,158	3,068,985	5,855	118,190	88,330
1896....	(c)	.....	6	790	457	3,605	2,890	32,440	284,594	5,447,925	7,550	152,960	66,450
1897....	208	4,900	42	5,730	330	3,065	4,752	58,380	275,807	5,205,525	2,454	41,860	99,545
1898....	785	9,800	37	4,365	433	3,635	5,166	60,445	396,057	6,684,955	8,794	152,655	149,360

Year.	Soda Ash.		Soda.				Stone.	Sulphur.		Tin.				
			Bicarbonate.		Caustic.					Ore.		Ingot.		Plate.
1894....	178	$6,560	50	$2,400	214	$14,675	$4,665	34	$1,305	214	$42,835	3,749	$1,276,725	$51,585
1895....	175	5,800	56	2,500	67	4,000	4,705	42	2,105	80	12,955	3,181	970,410	87,445
1896....	207	7,010	39	1,790	148	7,545	7,015	139	4,240	99	14,525	2,646	763,595	79,100
1897....	117	4,010	61	2,795	90	4,660	3,545	32	1,300	143	2,800	1,837	556,345	78,205
1898....	285	9,595	124	5,580	132	6,580	3,505	88	1,565	1	175	1,384	481,925	55,040

(a) From the *New South Wales Statistical Register*. (b) Values including those of manufactures for which no quantity is stated, as follows: In 1894, $5,955; 1895, $17,735; 1896, $17,970; 1897, $9,125; 1898, $14,445. (c) Not stated in the reports. (d) Probably bismuth metal. (g) There was also exported from N. S. W. in 1897, iron oxide: 246 metric tons, $2,890, and iron ore, 5 metric tons, $35.

MINERAL PRODUCTION OF NEW ZEALAND. (a) (b) (IN METRIC TONS; £1 = $5.)

Year.	Antimony Ore.		Coal.		Coke.		Gold—Kg.		Manganese Ore.	
1894.....	45	$3,805	731,059	$1,978,545	109	$800	6,898	$4,439,195	544	$5,780
1895.....	55	7,430	752,680	2,053,810	293	3,575	9,129	5,810,880	213	2,625
1896.....	21	2,250	805,537	2,143,240	107	1,315	8,202	5,207,140	66	1,025
1897.....	10	785	854,164	2,259,575	14	.....	7,827	4,901,020	133	2,705
1898.....	*Nil.*	.....	921,546	2,378,555	9	.....	8,714	5,408,445	220	3,515

MINERAL PRODUCTION—*Continued.*

Year.	Mixed Ores.		Silver—Kg.	
1894.....	25	$1,765	1,665	$33,485
1895.....	63	4,400	2,644	53,395
1896.....	38	6,675	2,983	52,945
1897.....	1,586	29,460	5,716	104,360
1898.....	1,857	23,960	9,140	165,535

MINERAL IMPORTS. (c) (IN METRIC TONS; £1 = $5.)

Year.	Coal.		Iron.	Machinery.	Railway Materials	Tools and Implements.
1894.....	114,768	$525,955	$1,908,210	$889,980	$213,995	$309,685
1895.....	109,929	488,105	1,739,640	770,045	234,940	299,325
1896.....	103,894	470,690	2,274,165	1,417,200	163,270	350,690
1897.....	112,681	490,695	2,359,525	2,005,265	682,855	450,905
1898.....	(d)	.....	(d)	.....	(d)	.....

(a) From *New Zealand Mines Statements*, by the Hon. A. J. Cadman, Minister of Mines, Wellington. (b) The exports are stated to be identical with the production, except in the case of coal, of which substance the exports were as follows: in 1894, 76,204 tons, value $367,190; in 1895, 87,363 tons, value $416,710; in 1896, 80,796 tons, value, $359,920; in 1897, 77,280 tons, value, $347,975; in 1898, 57,333 tons, value, $251,905. (c) From *British Statistical Abstracts*. (d) Statistics not yet available.

MINERAL PRODUCTION OF QUEENSLAND. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Antimony Ore.		Bismuth Ore.		Coal.		Copper Ore.		Gems.	Gold—Kg.		Lead.		Manganese Ore.	
1894.....	28	$1,400	66	$31,350	275,086	$572,965	422	$47,910	*b*$12,085	21,135	$11,891,442	*e*458	$21,950	142	$2,000
1895.....	(*c*)	.....	60	16,290	298,387	662,650	441	65,485	*b*29,575	19,647	11,054,426	369	18,170	361	5,515
1896.....	(*c*)	.....	(*c*)	.....	377,382	774,985	589	105,210	(*c*)	19,917	11,206,737	628	30,900	305	4,500
1897.....	(*c*)	.....	1	670	364,142	699,445	293	63,295	(*c*)	25,129	12,765,705	391	20,585	408	7,530
1898.....	(*c*)	.....	8	3,500	414,461	752,465	63	10,830	(*c*)	28,616	13,751,745	252	12,400	66	1,355

Year.	Opal.	Silver Ore.		Silver—Kg.		Stone. Building. (*b*)		Tin Ore.		Wolfram Ore	
1894.....	$60,000	188	$9,375	5,697	$110,385	42,997	$53,115	2,917	$511,385	107	$3,550
1895.....	168,750	77	10,060	6,999	150,210	52,206	58,305	2,148	340,665	25	2,710
1896.....	116,500	(*c*)	.....	8,687	160,810	(*c*)	.....	1,579	245,090	3	300
1897.....	51,250	(*c*)	.....	7,280	125,590	(*c*)	.....	1,222	187,545	18	975
1898.....	33,225	(*c*)	.....	3,285	52,925	(*c*)	.....	1,041	182,510	79	12,700

MINERAL IMPORTS OF QUEENSLAND. (*d*) (IN METRIC TONS; £1 = $5.)

Year.	Cement and Plaster of Paris—*Barrels.*		Coal.		Coke.		Glass and Glassware.	Gold Bullion—Kg.	
1893.....	27,353	$45,215	14,668	$38,030	74	$455	$74,175	15·1	$9,330
1894.....	25,715	49,085	18,166	44,980	50	565	99,675	24·0	12,925
1895.....	47,244	116,620	18,023	91,575	(*c*)	.....	147,615	56·3	33,375
1896.....	63,607	163,860	18,743	88,225	5	20	176,330	189·2	108,405
1897.....	65,230	159,375	21,624	98,370	81	505	103,060	176·9	91,490

Year.	Gold Specie.	Iron and Steel.	Petroleum—*Gallons.*		Rails, Track Material.	Silver Bullion. Kg.		Silver Specie.
1893.....	$4,342,435	$749,070	1,375,836	$172,690	$16,060	(*c*)	.....	$10,580
1894.....	1,452,980	1,020,590	1,311,129	201,890	110,805	1·5	$45	2,955
1895.....	3,001,000	1,468,785	1,688,975	282,530	139,980	7·1	210	1,750
1896.....	400,000	1,519,595	1,271,910	265,645	138,220	5·0	120	13,380
1897.....	59,350	1,586,605	1,369,178	278,070	121,950	11·1	265	113,145

MINERAL EXPORTS OF QUEENSLAND. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Antimony Ore.		Bismuth Ore.		Copper Ore.		Copper Ingot.		Copper Regulus.		Copper Matte.		Gold—Kg.		Gold Ore.	
1894....	109	$9,055	37	$23,510	64	$4,900	148	$25,400	376	$41,870	127	$18,500	20,030	$11,909,580	33	$2,160
1895....	44	1,935	44	24,760	102	10,840	315	62,695	211	51,885	225	31,200	21,529	11,360,545	199	9,985
1896....	(*c*)	.....	0·7	750	209	21,590	377	71,045	287	56,970	142	12,400	19,496	10,571,285	68	5,380
1897....	25	2,500	39	20,150	241	15,425	196	34,515	143	35,075	150	21,925	25,293	12,843,510	220	19,235
1898....	9	300	6	1,710	375	28,845	(*c*)	.....	2	1,980	5	1,325	29,390	14,278,905	748	75,790

Year.	Silver Ore.		Silver-Lead Bullion.		Silver Precipitate.		Doré Silver Bullion. Kg.		Mundic Ore.		Precious Stones	Tin Ore.		Tin.		Wolfram Ore.	
1894....	138	$9,375	1,008	$473,085	(*c*)	.....	(*c*)	.....	6	$355	$12,085	1,783	$368,745	473	$173,400	69	$6,570
1895....	77	10,060	561	196,710	1	.....	489	$26,340	(*c*)	....	29,575	1,180	224,175	459	146,760	34	2,965
1896....	181	46,350	174	103,630	0·8	$6,785	3,163	144,460	5	770	11,730	751	136,410	355	97,485	23	2,365
1897....	259	35,080	301	87,350	(*c*)	1,500	1,823	210,565	(*c*)	....	5,700	636	115,750	151	675	(*c*)	.....
1898....	207	89,370	(*c*)	.....	(*c*)	.....	1,951	206,065	(*c*)	....	7,735	535	103,455	150	55,900	2	120

(*a*) From *Annual Reports of the Under Secretary of Mines*, Queensland, when not otherwise stated. (*b*) From *Mineral Statistics of the United Kingdom.* (*c*) Not reported. (*d*) From *Statistical Abstracts for the Several Colonial and other Possessions of the United Kingdom.* (*e*) Quantity exported.

MINERAL IMPORTS OF SOUTH AUSTRALIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Coal.		Coke.		Gold and Silver Bullion and Specie.	Iron.				Silver-Lead.			
						Bar, Sheet, Hoop, and Rod.		Galvanized, Plain, and Corrugated.		Metal.		Ore.	
1894....	259,695	$768,670	119,557	$751,525	307,670	4,848	$175,620	6,236	$450,995	38,018	$6,750,430	109,386	$2,511,160
1895....	261,117	675,450	91,298	531,875	978,140	5,158	163,205	4,153	286,970	25,079	4,298,805	134,878	1,989,950
1896....	343,583	895,480	68,454	315,600	1,782,736	7,656	263,610	7,110	527,475	25,806	4,379,645	234,297	4,143,515
1897....	378,496	1,012,140	75,205	402,620	78,340	6,505	229,970	6,438	454,750	17,466	2,989,610	301,990	5,221,195
1898....	420,001	1,121,795	82,796	428,705	47,685	10,076	287,500	6,627	501,245	3,456	621,895	415,861	6,355,645

(*a*) From *British Statistical Abstracts*, except the figures for 1896, which are from the *Statistical Register of South Australia*.

MINERAL EXPORTS OF SOUTH AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal.		Copper.		Copper Ore.		Gold Bullion—Kg.		Lead.		Lead. Argentiferous.	
1894.....	19	$55	5,023	$1,043,195	314	$9,815	1,051·9	$642,580	19,978	$917,800	40,098	$6,619,730
1895.....	876	1,795	5,251	1,132,470	268	10,085	1,152·5	643,960	20,766	961,220	25,134	3,026,310
1896.....	37,423	110,480	*b*4,689	1,097,320	354	15,750	265·1	157,905	24,353	1,183,620	13,869	1,416,555
1897.....	39,480	121,210	4,784	1,192,895	554	23,200	556·5	345,670	21,283	1,099,020	16,474	2,518,515
1898.....	39,399	124,680	*b*4,857	1,227,020	545	19,960	593·2	373,450	22,250	1,298,650	27,107	2,944,635

Year.	Lead Ore. Argentiferous		Manganese Ore.		Matte.		Mica.	Salt.		Slate, Roofing.
1894.....	7,971	$194,110	177	$2,585	87	$14,605	$12,650	7,747	$55,235	(*c*)
1895.....	44,839	1,132,015	49	730	(*c*)	.....	(*c*)	(*c*)	.....	(*c*)
1896.....	129,438	3,652,360	(*c*)	.....	3,365	465,910	1,660	*d*19,332	127,260	$4,925
1897.....	180,870	5,394,000	(*c*)	.....	2,324	342,235	5,690	*d*29,374	174,805	18,865
1898.....	139,567	4,779,410	5	50	867	128,300	500	31,471	192,725	3,510

Year.	Spelter.		Tin Ore.		Various Ores.	
1894.....	46	$2,895	(*c*)	.....	.....	$2,155
1895.....	26	1,610	70	$9,075	6	360
1896.....	36	2,020	(*c*)	.....	2	90
1897.....	11	590	(*c*)	.....	52	2,095
1898.....	18	1,290	5	25	30,931	516,765

(*a*) From the *Mineral Statistics of the United Kingdom*, except the figures for 1896, which are from the *Statistical Register of South Australia*, Part IV. and those for 1897, which are from *British Statistical Abstracts*. (*b*) Includes yellow metal. (*c*) Not reported. (*d*) Exported in 1896 also rock salt, 3 metric tons ($50). In 1897, 2 metric tons ($30). (*e*) No tin ore exported in 1896; but tin, block and sheet, 5 metric tons ($1,795). In 1897, 2 metric tons ($825).

MINERAL PRODUCTION OF TASMANIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal.		Copper Ore.		Gold—Kg.		Iron Ore.		Lead-Silver Ore.		Stone.				
											Bluestone.			Limestone.	
											Cubic Yards	Tons.	Total Value.	*Tons*	Total Value
1894	31,417	$65,920	127	$25,000	1,806	$1,127,425	(*b*)	.....	21,401	$1,465,215	(*b*)	(*b*)	.....	1,433	$5,270
1895	33,703	70,145	34	2,600	1,709	1,061,645	(*b*)	.....	18,194	879,785	7,144	7,961	$13,540	1,419	7,500
1896	44,286	86,770	52	7,450	1,947	1,187,870	203	$250	21,150	1,114,740	4,256	(*b*)	3,435	2,621	3,101
1897	43,210	84,640	113,261	1,618,250	2,399	1,446,205	999	370	17,806	966,125	(*b*)	(*b*)	.....	1,702	6,200
1898	49,902	97,271	(*d*)	.....	2,309	1,407,425	1296	1,910	196,707	8,281,065	(*b*)	(*b*)	.....	45,324	53,195

Year.	Stone—*Continued.*						Tin Ore.					
	Freestone, Flagstones and Building Stones.			Rubble or Metal.			Quantity.			Value.		
	Cubic Feet.	*Loads*	Total Value.	*Cubic Feet.*	*Tons.*	Total Value.	Alluvial.	Lode.	Total.	Alluvial.	Lode.	Total.
1894....	48,019	506	$5,895	103,050	(*b*)	$11,770	4,352	1	4,353	$784,125	$200	$784,325
1895....	21,470	(*b*)	6,170	1,140	6,200	5,175	3,887	39	3,926	726,230	180	766,410
1896....	13,575	4,220	7,590	c4,556	(*b*)	1,635	3,867	(*b*)	3,867	804,395	(*b*)	804,395
1897....	82,197	(*b*)	5,805	95,202	13,274	11,010	3,281	1	3,282	545,545	85	545,630
1898....	19,560	(*b*)	4,025	525,258	70,701	89,235	2,882	(*b*)	2,882	578,810	(*b*)	578,810

(*a*) From *Statistics of the Colony of Tasmania*, Part V., Production. There was also mined in 1898 6 tons of nickel ore, valued at $2,000. (*b*) Not reported. (*c*) Represents cart-loads. (*d*) Included with silver-lead ore.

MINERAL IMPORTS OF TASMANIA. (a) (IN METRIC TONS AND DOLLARS; £1—$5.)

Year.	Cement and Whiting.		Coal and Coke.		Copper and Lead.	Earthen and Glassware	Glass, Sheet, Crown, and Plate.	Gold Specie.	Iron and Tin.	Jewelry and Plate.	Railway Material	Silver Specie and Bullion.
1894....	1,033	$8,480	39,881	$60,320	$5,090	$55,605	$8,220	$170,000	$115,050	$18,620	$26,745	$5,000
1895....	1,605	14,855	36,139	58,615	5,590	36,140	8,125	485,000	151,120	11,960	58,465	15,065
1896....	897	8,775	28,145	11,888	5,105	69,010	10,525	(b)	179,885	22,250	148,960	11,100
1897....	1,607	16,895	41,436	105,475	4,950	84,845	15,795	250,000	254,500	44,740	69,190	3,135
1898....	2,465	31,985	47,871	115,271	10,740	78,090	18,072	10,045	280,862	50,175	239,617	58,580

(a) From *Statistics of the Colony of Tasmania.* (b) Not reported.

MINERAL EXPORTS OF TASMANIA. (a) (IN METRIC TONS AND DOLLARS; £1—$5.)

Year.	Bismuth Ore.		Coal and Coke.		Copper Ore.		Earthen and Glassware	Galena.	Gold.				
									Ore.		Bullion—Kg.		Coin.
1894....	3·5	$4,115	891	$5,555	571	$553,065	$345	(b)	547·5	$29,380	1,644	$1,035,265	$28,500
1895....	(b)	.....	101	705	205	48,385	40	(b)	395·0	28,890	1,664	1,062,650	56,950
1896....	1·0	400	380	680	86	8,295	545	(b)	1,082·0	42,685	1,759	1,118,065	(b)
1897....	(b)	.....	2,271	3,095	58	4,560	360	(b)	421·6	19,970	1,829	1,131,440	(b)
1898....	4	550	2,605	3,269	467	30,660	(b)	(b)	1,074·9	18,065	1,521	924,325	325,000

Year.	Iron.						Lead.		Nickel Ore.		Silver.			
	Oxide.		Old Metal.		Rails.						Ore.		Bullion.	
1894....	29	$170	1	$15	(b)	$9,670	(b)	.....	110	$3,560	20,018	$1,076,970	109	$12,250
1895....	66	325	96	670	117	1,395	7·0	$380	(b)	.....	20,575	1,189,580	(b)	.....
1896....	29	145	300	2,710	61	900	2·5	100	(b)	.....	21,150	1,114,740	(b)	.....
1897....	(b)	.....	198	1,450	57	50	19·3	725	(b)	.....	19,838	1,084,465	(b)	.....
1898....	20	110	117	1,295	(b)	.....	(b)	.....	(b)	.....	14,163	838,090	(b)	.....

Year.	Tin.				Zinc.	
	Ore.		Metal.			
1894....	121	$20,780	2,981	$991,490	8·0	$345
1895....	13	1,465	2,771	887,305	9·0	500
1896....	2	10	2,743	795,180	10·4	380
1897....	5	2,960	2,462	749,970	42·9	1,890
1898....	6	835	2,006	705,810	6·0	265

(a) From *Statistics of the Colony of Tasmania.* Additional products in 1897: Jewelry and plate, $6,105; silver coin, $12,875; unclassified ore, 30 metric tons, $1,450; mineral sand, ½ metric ton, $5; copper metal, 1½ tons, $40; blister copper, 4,585 tons, $1,825,585; iron ore, 904 tons, $4,015; manganese ore, 2 tons, $30; mineral oxide, 5 tons, $45; in 1898: Jewelry and plate $8,395; iron ore, 1,795 tons, $8,540; blister copper, 5,092 tons, $1,861,165. (b) Not stated in the reports.

MINERAL PRODUCTION OF VICTORIA. (a) (IN METRIC TONS AND DOLLARS; £1—$5.)

Year.	Antimony Ore.		Coal.		Lignite.		Copper Ore		Gold—Kg. (b)		Lead Ore. Argentif.		Slate and Flagging.		Tin Ore.	
1894..	36	$875	174,406	$474,995	3,571	$10,040	492	$73,810	22,299	$14,339,080	(d)	.....	315	$425	61	$11,430
1895..	(d)	.....	197,334	592,000	1,988	5,239	7	1,050	23,019	14,801,720	20	$500	366	450	76	15,662
1896..	(d)	.....	220,187	565,060	5,908	10,705	(d)	.....	25,041	16,101,740	(d)	.....	386	485	47	8,995
1897..	5	100	240,057	543,200	4,594	6,885	(d)	.....	25,280	16,255,320	(d)	.....	(d)	e125,000	48	8,250
1898..	118	2,550	246,845	515,495	2,915	3,835	(d)	.....	26,041	16,745,140	20	1,200	(d)	100,000	87	19,565

MINERAL IMPORTS OF VICTORIA. (c)

Year.	Coal.		Iron and Steel.	
1893..	611,826	$2,092,420	23,141	$1,753,310
1894..	550,709	977,075	22,795	1,073,520
1895..	553,343	1,005,235	42,788	1,422,400
1896..	511,020	970,175	45,126	1,996,845
1897..	535,812	1,143,235	56,832	2,294,645

MINERAL EXPORTS OF VICTORIA. (c)

Year.	Coal.		Coke and Charcoal.		Gold.		Silver. (Specie)
					Bullion.	Specie.	
1893..	452	$2,115	456	$7,290	$479,960	$13,775,935	$56,905
1894..	495	1,555	198	1,485	178,990	18,414,385	76,090
1895..	161	600	210	2,060	612,505	18,141,180	41,860
1896..	319	1,770	295	3,050	328,025	16,171,535	53,185
1897..	283	1,980	1,212	15,690	969,055	31,392,535	11,605

(a) From Annual Reports of the Secretary for Mines of the Colony; additional products in 1897: Bricks, estimated value, $11,250; pottery, estimated value, $8,900; in 1898: Brick, $12,500; pottery, $10,000; infusorial earth, 142 tons, $1,400. (b) The values are not separately stated in the report, and are estimated at $20 per oz. = $643·02 per kg. (c) From *British Statistical Abstracts.* (d) Not stated in the reports. (e) Represents estimated value of building stones (basalt, sandstone, granite, slate, flagging, etc.).

MINERAL IMPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Brassware.	Cement. *No. of Barrels.*		Clay. Bricks.	Clay. Earthen and Chinaware.	Clay. Tiles.	Coal.		Coke and Patent Fuel. *No. of Sacks.*		Copper. Ingot.
1894.....	$1,530	7,650	$18,085	$325	$33,575	$100	26,348	$108,910	44	$80	(*c*)
1895.....	9,220	15,048	30,290	6,305	68,220	2,815	39,125	147,395	368	510	(*c*)
1896.....	18,980	(*c*)	70,845	19,340	87,485	7,585	71,680	197,515	(*c*)	2,485	$8,375
1897.....	23,935	47,457	81,305	12,120	101,320	16,365	139,369	474,690	(*c*)	10,825	1,695
1898.....	22,835	52,077	96,890	14,350	66,805	10,080	134,315	421,230	7,936	5,340	2,110

Year.	Copper. *Continued.* Rod, Sheet, and Wire.	Copper Sulphate.	Glass and Glassware	Gold Coin.	Gold and Silver Leaf.	Iron. Pig.		Iron. Galvanized (Corrugated Sheet).		Iron. Other Manufactures.
1894.....	$4,095	$775	$31,385	$1,139,500	$475	36	$1,100	2,135	$164,835	$172,465
1895.....	4,335	455	64,410	4,540,000	2,765	289	3,870	3,848	239,935	230,670
1896.....	9,440	735	103,210	4,751,850	2,565	689	12,455	7,467	525,050	771,945
1897.....	12,295	695	137,815	225,000	2,505	953	14,095	11,689	897,035	661,630
1898.....	18,095	885	86,740	675,000	4,265	1,431	20,620	8,997	685,575	631,145

Year.	Iron and Steel Wire.	Steel.	Lead—Sheet, Pig and Pipe.		Paraffine Wax.	Petroleum and Turpentine. *No. of Gallons.*		Plaster of Paris.		Quicksilver. *No. of Flasks.*	
1894.....	$69,095	$18,470	34	$2,255	$1,050	277,968	$48,690	58	$1,035	165	$6,435
1895.....	53,995	21,765	96	6,040	2,640	495,024	82,730	52	1,050	228	9,880
1896.....	67,715	32,935	321	20,250	3,780	853,649	159,765	245	3,545	540	20,225
1897.....	82,545	30,340	537	37,115	3,750	1,058,022	163,735	496	6,225	528	22,705
1898.....	97,580	64,195	469	27,240	8,510	1,165,375	195,685	246	2,725	691	25,280

Year.	Salt. Rock. *No of Sacks*		Salt. Other Kinds.		Silver Coin.	Soda Crystals.		Soda. Caustic	Soda Ash, Nitrate of Potash, etc.	Stone. Grindstones. *Number.*		Stone. Marble and other kinds
1894.....	157	$90	649	$8,020	$58,500	(*c*)	.....	$2,505	$2,920	*b*718	$680	$1,895
1895.....	212	595	883	9,710	92,000	(*c*)	$1,270	2,460	1,240	*b*700	660	7,140
1896.....	(*c*)	260	997	10,690	147,095	(*c*)	2,150	2,420	4,940	(*c*)	2,055	22,070
1897.....	(*c*)	410	1,211	11,568	103,000	143	2,730	3,955	4,940	(*c*)	2,445	20,190
1898.....	332	275	1,346	12,665	1,050	137	2,390	5,510	6,505	1,206	1,140	19,190

Year.	Sulphur. *No. of Casks.*		Tin. Ingot.	Tin. Block, Foil, Plate, and Wares.	Whiting.	Zinc, Sheet and Ingot.
1894.....	537	$1,715	(*c*)	$16,125	$1,080	$795
1895.....	344	1,220	(*c*)	15,900	1,065	3,970
1896.....	(*c*)	720	$4,090	34,395	1,950	6,295
1897.....	(*c*)	600	6,360	24,885	1,530	7,565
1898.....	283	970	6,490	19,150	1,625	7,710

(*a*) From the *Blue Books for Western Australia*, except the figures for 1896, which are from the *Annual Report of the Collector of Customs*, wherein only the values of the imports are stated. (*b*) Number of packages. (*c*) Not stated in the reports.

MINERAL EXPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal.		Copper Ore.		Gold. Bullion—Kg.		Gold. Coin.	Iron.	Lead Ore.		Mica.	Quartz.	Tin Ore.	
1894.....	8,322	$45,030	(*b*)	.....	6,446	$3,985,495	$22,500	.....	(*b*)	.....	(*b*)	$2,425	396	$76,370
1895.....	15,095	92,675	839	$64,760	7,201	4,396,740	460	.....	(*b*)	.....	$15	(*b*)	281	48,515
1896.....	(*b*)	75,205	(*b*)	500	8,748	5,344,040	(*b*)	$130	(*b*)	.....	(*b*)	(*b*)	(*b*)	21,690
1897.....	23,756	129,010	86	5,165	20,994	12,894,885	3,130,400	4,165	(*b*)	$20	1,045	(*b*)	96	16,875
1898.....	26,240	149,135	361	21,330	22,633	19,953,490	75,000	41,090	5	165	(*b*)	(*b*)	69	13,800

(*a*) From the *Blue Books for Western Australia*, except the figures for 1896, which are from the *Annual Report of the Collector of Customs*, Perth. In 1896 there were also exported brassware, $330; silver plate, $85; and salt, $80. (*b*) Not stated in the reports.

AUSTRIA-HUNGARY.

THE latest official statistics of the mineral production, imports, and exports of Austria-Hungary are summarized in the following tables:

MINERAL PRODUCTION OF AUSTRIA. (*a*) (IN METRIC TONS AND FLORINS.)
(1 florin = 40·59 cents.)

Year.	Alum and Pyritous Shale.		Antimony Ore.		Arsenic Ore.		Asphaltic Rock.		Bismuth Ore.		Coal.		Lignite.	
1894.....	10,854	13,495	696	68,632	2	100	116	1,907	570	15,194	9,572,952	33,182,691	17,332,538	32,290,005
1895.....	5,716	9,955	695	55,600	*Nil.*	....	404	8,118	185	.....	9,722,679	34,104,407	18,389,147	34,923,528
1896.....	25,184	20,086	905	80,680	*Nil.*	....	890	11,342	*Nil.*	.....	9,899,522	35,254,925	18,892,537	36,227,606
1897.....	21,585	17,945	864	73,449	*Nil.*	...	300	8,868	1	.....	10,492,771	38,404,864	20,458,093	40,084,423
1898.....	28,914	21,367	679	57,167	*Nil.*	....	643	15,604	*Nil.*	.....	10,947,522	41,142,493	21,088,361	43,492,791

Year.	Copper Ore.		Gold Ore.		Graphite.		Iron Ore.		Lead Ore.		Manganese Ore.	
1894.....	7,235	278,849	86	9,907	24,121	881,980	1,214,768	2,676,114	12,060	836,744	5,055	55,417
1895.....	7,435	286,997	104	38,997	28,443	985,771	1,384,911	2,971,384	12,919	893,944	4,352	41,600
1896.....	6,823	273,269	416	48,412	35,972	1,216,458	1,448,615	3,446,479	14,563	1,058,564	3,950	34,992
1897.....	7,405	279,398	647	32,938	38,504	1,352,646	1,613,876	3,766,048	14,145	1,161,899	6,012	47,961
1898.....	6,791	253,248	448	54,876	33,062	873,566	1,733,649	4,227,688	14,363	1,230,391	6,132	47,795

Year.	Nickel and Cobalt Ore		Petroleum.		Quicksilver Ore		Salt.		Silver Ore.		Sulphur, Crude Rock		Tin Ore	
1894.....	0·55	.....	111,980	3,252,554	84,127	988,754	311,597	24,281,350	18,338	3,000,171	2,435	34,484	24	2,465
1895.....	*Nil.*	.....	188,634	4,464,353	86,683	797,218	278,875	23,901,782	18,113	2,294,043	830	8,393	24	2,556
1896.....	*Nil.*	.....	262,356	5,188,855	83,305	778,455	308,933	22,985,358	18,701	1,921,533	643	8,371	15	2,473
1897.....	*Nil.*	.....	275,204	5,876,692	88,238	814,059	331,084	25,561,441	20,628	1,871,801	530	4,837	16	2,428
1898.....	*Nil.*	.....	.....	.....	88,519	801,076	341,959	26,519,497	20,896	1,762,595	496	4,158	13	1,896

Year.	Uranium Ore.		Tungsten Ore.		Zinc Ore.		METALLURGICAL PRODUCTION. (*a*)							
							Alum.		Antimony.		Bismuth.		Copper.		Copperas.	
1894.....	26	59,404	40	10,908	28,491	439,780	1,150	73,656	279	107,341	0·02	2,023	1,341	744,772	1,480	27,800
1895.....	31	51,761	35	9,154	25,862	384,330	885	54,935	296	92,031	*Nil.*	.....	865	460,990	160	3,593
1896.....	30	28,435	22	6,980	26,887	474,033	919	56,555	422	130,532	*Nil.*	.....	1,001	552,954	170	4,172
1897.....	44	45,011	31	19,248	27,463	530,141	851	55,411	425	113,230	*Nil.*	.....	1,083	614,684	125	3,135
1898.....	51	51,719	36	33,082	27,395	799,290	1,037	72,816	343	104,235	*Nil.*	.....	1,041	627,810	360	11,315

Year.	Copper Sulphate.		Gold—Kg.		Iron, Pig.		Lead.		Litharge.		Nickel and Cobalt Products (*b*)		Mineral Paint.	
1894.....	140	28,832	61·0	96,779	742,372	26,750,603	7,570	1,113,088	2,057	301,514	0·01	156	3,002	76,379
1895.....	246	49,231	75·0	116,753	660,549	22,858,257	8,085	1,204,980	2,034	297,708	*Nil.*	.....	3,164	73,632
1896.....	265	52,557	69·8	98,913	693,188	23,737,250	9,769	1,527,216	1,738	271,508	*Nil.*	.....	3,979	88,760
1897.....	276	58,010	67·6	93,676	762,695	26,626,238	9,640	1,576,210	1,626	265,669	37·6	25,055	3,653	84,049
1898.....	209	47,164	71·5	102,001	837,767	29,513,363	10,340	1,789,512	1,520	280,700	58·8	27,023	3,213	74,392

Year.	Quicksilver.		Silver—Kg.		Sulphur.		Sulphuric Acid.		Tin.		Uranium Salts.		Zinc.	
1894.....	519	1,056.718	38,246	3,081,631	76	536	9,988	264,280	80	80,860	5·0	69,742	6,810	1,268,940
1895.....	535	1,168,512	40,081	2,524,993	*Nil.*	.....	7,431	204,485	60	50,536	4·5	64,009	6,456	1,096,006
1896.....	564	1,149,695	39,904	2,140,913	*Nil.*	.....	7,972	221,182	54	45,650	4·2	47,059	6,888	1,285,891
1897.....	532	1,190,121	40,026	1,952,413	*Nil.*	.....	8,515	201,054	48	42,014	4·4	42,522	6,236	1,216.609
1898.....	491	1.148,537	40,304	1,885,054	*Nil.*	.....	7,008	159,423	48	47,072	4·3	39,578	7,302	1.758,994

(*a*) From *Statistisches Jahrbuch des K. K. Ackerbau Ministeriums; Der Bergwerksbetrieb Oesterreiche.* Iron, common, cast, second fusion. (*b*) In 1897 and 1898 the figures represent nickel speiss, nickel sulphate and cobalt sludge.

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 crown = 20·3 cents.)

Year.	Alum Ore.		Antimony. Ore. (*b*)		Antimony. Crude and Regulus.		Asphaltum.		Carbon Bisulphide.		Coal. Briquettes.	
1894.....	634	$462	1,265	$27,851	385	$50,505	2,370	$63,900	248	$18,069	80,057	$94,512
1895.....	634	462	1,240	26,877	465	57,507	2,285	46,517	237	17,117	29,421	95,134
1896.....	*d* 20	97	1,361	22,572	500	59,313	2,740	57,619	352	25,659	31,179	100,212
1897.....	*d* 60	243	1,800	34,563	528	62,360	3,057	65,001	432	7,132	27,022	87,605
1898.....	*Nil.*	.....	2,201	20,219	855	109,681	3,125	66,422	771	56,393	31,781	102,840

Year	Coal.—*Continued.* Coal.		Coal.—*Continued.* Lignite.		Coke.		Copper.		Copperas.		Copper Sulphate.	
1894.....	1,037,322	$2,207,206	3,181,071	$4,174,249	10,250	$35,968	271	$54,851	795	$3,221	0·8	$3,242
1895.....	1,068,046	2,285,536	3,517,901	4,544,071	12,033	4,580	266	56,488	521	845	(*c*)	.....
1896.....	1,132,625	2,368,715	3,761,728	5,094,630	25,550	58,914	159	31,305	595	964	3·0	288
1897.....	1,118,094	1,819,409	3,870,530	4,118,139	(*c*)	.....	213	46,091	592	1,199	6·5	625
1898.....	1,239,498	2,667,095	4,516,581	5,693,094	(*c*)	.....	153	30,917	745	1,807	*Nil.*	.....

Year.	Galena.		Gold—Kg.		Iron, Pig.		Iron Pyrites.		Lead.	
1894.....	224	$15,681	2,687·07	$1,822,438	312,148	$4,345,074	76,870	$135,337	2,113	$119,540
1895.....	80	5,511	3,187·27	1,973,307	322,206	4,782,382	69,195	113.456	2,277	130,950
1896.....	e405	17,099	3,208·04	2,131,866	333,698	5,649,301	52,697	91,730	1,911	107,711
1897.....	e525	21,435	3,068·00	2,088,224	402,508	5,866,552	44,454	71,500	2,527	149,902
1898.....	e771	22,026	2,768·00	1,842,651	448,621	6,899,188	58,079	93,522	2,305	137,817

Year.	Litharge.		Manganese Ore.		Mineral Paints.		Nickel and Cobalt. Ore.		Nickel and Cobalt. Products.		Petroleum.	
1894.....	689	$47,616	3,953	$4,678	608	$5,618	25	$3,285	23	$3,796	2,061	$26,784
1895.....	615	48,459	3,525	1,206	371	4,543	55	8,583	18	2,200	2,083	28,486
1896.....	465	31,239	2,101	1,591	334	2,435	46	1,829	18	2,971	2,168	21,885
1897.....	155	13,888	4,030	4,710	460	2,966	22	1,269	7·9	957	2,299	22,939
1898.....	188	15,854	8,087	3,776	247	893	*Nil.*	.....	*Nil.*	.....	2,471	24.746

Year.	Quicksilver—Kg.		Salt.		Silver—Kg.		Sulphur.		Sulphuric Acid.		Zinc Ore.	
1894.....	1,837	$1,904	169,282	$6,088,696	20,155·0	$493,204	93	$3,164	4,018	$23,115	(*c*)	.....
1895.....	1,129	972	169,395	5,450,345	20,422·3	499,088	102	3,171	4,223	37,076	(*c*)	.....
1896.....	1,100	906	180,133	6,191,861	19,916·0	477,677	138	3,781	3,550	26,441	(*c*)	$338
1897.....	700	874	171.711	5,375,788	26,790·0	622,634	112	3,317	3,397	26,258	80	*Nil.*
1898.....	6,800	5,562	178,551	5,679,534	18.799·0	448.427	93	2,883	1,318	13,906	80	3,634

(*c*) From *Magyar Statisztikai Evkönyv.* (*b*) This does not include the ore consumed in smelting the product of antimony. (*c*) Not stated in the report. (*d*) Represents refined alum. (*e*) Galena and other ores of lead.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (a) (IN METRIC TONS AND FLORINS.)

(1 florin = 40·52 cents.)

Year.	Alum.		Aluminum and Alloys.		Aluminum Sulphate and Chloride.		Amber.		Ammonium. Chloride and Sulphate.		Ammonium. Hydrate.		Ammoniacal Liquor.		Antimony Ore.	
1894.....	404	40,400	32·2	74,760	717	39,816	65·7	990,010	227	78,016	70	9,048	1,048	15,720	71·6	8,592
1895.....	338	30,465	47·8	98,330	1,278	68,900	81·8	879,270	305	100,797	103	13,442	877	13,151	15·0	3,150
1896.....	359	32,355	50·3	94,790	1,128	56,385	63·6	1,892,400	323	116,866	71	9,243	507	7,604	15·5	3,255
1897.....	346	31,158	67·1	125,240	1,351	67,597	64·9	726,000	339	106,074	128	16,692	565	6,775	8·0	1,680
1898.....	338	30,033	101·2	155,357	1,822	72,864	45·2	526.000	430	105,719	80	10,374	230	2,762	12·3	2,583

Year.	Antimony—*Continued.* Regulus.—Kg.		Arsenic, Arsenious Acid, and Sulphide.		Asbestos. Crude.		Asbestos. Manufactures.		Asphalt. Crude Rock.		Asphalt. Mastic and Bitumen.	
1894.....	1,300	481	320·0	74,516	556	63,541	108·7	153,090	3,918	72,256	758	53,090
1895.....	2,100	777	292·5	67,869	432	56,550	108·3	110,175	2,410	44,873	372	52,336
1896.....	700	231	309·2	76,318	185	29,596	165·1	167,615	4,715	77,952	1,621	89,145
1897.....	600	195	256·9	60,052	625	82,602	134·4	121,935	5,824	97,296	1,309	72,012
1898.....	28,200	10,011	287·4	85,200	609	58,289	137·9	135,500	5,973	101,552	1,117	61,435

Year.	Barium Sulphate (Including M'f'd Product).		Borax. Crude, and Boracic Acid.		Borax. Refined.		Brass, German Silver and Tombac. Crude, Old, and Remnants.		Brass, German Silver and Tombac. Bars, Sheets, Wire, etc.		Brass, German Silver and Tombac. Wares.	
1894.....	4,876	92,380	1,320	158,364	64	17,948	2,286	894,698	72·7	34,686	462·2	1,386,600
1895.....	5,098	134,565	1,908	228,863	62	14,237	2,742	1,166,927	131·4	63,425	510·4	1,531,200
1896.....	5,377	142,126	1,363	208,055	76	17,595	3,118	1,345,163	118·4	58,398	526·2	1,578,600
1897.....	4,947	130,321	1,206	171,980	63	12,888	2,660	1,223,631	151·9	126,598	549·1	1,647,300
1898.....	5,012	131,940	784	112,177	185	37,060	3,232	1,575,367	182·0	128.942	607·4	1,913,310

Year.	Calcium Chloride.		Cement.		Chrome Ore.		Clay Products. Kaolin and Feldspar.		Clay Products. Manufactures.		Clay Products. Potters' and Other Clays.	
1894.....	1,967	157,336	33,286	599,882	1,081	51,634	5,798	172,913	159,001	2,549,000	24,981	249,813
1895.....	2,538	188,616	32,012	514,413	1,827	94,905	6,532	190,383	194,476	2,838,000	27,498	249.254
1896.....	1,989	159,088	35,290	566,240	1,891	104,005	7,425	206,652	213,208	3,213,000	30,072	270.645
1897.....	1,820	145,640	32,479	521,891	1,109	55,470	6,913	188,439	186,297	3,083,000	28,925	261,638
1898.....	2,851	213,855	30,745	493,697	2,206	99,252	7,991	220,401	183,822	3,076,000	31,905	287,147

Year.	Coal.		Coal Lignite.		Coke.		Copper. Ore.		Copper. Crude and Old.		Copper. Bars, Sheets, Wire, etc.	
1894.....	4,048,477	26,135,871	17,453	94,247	437,299	4,399,109	24·8	2,976	13,383	7,286,434	97·2	66,375
1895.....	4,503,003	27,790,484	16,797	90,702	533,402	5,605,287	31·0	3,720	11,747	6,922,160	97·9	72,105
1896.....	5,174,321	31,390,639	19,981	107,898	491,028	5,110,085	0·7	56	13,666	8,372,122	126·2	98,256
1897.....	5,121,475	31,382,217	19,609	107,720	533,463	5,699,121	81·4	11,396	15,926	9,952,699	94·4	77,453
1898.....	5,396,760	33,860,798	19,393	106,114	606,783	7,190,853	63·6	8,180	17,443	11,765,337	158·6	137,856

Year.	Copper Sulphate.		Copper and Iron Sulphates, Mixed.		Copperas.		Cryolite.		Fertilizers, Mineral.		Fluorspar.		Glass. Crude and Ground.	
1894.....	1,916 4	925,788	86	7,704	718	18,663	219	54,196	102,173	306,519	3,618	83,745	198·4	79,636
1895.....	895·3	170,107	56	5,058	871	13,068	229	120,485	89,286	267,858	3,598	36,836	236·1	94,440
1896.....	2,084·4	449,987	40	3,600	575	8,633	265	139,300	96,575	289,728	3,821	94,267	255·8	102,320
1897.....	6,822·5	1,444,976	153	13,779	401	6,020	211	110,485	75,834	227,502	4,201	103,360	269·7	107,880
1898.....	5,270·8	1,133,496	93	8,379	466	5.824	275	146,690	84,820	254,460	4,169	104,522	299·0	119,600

Year.	Glass.—*Continued.*				Gold.					
	Remnants.		Manufactures.		Bullion—Kg.		Old and Dross. Kg.		Coin—Kg.	
1894.....	4,736	82,992	4,177	1,754,000	3,827	5,950,880	6,260	50,200	17,416	25,422,862
1895.....	4,213	73,727	4,088	1,976,000	5,228	8,058,617	4,761	190,725	30,279	42,820,107
1896.....	3,744	65,515	3,989	2,119,000	13,053	19,398,565	5,051	43,150	30,149	41,764,178
1897.....	4,651	81,396	3,782	2,187,000	33,668	50,326,926	5,572	57,800	32,752	44,739,232
1898.....	6,860	120,052	4,008	2,259,000	487	138,906	7,920	28,800	15,917	21,806,290

Year.	Graphite.		Gypsum. Crude.		Gypsum. Burned.		Iron.					
							Ore.		Pig and Old.		Manufactures.	
1894.....	733	36,645	1,375	15,130	10,629	265,725	88,252	927,746	133,849	3,957,000	20,171	9,334,000
1895.....	640	26,066	850	9,350	10,916	272,909	117,600	1,845,728	175,400	5,127,000	23,078	9,976,000
1896.....	697	28,102	821	9,032	11,736	258,181	107,018	1,327,982	148,217	5,143,000	21,910	10,646,000
1897.....	948	38,305	980	10,781	12,101	266,217	134,778	1,798,413	164,433	5,977,000	28,876	11,456,000
1898.....	1,109	45,398	991	10,900	13,300	292,600	178,507	1,712,508	173,919	6,125,000	23,688	11,568,000

Year.	Iron and Steel. Bars, Sheets, Wire, etc.		Lead.									
			Ash—Kg.		Alloys, Crude.		Bars.		Litharge.		Ore.	
1894.....	24,313	2,058,000	·5	25	8,842	1,088,914	200·3	30,045	358·7	64,644	290	19,602
1895.....	30,909	2,556,000	·9	45	8,974	1,166,607	207·9	41,580	355·5	63,990	416	32,479
1896.....	27,809	2,635,000	2·5	200	7,221	974,849	218·3	39,294	333·4	44,346	540	39,435
1897.....	18,625	1,745,000	6·7	469	5,867	841,784	147·9	28,101	224·0	40,320	441	35,312
1898.....	26,421	2,508,000	7·6	532	9,746	1,559,360	152·6	30,520	280·5	56,100	459	41,328

Year.	Lead.—*Continued.*				Magnesium Chloride.		Manganese Ore.		Millstones.		Muriatic Acid.	
	Red and Yellow.		White.									
1894.....	470·2	94,040	243·4	68,152	1,148	55,095	2,444	95,011	1,160·4	162,456	498	9,952
1895.....	371·3	74,260	187·5	46,406	1,353	54,136	2,772	98,329	1,229·5	172,130	467	9,334
1896.....	431·7	82,023	156·1	34,784	1,333	53,320	7,371	264,044	1,204·9	168,686	529	10,576
1897.....	543·3	102,771	111·4	24,229	1,580	57,383	8,018	304,684	1,274·9	178,486	721	14,412
1898.....	555·3	117,060	115·1	26,473	2,196	78,604	5,396	172,675	1,428·9	200,046	760	15,320

Year.	Nickel.				Nickel and Cobalt Ores.		Nitric Acid.		Colored Earths.		Ozokerite.	
	Crude and Old.		Sheet, Wire, etc.									
1894.....	130·5	261,000	2·8	7,250	1,394	129,500	19	3,281	4,205	144,696	(*b*)	.....
1895.....	168·4	255,968	4·9	14,240	1,020	260,025	16	2,240	4,244	145,188	2·2	688
1896.....	160·7	252,299	5·2	13,360	719	190,482	21	2,898	4,362	145,840	2·0	600
1897.....	157·4	254,988	6·9	18,090	55	16,470	23	3,178	4,553	154,244	2·4	720
1898.....	137·0	221,940	9·1	22,980	510	163,168	22	3,150	4,979	172,280	2·5	875

Year.	Peat and Peat Coke.		Petroleum.				Phosphorus and Phosphoric Acid.		Potash.		Potassium. Chloride.	
			Crude.		Refined.							
1894.....	1,649	12,089	126,159	4,430,006	17,019	1,289,322	334·8	401,760	102	18,679	2,406	186,426
1895.....	1,998	14,552	120,479	6,078,067	16,876	1,416,150	206·1	247,320	285	58,590	2,679	200,888
1896.....	2,002	14,618	69,013	2,822,506	17,943	1,303,280	223·9	268,680	987	164,863	2,475	185,603
1897.....	2,189	15,980	70,573	2,271,287	21,249	1,461,831	209·5	248,885	333	62,761	2,206	198,495
1898.....	1,511	11,030	58,580	1,814,470	22,299	1,710,952	209·3	244,960	300	49,290	2,258	203,202

Year.	Potassium Chromate.		Pyrites.		Quicksilver. Kg.		Salt.		Silica, Quartz and Sand.		Silver.	
											Bullion—Kg.	
1894.....	75·9	34,155	59,685	634,883	11,100	18,426	32,822	328,828	61,544	436,737	73,140	3,705,004
1895.....	29·5	12,685	54,610	655,314	4,200	8,316	40,396	405,359	58,494	411,225	49,370	2,482,147
1896.....	34·1	14,663	50,691	608,289	1,300	2,418	53,680	537,367	59,150	449,831	138,420	7,219,890
1897.....	33·8	14,534	49,462	593,543	1,000	1,930	46,057	460,651	61,532	487,895	99,900	4,535,460
1898.....	2·8	1,008	52,282	62,738	6,300	12,726	41,870	418,801	70,296	548,001	15,400	685,328

Year.	Silver.—*Continued.* Old and Dross—Kg.		Silver.—*Continued.* Coin—Kg.		Slag and Slag Wool.		Slate, Roofing and Other.		Soda. Crude and Crystallized.		Soda. Calcined.	
1894.....	4,210	78,620	56,130	2,370,840	1,065	5,008	17,607	1,069,000	258	9,088	1,096	56,459
1895.....	60	1,740	4,400	225,882	961	4,612	15,667	982,000	40	1,197	551	27,530
1896.....	550	2,150	5,730	343,110	240	1,129	15,932	970,000	57	1,725	1,332	79,908
1897.....	1,000	6,520	3,670	189,860	4,717	22,168	16,758	1,015,074	45	1,359	2,787	167,208
1898.....	3,000	4,200	5,470	258,560	9,655	48,298	16,025	973,000	53	1,605	2,408	121,128

Year.	Sodium. Bi-Sulphate.		Sodium. Chromate.		Sodium. Hydrate.		Sodium. Nitrate.		Sodium. Sulphate.		Stone. Marble.	
1894.....	140	16,860	248·4	94,392	1,669	167,084	39,522	4,149,791	8,503	212.723	1,898	56,634
1895.....	137	6,860	152·7	53,445	1,163	113,002	43,059	4,521,176	6,617	132,350	1,886	56,592
1896.....	144	7,205	40·3	14,105	835	71,794	33,086	3,308,630	4,678	81,867	2,347	70,398
1897.....	91	4,565	10·6	3,710	1,450	123,796	39,600	3,960,040	2,879	50,395	2,333	69,997
1898.....	89	4,460	·4	104	1,498	129,656	41,773	4,177,320	4,476	78,339	2,769	83,061

Year.	Stone.—*Continued.* Lithographic Stone.		Stone.—*Continued.* Paving.		Stone.—*Continued.* Not Elsewhere Specified.		Sulphur.		Sulphuric Acid.	
1894.....	670·8	63,726	24,860	248,598	97,609	1,161,016	14,410	684,482	1,530	46.656
1895.....	683·9	67,707	5,459	54,590	79,869	822,608	14,709	661,902	1,566	43.053
1896.....	647·3	64,068	8,476	84,759	87,796	726,813	15,221	684,945	3,522	96,859
1897.....	523·7	51,847	16,961	169,607	118,848	922,149	21,406	899,856	5,877	147,418
1898.....	786·5	77,864	11,234	112,341	99,198	861,263	20,655	895,790	9,724	243,472

Year.	Tin. Ingot, Crude, Old, etc.		Tin. Salts.		Whetstones.		Zinc. Metal.		Zinc. Bars, Sheets, Wire, etc.		Zinc. Calamine and Other Ores.	
1894.....	3,186	2,823,456	84·7	59,290	3,673	269,816	15,315	3,139,519	979·0	245,517	6,760	214,663
1895.....	3,088	2,430,745	46·0	28,580	3,559	265,978	17,156	3,053,715	611·0	148,087	7,691	312,891
1896.....	3,344	2,455,002	27·7	15,789	3,851	289,022	17,539	3,507,720	551·6	144,285	9,022	297,946
1897.....	3,467	2,652,332	22·3	12,488	4,151	269,822	16,599	3,485,748	355·8	93,036	7,863	283,054
1898.....	3,769	3,282,540	20·9	13,794	3,490	226,876	17,471	4,280,420	453·1	131,153	14,112	649,147

Year.	Zinc.—*Con.* White.		Metal Wares Other than Iron and Precious Metals.		Ore and Minerals Unspecified.	
1894.....	636	165,386	1,732	3,205,000	1,152,054	3,162,982
1895.....	510	127,475	1,890	3,626,000	1,268,675	3,525,669
1896.....	590	147,575	1,979	3,708,000	1,311,950	3,657,339
1897.....	577	139,476	2,056	4,084,000	1,557,410	4,328,787
1898.....	697	174,457	2,121	4,387,000	1,589,972	4,185,663

From *Statistik des Auswärtigen Handels Oesterreichisch Ungarischen Zollgebiets.* Although the present monetary system of Austria-Hungary, inaugurated in 1892, has for its unit the crown, the florin of the former system still continues in use during the transitional period, and the imports, exports, and production are stated in florins in the Austrian and Hungarian statistical publications. Since 1892 the florin is accepted as equivalent in value to two crowns, the value of the gold crown being 20·26 cents in United States money, while the silver crown is equal to 17·4 cents at the United States coining rate. The present value of the florin in gold is therefore equal to 40·52 cents in United States money, which equivalent should be used for converting the values in the table into United States currency. (*b*) Not reported.

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (*a*) (IN METRIC TONS AND FLORINS.)

(1 florin = 40·52 cents.)

Year.	Alum.		Aluminum and Alloys. Unman'fac'd		Aluminum, Sulphate and Chloride.		Amber.		Ammonium. Chloride and Sulphate.		Ammonium. Hydrate.		Ammoniacal Liquor.	
1894.....	107·0	8,829	1·4	3,010	179	10,008	5·7	181,200	3,581	501,801	114	21,087	333·0	4,845
1895.....	59·6	5,068	0·2	350	231	11,540	6·9	144,000	876	115,999	92	17,021	412·6	6,189
1896.....	47·0	3,995	(*b*)	.....	287	13,330	6·7	185,400	2,594	338,611	70	12,918	603·7	9,056
1897.....	69·9	5,767	0·4	690	210	11,561	6·2	169,200	4,188	483,238	39	7,074	582·2	8,883
1898.....	82·7	6,823	0·5	665	253	13,893	6·0	150,000	4,886	572,565	23	4,068	723·9	10,859

Year.	Antimony. Ore.		Antimony. Regulus.		Arsenic, Arsenious Acid, and Orpiment.		Asbestos. Crude.		Asbestos. Manufactured.		Asphalt. Rock and Earth.		Asphalt. Mastic and Bitumen	
1894.....	87·9	19,338	354·6	120,564	3·5	941	113	8,416	6·3	14,000	376	11,858	985	98,560
1895.....	193·5	42,570	368·9	129,115	6·4	1,688	122	12,569	9·7	15,000	145	2,915	1,188	29,579
1896.....	218·0	47,960	440·7	143,238	26·1	6,616	48	4,988	9·8	20,000	134	2,642	1,692	42,304
1897.....	259·4	63,668	358·8	113,022	16·5	4,072	56	4,111	18·6	30,000	102	1,934	2,593	64,825
1898.....	266·1	58,542	679·5	224,235	28·7	8,190	150	8,918	28·0	55,873	183	3,532	2,126	53,160

Year.	Barium Sulphate, Including Manufactured Product.		Borax. Crude, and Boracic Acid.		Borax. Refined.		Brass, German Silver and Tombac. Crude, Old and Remnants.		Brass, German Silver and Tombac. Bars, Sheets, Wires, Etc.		Calcium Chloride.	
1894.....	278	25,879	28·1	8,430	26·7	8,010	893·5	351,826	595·1	466,860	314	26,244
1895.....	392	21,522	30·9	9,270	14·1	3,948	1,115·4	472,727	458·9	356,946	267	22,186
1896.....	203	16,733	5·7	1,767	1·5	405	1,469·2	589,839	399·0	380,074	114	9,559
1897.....	70	7,308	8·6	2,580	2·2	556	1,698·9	710,493	679.9	674,639	111	8,890
1898.....	111	9,050	5·9	944	3·5	875	1,879·3	849,411	889·3	877,944	113	9,016

Year.	Cement.		Chrome Ore.		Clay Products. Manufactures.		Clay Products. Kaolin and Feldspar.		Clay Products. Potters' and Other Clays.		Coal.	
1894.....	16,856	274,866	241	15,582	50,339	7,980,000	53,688	1,304,979	33,008	330,075	629,670	4,843,901
1895.....	12,804	180,492	385	24,999	51,767	8,014,000	56,208	1,606,557	37,667	376,665	640,963	4,868,889
1896.....	16,721	238,228	142	9,237	67,270	8,591,000	67,381	2,241,422	41,276	417,759	658,868	5,001,742
1897.....	19,786	281,339	153	9,204	55,895	8,200,000	66,609	2,296,891	46,968	469,680	701,919	5,333,127
1898.....	23,989	339,653	121	6,633	66,005	8,511,000	74,008	2,478,271	52,851	526,507	824,780	6,719,489

Year.	Coal, Lignite.		Coke.		Copper. Ore.		Copper. Crude and Old.		Copper. Bars, Sheets, Plates, etc.		Copper. Sulphate.	
1894.....	6,902,494	23,412,076	110,460	1,504,569	15·3	459	254·8	129,948	469·1	319,266	48·7	9,499
1895.....	7,143,234	23,902,409	119,051	1,618,155	16·6	1,328	151·1	86,127	353·8	250,426	161·9	33,999
1896.....	7,562,721	24,320,812	116,608	1,550,016	12·4	992	227·8	134,402	199·2	148.717	46·9	10,558
1897.....	8,108,975	26,857,181	145,056	1,976,486	0·1	8	159·3	95,580	180·5	147,478	14·5	3,263
1898.....	8,351,955	29,372,774	194,289	2,739,712	11·9	1,190	172·8	112,320	265·6	228,672	29·5	6,638

Year.	Copperas.		Cryolite.		Fertilizers, Mineral.		Fluorspar.		Glass. Remnants.		Glass. Manufactures.	
1894.....	256	7,049	15·9	2,520	87,144	217,861	70	1,229	1,568	27,445	47,507	19,276,000
1895.....	301	6,915	10·9	5,585	62,663	156,658	44	1,252	1,070	18,725	50.843	23,048,000
1896.....	392	7,832	2·3	1,235	65,667	164,168	40	1,272	1,638	28,669	52,752	24,298,000
1897.....	648	17,176	9·8	5,110	44,979	112,448	27	962	2,086	36,498	50,082	22,370,000
1898.....	539	11,860	22·8	11,656	42,673	106,683	29	568	1,904	33,315	50,304	21,724,000

Year.	Gold. Ore.		Gold. Bullion—Kg.		Gold. Old, Remnants, and Dross—Kg.		Gold. Coin—Kg.		Graphite.		Gypsum. Crude.	
1894.....	(b)	.....	85	82,450	38,473	227,680	15,266	22,455,120	11,536	442,374	718	7,899
1895.....	1·1	440	306	338,030	116,911	582,480	15,452	21,886,245	11,928	500,262	662	7,285
1896.....	45·5	7,280	381	445,770	131,942	581,120	23,926	33,388,428	13,091	549,190	899	9,892
1897.....	36·9	5,535	239	277,240	119,336	881,275	33,316	45,810,175	14,229	571,496	1,496	16,456
1898.....	13·5	2,025	27	31,320	137,478	929,400	42,620	58,571,080	17,109	630,589	901	9,909

Year.	Gypsum—*Con.* Calcines.		Iron. Ore.		Iron. Pig, Old, etc.		Iron. Manufactures.		Iron and Steel. Bars, Sheets, Wire, etc.	
1894.....	1,529	22,872	144,309	1,010,164	11,583	467,000	20,884	11,099,000	10,963	989,000
1895.....	1,439	25,990	165,402	1,488,615	9,786	441,000	18,698	12,423,000	9,993	1,782,000
1896.....	1,376	20,265	214,390	1,929,509	11,712	510,000	17,674	10,599,000	12,428	2,175,000
1897.....	1,804	29,688	247,856	2,354,631	12,064	609,000	21,064	10,906,000	17,887	3,242,000
1898.....	2,166	47,597	302.317	3,174,325	15,803	653,000	22,724	12,395,000	23,231	4,008,000

Year.	Lead.											
	Ore.		Metal and Alloys.		Ash.		White.		Red and Yellow		Litharge.	
1894.....	1,232	86,219	242·4	35,513	70	3,500	132·2	47,592	3·9	801	1,466·3	209,030
1895.....	3,758	225,618	207·8	33,197	118	5,895	222·7	79,118	23·7	4,749	781·7	156,340
1896.....	3,076	215,318	272·8	47,006	113	5,660	171·0	54,720	33·0	6,600	597·0	119,400
1897.....	2,488	195,056	241·1	39,657	114	7,952	47·5	15,798	23·9	5,019	354·7	70,940
1898.....	2,253	202,806	544·6	140,751	100	8,024	54·9	19,490	45·4	10,329	188·4	41,448

Year.	Lime. (Hydraulic and Caustic.)		Manganese Ore.		Metal Wares Other than Iron and Precious Metals.		Millstones.		Muriatic Acid.		Nickel, Bars, Sheets, Wires, Etc.	
1894.....	63,326	696,789	2,099	52,509	3,686	7,107,000	1,853	296,400	1,908	51,196	378·1	869,630
1895.....	34,698	381,675	425	14,988	3,807	6,854,000	1,977	296,490	1,460	31,832	54·8	145,305
1896.....	76,895	878,096	701	21,740	4,362	7,360,000	1,831	274,605	1,246	25,090	273·1	684,100
1897.....	83,110	947,914	622	20,536	4,486	7,827,000	1,773	265,965	1,439	30,222	169·6	362,960
1898.....	89,067	1,018,922	1,961	58,827	4,841	8,086,000	2,109	316,365	1,614	34,710	75·8	165,790

Year.	Nickel and Cobalt Ores.		Nitric Acid.		Colored Earths.		Ores and Minerals Unspecified.		Ozokerite.	
1894.....	129	25,432	400·7	64,926	2,580	95,952	494,541	1,594,711	5,166	1,433,678
1895.....	139	24,507	418·1	69,188	2,244	86,458	795,818	2,970,671	5,054	1,498,942
1896.....	113	23,186	359·8	59,520	1,700	64,565	458,484	1,571,905	5,722	1,916,708
1897.....	117	27,720	310·1	52,256	1,621	60,994	478,828	1,700,910	5,153	1,906,435
1898.....	121	12,007	293·7	48,461	2,153	81,452	675,310	2,285,834	4,462	1,740,219

Year.	Petroleum.				Peat and Peat Coke.		Potash.		Potassium Chloride.		Pyrites.		Quicksilver.	
	Crude.		Refined.											
1894.....	975	34,117	208	13,505	3,823	27,906	8,189	1,363,074	627·4	48,987	597	8,062	509·5	861,435
1895.....	1,454	54,344	3,863	280,128	3,753	27,395	5,665	966,586	1,073·9	75,173	383	5,173	493·3	975,734
1896.....	1,815	51,811	23,106	993,567	2,701	19,715	4,164	706,084	1,026·2	71,834	341	4,605	569·4	1,059,084
1897.....	1,600	38,410	13,082	534,249	1,655	12,080	5,997	953,488	1,004·6	90,414	255	3,447	529·4	1,021,742
1898.....	1,076	25,826	3,062	117,095	3,400	24,823	7,252	1,163,698	994·4	89,496	3,089	41,032	468·5	946,370

Year.	Salt.		Silica. (Quartz and Quartz Sand.)		Silver.							
					Ore.		Bullion—Kg.		Old and Dross. Kg.		Coin—Kg.	
1894.....	6,811	102,167	26,748	199,799	0·1	7	1,630	74,980	151,740	220,800	102,010	4,228,935
1895.....	6,554	98,318	30,213	226,032	.....	.....	2,130	96,915	88,770	133,750	47,275	2,014,565
1896.....	11,811	177,170	32,844	246,818	16·0	2,080	2,360	107,380	123,220	192,380	182,950	7,688,706
1897.....	182	2,727	31,347	236,818	.....	.....	4,120	165,624	116,700	171,620	114,560	4,203,750
1898.....	240	3,598	39,083	294,514	1·4	140	4,870	195,774	121,400	178,200	26,130	945,940

Year.	Slag and Slag Wool.		Slate, Roofing and Other.		Soda.				Sodium.			
					Crude and Crystallized.		Calcined.		Nitrate, Crude.		Potassium Nitrate, Refined.	
1894.....	58,219	395,891	4,750	255,369	217	10,299	450	28,782	121·1	13,019	52·0	11,960
1895.....	52,688	421,504	4,569	254,750	278	9,832	1,504	81,042	148·7	17,102	294·1	56,025
1896.....	83,187	665,500	4,602	257,426	427	12,810	2,487	149,888	54·9	5,765	159·7	39,925
1897.....	65,966	329,828	4,581	256,153	590	19,455	3,372	215,184	61·0	6,405	120·1	24,320
1898.....	75,018	450,108	4,972	277,154	952	32,851	2,895	150,961	33·1	3,310	179·6	32,328

Year.	Sodium—*Con.*		Stone.									
	Sulphate.		Lithographic		Limestone.		Marble.		Paving.		Not Elsewhere Specified.	
1894.....	692	20,097	27·8	3,477	38,318	191,592	2,823	84,684	41,328	599,260	321,167	2,660,915
1895.....	661	19,163	3·6	452	33,479	167,398	2,737	82,113	29,392	426,167	241,040	1,455,482
1896.....	2,291	51,543	5·7	713	23,249	116,245	3,595	107,888	30,577	443,369	216,666	1,334,424
1897.....	6,910	138,202	22·3	2,788	11,373	56,867	2,994	89,829	38,256	554,715	223 321	1,321,344
1898.....	7,248	145,519	8·1	1,013	25,117	125,584	2,954	88,629	54,958	796,818	262,229	1,560,175

Year.	Sulphur.		Sulphuric Acid.		Tin.						Whetstones.	
					Ingot and Old.		Bars, Plates, Sheets, etc.		Ash.			
1894.....	1,231·6	64,660	8,210	360,307	78·0	129,948	104·2	102,116	273	177,450	2,316	196,826
1895.....	989·3	50,136	6,466	219,637	53·2	86,127	90·4	76,840	248	143,666	2,323	197,447
1896.....	1,230·9	62,056	6,212	197,449	130·2	134,402	77·7	63,714	281	151,632	2,084	172,941
1897.....	947·5	47,375	7,903	235,383	87·0	52,200	75·1	63,835	306	165,024	2,169	184,370
1898.....	923·2	46,160	9,880	244,778	95·7	54,071	72·4	65,884	324	203,994	2,021	171,825

Year.	Zinc.									
	White.		Metal.		Sheets, etc.		Ash.		Calamine and Other Ores.	
1894.....	1,626·2	390,288	446·7	89,340	1,201·7	312,825	134	24,156	6,979	187,680
1895.....	1,686·1	371,382	508·8	94,463	1,157·7	298,089	179	30,362	7,491	269,676
1896.....	1,825·3	383,313	1,256·2	238,678	1,139·2	315,486	277	47,124	9,453	378,104
1897.....	1,673·0	351,330	770·4	120,456	998·1	265,584	197	35,496	12,914	619,876
1898.....	1,240·1	285,223	1,184·4	218,079	756·9	240,352	296	41,692	14,065	843,906

(*a*) From *Statistik des Auswärtigen Handels des Oesterreichisch Ungarischen Zollgebiets.* (*b*) Not reported.

MINERAL PRODUCTION OF BOSNIA AND HERZEGOIVINA. (*a*) (METRIC TONS AND FLORINS.)

Year.	Chrome Ore.		Copper.		Copper Ore.	Iron, Pig.		Iron Ore.	Lignite.		Manganese Ore.		Quicksilver	Salt.	
1895	707	(*b*)	105	(*b*)	(*b*)	2,599	(*b*)	(*b*)	195,422	(*b*)	8,145	(*b*)	(*b*)	12,758	(*b*)
1896	443	13,566	206	(*b*)	(*b*)	10,120	295,158	(*b*)	222,724	469,674	6,821	112,548	(*b*)	13,720	960,393
1897	396	.....	135	.....	3,847	15,606	.....	37,095	229,643	.....	5,344	84,429	(*b*)	13,919	974,323
1898	458	.....	156	.....	3,760	15,263	.....	57,935	270,752	.....	5,320	.....	4·0	14,496	.....
1899	200	.....	180	.....	3,980	13,730	.....	67,030	308,000	.....	5,270	.....	3·3	15 080	.....

(*a*) From *Oesterr. Zeits. für Berg-, Hütten und Salinenwesen*, except the figures for 1897 and 1898, which were furnished by the "Bosnisches Bureau des K. und K. gemeinsamen Ministeriums." Besides the substances specified in the table there was also produced: In 1897, fahlore, 40 tons; in 1898 452 tons; in 1899, 620 tons. Iron pyrites, in 1898, 3,670 tons. (*b*) Not reported.

BELGIUM.

THE most important articles of mineral production in Belgium are coal, iron, lead, zinc, manganese ore and phosphate of lime. Developments in these industries are described specifically under the respective captions elsewhere in this volume. The official statistics of production, imports and exports are summarized in the following tables:

MINERAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f.—$1.)

Year.	Blende.		Calamine.		Iron Ore.		Lead Ore.		Manganese Ore.		Pyrites.		Totals.
1894.....	7,570	$76,000	4,015	$39,700	311,223	$316,440	160	$3,380	22,048	$55,540	3,050	$5,980	$497,040
1895.....	8,080	72,510	4,150	40,340	312,637	296,090	220	5,100	22,478	57,254	3,510	7,230	478,524
1896.....	7,070	71,280	4,560	48,970	307,081	283,564	70	1,610	23,265	69,004	2,560	5,370	479,780
1897.....	6,804	74,340	4,150	41,270	240,774	252,902	108	3,230	28,372	68,540	1,828	3,990	444,272
1898.....	7,350	98,262	4,125	51,250	217,370	211,644	133	4,301	16,440	42,300	147	177	407,934

PRODUCTION OF MINERAL FUEL IN BELGIUM.

Year.	Coal.							Coke.		
	Quantity. Metric Tons.	Value.		Profit.		Number of Workmen.	Average Annual Wages.	Ovens, Active.	Number of Workmen.	Consumption of Coal. Metric Tons
		Total.	Per Ton	Total.	Per Ton					
1894.....	20,534,501	$38,258,420	$1·86	$1,619,560	$·078	117,103	$188	3,201	2,108	2,381,896
1895.....	20,457,604	38,671,540	1·89	1,659,480	·082	118,957	189	3,233	2,130	2,358,668
1896.....	21,252,370	40,402,020	1·90	2,179,400	·102	119,246	196	3,555	2,415	2,709,720
1897.....	21,492,446	44,134,420	2·05	3,911,350	·182	120,382	205	3,845	2,566	2,968,820
1898.....	22,088,335	48,578,780	2·20	4,654,921	·210	122,846	219	4,028	2,519	2,944,096

Year.	Coke—*Continued.*			Briquettes from Coal.					
	Quantity. Metric Tons.	Value.		Number of Works, Active.	Number of Workmen.	Consumption of Coal. Metric Tons	Quantity. Metric Tons.	Value.	
		Total.	Per Ton					Total.	Per Ton
1894.....	1,756,622	$4,546,138	$2·59	37	(*b*)	(*b*)	1,826,226	$3,095,411	$2·33
1895.....	1,749,109	4,810,050	2·75	38	(*b*)	(*b*)	1,217,795	2,956,806	2·43
1896.....	2,004,430	5,700,599	2·84	36	1,334	1,092,340	1,213,760	2,910,596	2·40
1897.....	2,207,840	7,572,891	3·43	37	(*b*)	1,129,791	1,245,114	3,112,785	2·50
1898.....	2,161,162	8,104,357	3·75	37	1,191	1,227,720	1,351,884	3,623,049	2·68

METALLURGICAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. = $1.

Year.	Iron, Crude.								
	Forge Pig.			Foundry Pig.			Bessemer Pig.		
	Quantity.	Value.		Quantity.	Value.		Quantity.	Value.	
		Total.	Per Ton		Total.	Per Ton		Total.	Per Ton
1894.....	378,045	$3,525,100	$9·32	80,110	$757,200	$9·45	170,420	$1,948,440	$11·43
1895.....	329,750	2,949,180	8·94	85,450	747,560	8·75	161,606	1,809,020	11·07
1896.....	362,451	3,734,840	10·30	84,275	805,880	9·56	193,518	2,284,790	11·81
1897.....	426,332	4,653,486	10·91	78,410	912,300	11·63	183,701	2,377,210	12·94
1898.....	308,875	3,329,775	10·78	93,645	1,000,720	10·69	173,085	2,243,380	12·96

Year.	Iron, Crude—*Continued.*						Iron, Manufactures of.			
	Thomas Pig.			Total Pig.			Bar, Crude.		Rails.	
	Quantity.	Value.		Quantity.	Value.					
		Total.	Per Ton		Total.	Per Ton				
1894.....	190,022	$1,934,680	$10·18	818,597	8,165,620	$9·97	107,881	$2,508,080	1,285	$32,300
1895.....	252,428	2,536,020	10·05	829,234	8,041,790	9·65	76,101	1,718,760	525	12,820
1896.....	307,779	3,336,580	10·84	959,414	10,316,180	10·75	81,394	2,080,500	1,027	27,720
1897.....	333,958	4,001,380	11·98	1,095,037	12,144,076	11·73	108,608	2,801,200	1,443	40,400
1898.....	397,801	4,898,140	12·31	979,755	11,580,970	11·82	123,998	3,216,770	837	23,350

Year.	Iron, Manufactures of—*Continued.*						Lead, Crude.		Silver—Kg.*	
	Sheet and Plate.		Wrought.		Other Manufactures.					
1894.....	118,596	$3,545,520	1,236	$70,420	224,292	$5,261,540	14,120	$704,620	28,901	$605,220
1895.....	109,209	3,189,728	741	56,300	259,328	6,168,140	15,573	840,760	31,543	666,000
1896.....	112,597	3,365,698	851	50,240	298,163	7,326,770	17,222	1,029,980	28,509	637,900
1897.....	100,252	3,148,800	872	47,800	263,644	6,834,000	17,028	1,101,760	30,073	631,422
1898.....	91,686	2,831,800	998	61,440	267,521	7,061,230	19,330	1,252,420	116,085	2,477,170

Year.	Steel.									
	Ingots, Cast and Blooms.		Rails.		Tires.		Sheet.		Forge.	
1894.....	405,661	$6,602,140	118,661	$2,326,300	9,769	$328,360	166,981	$3,629,680	5,627	$170,060
1895.....	454,619	6,885,200	122,357	2,508,160	7,359	259,790	179,249	3,916,260	4,551	141,830
1896.....	598,974	10,102,490	147,183	3,174,780	10,497	367,670	268,009	6,070,020	6,702	204,910
1897.....	616,541	11,104,800	136,911	3,295,800	10,870	427,200	272,839	6,743,400	23,104	586,400
1898.....	653,523	11,877,040	117,751	2,868,960	10,958	408,920	314,150	7,981,610	17,902	516,700

Year.	Steel—*Continued.*						Zinc.			
	Plates.		Wire.		Total.		Crude.		Sheet.	
1894.....	37,980	$1,166,280	8,300	$211,420	341,318	$8,040,100	97,041	$7,233,360	31,794	$2,560,260
1895.....	42,444	1,349,144	11,987	308,670	367,947	8,483,854	107,664	7,999,340	34,061	2,668,400
1896.....	64,653	2,216,640	22,267	591,820	519,311	12,625,840	113,361	9,182,440	36,238	3,244,610
1897.....	64,366	2,354,000	19,567	558,400	527,617	13,965,600	116,067	9,936,090	37,011	3,450,710
1898.....	87,219	2,978,590	19,753	567,620	567,726	15,322,000	119,671	11,881,660	35,567	3,841,190

QUARRY PRODUCTION OF BELGIUM. (*a*) (VALUES IN DOLLARS; 5 f. = $1.)

Year.	Barytes. Metric Tons.		Building Stone—Cubic Meters.						Chalk, Marl. Cubic Meters.		Clay. Metric Tons.	
			Conglomerate.		Freestone.		Limestone.					
1894....	40,000	$56,000	(*b*)	.....	135,094	$2,049,100	2,596,305	$2,089,000	33,700	$25,140	172,010	$241,160
1895....	32,750	45,860	245	$6,680	197,353	2,985,660	2,488,840	2,135,940	100,160	81,200	195,485	272,940
1896....	25,000	35,000	160	4,000	152,420	2,742,300	2,646,305	2,143,500	191,100	66,400	83,020	84,500
1897....	23,000	32,200	220	5,500	181,746	3,076,924	3,010,877	2,730,790	204,600	90,680	270,715	3[illegible],952
1898....	21,700	30,380	180	4,680	215,417	3,177,534	2,968,997	2,580,695	287,805	136,220	287,805	436,240

Year.	Dolomite. Cubic Meters.		Feldspar. Cubic Meters.		Flagstones. Square Meters.		Flint for Earthenware. Cubic Meters		Flint and Gravel for Ballast. Cubic Meters.		Limestone for Flux. Cubic Meters.		Marble. Cubic Meters.		Ochers. Cubic Meters.	
1894	(b)	.....	2,150	$3,580	96,109	$52,800	28,625	$34,860	6,640	$3,180	77,900	$44,980	11,849	$383,760	250	$1,000
1895	(b)	.....	1,900	1,560	95,187	29,640	24,870	30,000	202,590	108,200	163,800	87,400	12,790	424,240	500	2,000
1896	21,500	$7,200	2,000	3,200	131,400	91,720	23,450	19,160	244,050	122,140	164,900	75,360	16,315	544,680	700	2,800
1897	52,720	13,908	1,100	1,940	107,572	71,646	25,050	17,720	235,495	110,819	225,300	89,085	17,797	471,954	350	1,450
1898	37,100	13,068	1,000	1,980	170,672	157,650	22,150	17,700	360,960	177,700	212,685	74,740	16,610	547,100	290	1,180

Year.	Paving Stones. *Pieces.*		Phosphate of Lime. Cubic Meters.		Sand. Cubic Meters.		Slate.			Whetstones and Hones—*Pieces.*	
							Pieces.	Cubic Meters	Total Value.		
1894....	84,309,000	$1,441,660	371,776	$879,300	374,700	$141,640	32,011,000	475	$231,540	87,000	$7,360
1895....	92,378,800	1,575,160	506,730	683,140	381,170	158,120	33,652,000	690	251,720	70,000	11,120
1896....	102,295,950	1,798,730	297,470	587,320	418,720	162,960	35,980,000	1,150	269,500	45,850	6,880
1897....	95,542,700	1,739,875	350,056	436,762	559,141	237,196	41,422,000	1,445	335,780	43,150	16,740
1898....	108,025,000	2,016,314	156,920	303,280	287,805	136,220	42,311,000	210	350,660	89,150	22,590

(a) From *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques*, by Emil Harzé. Belgium also produced, in 1896, manganese pig-iron, 11,891 tons, $154,160; kaolin, 180,960 cu. m., $218,560; and in 1897, manganese pig-iron, 12,636 tons, $199,700. (b) Not stated in the report.

MINERAL IMPORTS OF BELGIUM. (a) (IN METRIC TONS AND DOLLARS; 5 f. = $1.)

Year.	Ashes.		Cement.		Chemical Products.			Coal.		Coal Briquettes.	
					Soda Salts.		Not Specified.				
1894.....	4,650	$18,599	23,736	$142,413	175,704	$18,521,762	$4,269,520	1,377,099	$4,227,418	4,317	$13,874
1895.....	6,173	24,694	23,198	139,190	184,306	14,630,218	4,390,884	1,530,364	4,713,521	3,452	11,046
1896.....	6,747	26,988	30,565	183,392	194,202	11,989,011	5,438,285	1,698,376	5,249,466	1,561	5,151
1897.....	10,870	43,481	17,681	113,160	181,676	7,419,315	6,048,428	2,017,344	6,455,501	632	2,212
1898.....	8,199	32,796	34,039	231,471	55,783	1,115,674	5,880,745	2,202,517	7,268,306	1,756	6,585

Year.	Coke.		Copper and Nickel.					Glass and Glassware.		
			Crude.		Hammered, Drawn or Rolled.		Wrought.	Common (Bottles, Broken Glass, etc.)		All Other Kinds.
1894.....	326,188	$1,154,706	9,600	$2,112,060	926	$268,497	$155,098	4,726	$126,810	$429,615
1895.....	362,834	1,327,972	10,480	2,431,424	926	268,462	168,720	6,078	167,085	441,558
1896.....	260,273	976,024	15,506	3,597,466	1,109	321,716	188,981	6,980	213,380	541,225
1897.....	299,606	1,159,306	14,821	3,556,943	1,418	405,576	193,242	4,699	143,130	664,176
1898.....	180,590	784,590	14,947	5,082,274	1,821	765,000	205,705	4,947	99,608	635,021

Year.	Gold (Including Platinum).						Guano.		Iron and Steel Filings.	
	Ore—Kg.		Unwrought—Kg.		Coin.	Jewelry, etc.				
1894.....	.....	.....	1,389	$956,743	$2,310,740	$522,884	27,810	$1,223,650	23·9	$191
1895.....	65	$429	2,714	1,869,403	616,280	523,491	43,017	1,806,725	61·6	492
1896.....	93	614	4,923	3,390,962	599,540	757,507	25,946	1,037,846	11·1	93
1897.....	.....	.....	3,824	2,633,971	1,726,700	701,535	5,162	185,825	26·4	227
1898.....	8,390	55,374	1,282	883,041	372,000	840,593	10,657	383,664	18·0	158

Year.	Iron.									
	Ore.		Pig.		Old.		Hammered, Drawn or Rolled.		Wrought.	
1894.....	1,942,883	$3,691,478	228,593	$2,404,880	24,704	$296,443	21,007	$506,901	4,542	$326,025
1895.....	1,857,624	3,529,485	225,037	2,326,123	22,817	273,809	17,616	462,134	4,922	362,857
1896.....	2,069,676	3,932,385	324,389	3,397,069	53,802	688,668	22,812	646,150	6,116	469,081
1897.....	2,544,377	4,834,316	302,193	3,815,918	51,985	670,603	26,054	792,873	6,336	491,296
1898.....	2,252,553	4,279,850	317,829	3,782,157	52,206	690,390	25,470	781,167	14,918	879,347

Year.	Lead.			Lime.		Petroleum.			
	Pig.		Manufactures.			Crude.		Refined.	
1894.....	43,491	$2,087,566	$97,081	19,209	$37,458	3,718	$52,055	142,429	$3,708,151
1895.....	45,594	2,370,874	21,097	9,083	17,898	1,894	26,540	159,980	5,759,266
1896.....	35,231	1,937,173	17,231	11,522	22,699	95	1,336	158,979	5,405,275
1897.....	43,840	2,630,429	91,580	13,184	26,236	988	13,831	149,501	4,843,844
1898.....	54,867	3,489,549	50,728	12,674	25,349	382	7,446	161,281	5,419,070

Year.	Pottery (*e*).						Resins and Bitumens, Not Specified.		Salt.			
	Terra Cotta. *Pieces.*		Common.		Earthenware.	Porcelain			Crude.		Refined.	
1894.....	39,113	$234,677	2,243	$112,169	287,003	$145,262	194,015	$9,700,769	92,215	$553,288	44,414	$444,143
1895.....	40,531	243,185	2,344	117,216	254,547	155,283	226,133	11,306,672	81,188	487,126	40,625	406,245
1896.....	*b*85,486	629,231	2,065	103,231	*c* 272,894	*d*177,207	216,278	10,813,881	92,408	554,449	38,785	387,852
1897.....	*b*86,493	549,631	2,115	105,756	*c* 298,535	*d*188,434	237,570	11,878,487	96,805	580,827	39,193	391,926
1898.....	92,149	610,058	2,007	104,436	179,866	345,067	269,914	13,495,710	92,300	553,804	50,136	501,364

Year.	Silver.						Steel.					
	Ore—Kg.		Bullion—Kg.		Coin.	Jewelry, etc.	Ingot.		Bars, Sheets, and Wire.		Wrought.	
1894.....	1,300,333	$156,040	17,471	$454,246	$40,920	$323,802	18,518	$296,287	12,040	$352,063	1,028	$257,119
1895.....	1,297,070	155,648	7,936	206,336	2,793,680	343,840	18,405	294,478	17,582	538,880	1,136	283,081
1896.....	1,476,582	177,190	8,980	233,480	6,461,840	415,967	28,435	561,744	15,263	479,341	936	233,924
1897.....	2,533,237	253,324	467,851	9,357,020	*f* 2,083,040	460,244	25,370	518,271	25,869	884,766	1,147	286,415
1898.....	461,914	46,191	299,369	5,987,990	7,655,200	449,244	25,142	526,442	24,761	805,918	1,013	256,925

Year.	Stone.								Sulphur.	
	Roofing Slate. *1,000 Pieces.*		Building Stone, Including Marble and Alabaster.		Cut, Polished, etc.	Paving.		All Other Kinds.		
1894.....	39,504	$158,018	29,546	$590,916	$59,684	2,281	$6,843	75,502	$830,525	24,982	$899,337
1895.....	37,720	150,879	31,156	623,114	53,450	4,249	14,023	83,569	919,263	18,438	590,029
1896.....	38,209	152,834	40,511	891,237	61,769	6,163	22,187	81,360	894,964	14,399	403,177
1897.....	38 754	162,765	47,929	1,054,432	111.313	13,197	47,509	182,950	1,097,417	13,261	371,307
1898.....	38,216	198,724	45,544	1,093,052	63,665	8,926	37,489	239,281	1,312,764	13,322	373,037

Year.	Tin.			Tin Plate.			Zinc.			Ores, Crude Not Elsewhere Specified.
	Ingot.		Manufactures.	Unwrought.		Wrought.	Ingot.		Manufactures.	
1894.....	4,724	$2,220,258	$4,245	1,919	$145,819	$13,781	9,130	$675,633	$10,554	$9,801,923
1895.....	3,216	1,286,230	3,501	2,046	151,511	23,712	8,551	615,643	13,095	8,361,581
1896.....	4,617	1,662,044	2,845	3,208	243,403	27,425	20,182	1,614,588	11,230	10,227,349
1897.....	1,609	547,118	2,634	3,875	294,538	22,972	16,820	1,387,218	10,661	13,048,392
1898.....	1,208	628,084	2,545	3,848	256,180	29,990	17,441	1,757,103	11,575	13,497,359

(*a*) From *Statistique de la Belgique; Tableau General du Commerce avec les Pays, Etrangers, Brussels.* (*b*) In metric tons. (*c*) Corresponding in 1896 to 3,668 metric tons, and in 1897 to 3,713 metric tons. (*d*) Corresponding in 1896 to 824 metric tons, and in 1897 to 815 metric tons. (*e*) There was imported in 1897 also slabs of pottery for paving, building, etc., 7,755 metric tons, $172,795. (*f*) Corresponding to 52,076 kg.

MINERAL EXPORTS OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. — $1.)

Year.	Ashes.		Cement.		Chemical Products.			Coal.		Coal Briquettes.	
					Soda Salts.		Not Specified.				
1894.....	776	$3,105	195,533	$1,173,197	44,678	$3,470,711	$6,235,984	4,589,525	$13,936,342	573,463	$1,825,082
1895.....	1,030	4,121	273,807	1,642,845	55,089	3,965,041	5,555,916	4,661,477	14,357,349	459,702	1,471,046
1896.....	1,084	4,338	277,615	1,665,690	42,857	2,298,603	6,819,396	4,649,799	14,414,377	459,974	1,517,914
1897.....	2,675	10,701	322,024	2,060,951	59,054	2,202,874	7,236,116	4,448,544	14,236,341	615,074	2,152,759
1898.....	615	2,459	419,132	2,850,092	106,252	2,718,179	7,300,805	4,579,955	15,113,851	666,265	2,498,493

Year.	Coke.		Copper and Nickel. Crude.		Copper and Nickel. Hammered, Drawn or Rolled.		Copper and Nickel. Wrought.	Copper and Nickel. Foreign Coin. Kg.		Glass and Glassware. Common (Bottles, Broken Glass, etc.).		Glass and Glassware. Plate.
		$		$		$	$		$		$	$
1894.....	879,278	3,112,644	4,936	1,085,995	1,856	538,237	164,448	494,422	494,422	3,620	33,066	2,562,004
1895.....	870,983	3,187,796	5,205	1,207,508	2,558	741,944	129,893	676	676	4,181	42,923	2,768,243
1896.....	863,067	3,236,501	11,700	2,714,477	2,078	601,277	168,524	94	94	3,647	26,574	3,468,698
1897.....	909,486	3,910,789	9,994	2,398,659	1,996	570,761	198,635	266	266	3,546	26,468	3,761,319
1898.....	878,435	3,865,114	8,511	2,893,856	1,770	743,802	161,726	89	89	2,747	37,480	13,007,365

Year.	Glass—*Continued.* All Other Kinds.		Gold (Including Platinum). Unwrought—Kg.		Gold (Including Platinum). Coin.	Gold (Including Platinum). Jewelry, etc.	Guano.		Iron and Steel Filings.	
		$		$	$	$		$		$
1894.....	160,319	$7,468,692	375	$258,300	$29,760	$45,569	17,820	$784,068	2,217	$17,738
1895.....	156,304	7,400,444	893	615,098	3,547,640	77,426	18,968	796,644	1,488	11,904
1896.....	178,611	12,194,434	3,713	2,557,514	2,666,620	75,570	14,633	585,308	7,025	59,007
1897.....	174,232	11,448,839	2,547	1,754,374	605,120	126,385	14,044	505,585	534	4,595
1898.....	25,226	3,088,385	1,231	817,213	578,120	118,897	21,626	778,542	407	3,561

Year.	Iron. Ore.		Iron. Pig.		Iron. Old.		Iron. Hammered, Drawn or Rolled.		Iron. Wrought.	
		$		$		$		$		$
1894.....	258,529	$491,205	34,386	$906,266	8,491	$101,894	279,891	$7,043,983	33,375	$2,349,299
1895.....	325,909	619,036	33,237	912,301	12,519	150,228	271,066	6,697,454	39,985	2,983,007
1896.....	399,285	739,546	38,465	1,132,480	18,416	235,725	343,072	9,072,611	44,952	3,497,367
1897.....	410,817	780,552	42,554	1,325,352	17,590	226,911	356,835	10,097,254	44,521	3,501,950
1898.....	384,047	729,683	16,789	223,453	23,326	309,676	234,562	4,155,448	74,889	4,984,032

Year.	Lead. Pig.		Lead. Wrought.	Lime.		Petroleum. Crude.		Petroleum. Refined.	
		$	$		$		$		$
1894.....	34,690	$1,665,122	$16,087	411,808	$808,017	1,408·0	$19,689	26,500	$689,005
1895.....	39,906	2,079,800	18,192	496,635	860,172	2,312·0	32,364	29,239	1,052,591
1896.....	31,866	1,725,144	26,821	477,213	940,110	1·6	23	29,321	996,913
1897.....	35,998	2,159,298	33,286	530,588	1,035,969	1·1	15	18,088	586,067
1898.....	40,302	2,563,246	16,450	546,199	1,092,399	782·4	13,256	19,556	657,067

Year.	Pottery. Terra Cotta. *No. of Pieces.*		Pottery. Common.		Pottery. Earthenware.		Pottery. Porcelain.		Resins and Bitumens, not Specified.	
				$		$		$		$
1894.....	101,149	$606,991	3,833	$191,664	5,724	$1,144,827	565·5	$497,647	62,197	$3,251,857
1895.....	127,413	764,481	3,463	173,140	4,048	809,509	400·7	352,599	76,087	3,804,331
1896.....	*b*302,596	1,913,054	2,628	131,394	4,992	513,950	586·7	54,332	86,906	4,345,315
1897.....	*b*294,815	1,773,837	3,197	159,857	4,905	534,532	264·0	45,395	92,591	4,629,538
1898.....	*b*247,970	1,497,938	6,186	81,066	10,981	640,025	218·4	35,382	107,806	5,390,353

Year.	Salt. Crude.		Salt. Refined.		Silver. Ore—Kg.		Silver. Bullion—Kg.		Silver. Coin.		Silver. Jewelry, etc.
		$		$		$		$		$	$
1894.....	871	$5,224	158	$1,579	47,260	$5,671	70,145	$1,823,770	32,244	$1,289,760	$35,597
1895.....	2,136	12,814	117	1,172	19,400	2,328	45,299	1,177,774	42,326	1,933,040	51,452
1896.....	1,434	8,604	129	1,287	19,400	2,328	40,118	1,043,068	16,696	667,840	137,137
1897.....	493	2,959	231	2,309	423,160	42,316	57,933	1,158,660	521	20,851,320	189,959
1898.....	298	1,789	386	3,864	595	59·2	107,385	2,125,700	327	13,083,640	127,194

Year.	Steel. Ingot.		Steel. Bars, Sheets and Wire.		Steel. Wrought.		Stone. Roofing Slate. *1,000 Pieces.*		Stone. Building Stone, Including Marble and Alabaster.		Stone. Cut, Polished, etc.
		$		$		$		$		$	$
1894.....	659	$10,541	120,768	$2,604,402	34,839	$8,709,782	11,395	$88,680	130,754	$889,130	$687,326
1895.....	1,315	21,038	170,328	4,052,108	27,488	5,508,067	15,227	106,657	147,679	1,004,216	645,690
1896.....	1,145	23,272	179,873	4,597,877	38,306	7,843,777	15,495	111,190	161,298	1,290,387	922,147
1897.....	1,291	22,302	183,386	4,842,221	37,353	7,764,848	17,304	138,428	187,180	1,591,033	934,366
1898.....	1,018	19,397	175,249	4,856,964	33,652	6,464,909	16,948	152,584	178,949	1,742,491	861,015

Year.	Stone—*Continued.*				Sulphur.		Tin.		
	Paving.		All Other Kinds.				Ingot.		Manufactures.
1894.....	152,720	$ 916,323	736,611	$2,946,445	3,550	$127,811	1,194	$561,283	$2,566
1895.....	134,338	899,980	730,863	2,923,451	4,576	146,423	1,051	420,453	1,176
1896.....	154,737	1,145,052	796,231	3,184,926	5,335	149,371	1,055	379,732	873
1897.....	153,504	1,197,335	773,531	3,248,832	6,041	169,157	347	118,089	839
1898.....	159,455	1,339,421	917,654	4,221,212	6,355	177,960	508	263,982	2,394

Year.	Tin Plate.			Zinc.			Ores Not Specified.
	Unwrought.		Wrought.	Ingot.		Manufactures.	
1894.....	473·4	$35,975	$210,987	81,248	$6,012,364	$54,486	$3,717,373
1895.....	1,750·2	133,016	21,698	88,316	6,358,748	54,210	4,161,023
1896.....	3,952·3	309,635	11,750	100,369	8,039,501	56,349	4,236,708
1897.....	1,191·0	90,545	15,216	100,228	8,519,380	90,749	6,216,589
1898.....	972·7	74,903	10,562	108,507	10,807,313	102,568	7,080,013

(*a*) From *Statisque de la Belgique: Tableau General du Commerce avec les Pays Etrangers.* (*b*) Metric tons.

CANADA.

THE mineral statistics of the Dominion of Canada as collected by the Geological Survey and the Bureaus of Mines of the various provinces, are summarized in the following tables:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Arsenic.		Asbestos and Asbestic.		Barytes.		Building Material.							
							Stone.	Flagstones. *Sq. Ft.*		Granite.		Marble.		
1895....	*Nil.*	.....	7,943	$368,175	(*q*)	.....	$1,095,000	80,005	$6,667	17,453	$84,888	181	$2,000	
1896....	*Nil.*	.....	11,113	429,856	131	$715	1,000,000	(*q*)	6,710	16,980	106,709	208	2,405	
1897....	*Nil.*	.....	27,617	445,368	518	3,060	*b*1,000,000	(*q*)	7,190	9,385	61,934	(*q*)	.....	
1898....	*Nil.*	.....	21,577	486,227	971	5,258	*c*3,600,000	(*q*)	4,250	(*q*)	73,573	(*q*)	.....	
1899....	52	$4,872	22,938	483,299	658	4,402	*t*4,250,000	(*q*)	7,600	(*q*)	90,542	(*q*)	.....	

Year.	Cement—*Barrels.*				Clay.						Coal.	
	Natural Rock.		Portland.		Bricks. *Thousands.* (*b*)		Fire Clay.		Pottery	Tile and Sewer Pipe. (*d*)		
1895....	(*s*)	.....	128,294	$173,675	*c*808,836	$1,670,000	1,206	$3,492	$151,588	$467,045	3,155,533	$6,739,153
1896....	70,705	$60,500	78,385	141,151	(*q*)	1,600,000	764	1,805	163,427	378,575	3,398,091	7,226,462
1897....	85,450	65,893	119,758	209,380	(*q*)	1,600,000	1,921	5,759	129,629	389,250	3,434,756	7,303,597
1898....	87,125	73,412	163,084	324,168	(*q*)	.....	*b*1,969	5,000	135,000	166,421	3,785,408	8,227,958
1899....	131,387	119,508	255,366	513,983	(*q*)	.....	543	1,295	200,000	161,546	4,142,242	9,040,058

Year.	Coke. (*e*)		Copper (In Ore, etc.). (*f*)		Feldspar. (*g*)		Gold—Kg. (*h*)		Graphite.		Grindstones.	
1895....	48,404	$143,047	3,987	$945,714	.....	*t*$2,545	3,135	$2,083,674	199	$6,150	3,153	$31,932
1896....	45,014	110,257	4,260	1,021,960	882	*t*2,583	4,145	2,754,774	126	9,455	3,368	33,310
1897....	55,044	176,457	6,033	1,501,660	1,270	3,290	9,380	6,027,016	396	16,240	4,148	42,340
1898....	65,721	219,200	8,146	2,159,556	2,268	6,250	20,618	13,700,000	(*q*)	11,098	(*q*)	39,465
1899....	91,463	350,022	6,840	2,655,319	2,722	6,000	31,673	21,049,730	1,105	16,179	4,092	43,265

Year.	Gypsum.		Iron.							
			Chromite.		Ore.		Pyrites.		Pig. (*j*)	
1895....	205,187	$202,608	2,882	$41,801	98,257	$238,070	31,034	$102,594	47,586	$696,440
1896....	187,818	178,061	2,124	27,004	83,377	191,557	30,586	101,155	(*q*)	.....
1897....	217,392	244,531	2,392	32,474	50,000	130,290	35,299	116,750	(*q*)	.....
1898....	198,908	230,440	1,833	24,252	52,763	152,510	29,228	128,872	(*q*)	.....
1899....	221,869	257,329	1,796	23,760	62,997	248,372	25,117	110,748	(*q*)	.....

Year.	Lead (In Ore, etc.). (*k*)		Lime—*Bushels.* (*b*)		Limestone for Flux.		Manganese Ore.		Mica.	Nickel (In Ore, etc.). Kg.	
1895....	7,467	$531,716	5,225,000	*c*$700,000	31,370	$32,916	113	$8,464	$65,000	1,763,823	$1,320,954
1896....	10,977	721,159	(*q*)	650,000	33,985	36,140	112	*t*3,975	60,000	1,540,930	1,188,990
1897....	17,698	1,396,853	(*q*)	650,000	27,371	30,258	14	1,166	76,000	1,813,891	1,399,176
1898....	14,477	1,206,399	(*q*)	.....	30,766	31,153	45	1,600	117,598	2,502,806	1,930,838
1899....	9,917	977,250	(*q*)	.....	48,265	45,662	279	3,960	163,000	2,605,461	*t*2,067,860

Year.	Ochers.		Petroleum, Crude. *Barrels.* (*m*)		Phosphate (Apatite).		Platinum.	Roofing Cement.		Sand and Gravel (Exports).		Moulding Sand.	
1895 ...	1,215	$14,600	726,188	$1,086,788	1,658	$9,565	$3,800	(*q*)	$3,158	251,440	$118,859	6,137	$13,530
1896....	2,043	16,045	726,822	1,155,647	517	3,420	750	178	430	208,909	80,110	5,207	11,478
1897....	3,542	23,560	709,857	1,011,546	894	3,984	1,600	*Nil.*	.....	118,768	76,739	4,976	10,981
1898....	2,124	18,600	700,790	981,106	665	3,665	(*q*)	(*q*)	.....	(*q*)	.....	9,591	21,088
1899....	3,555	19,900	808,570	1,202,020	2,729	16,000	895	(*q*)	.....	(*q*)	.....	12,450	27,480

Year.	Salt.		Silver (In Ore, etc.)—Kg. (*n*)		Slate.	Soapstone.		Terra-Cotta.	Various Products. (*p*)
1895.....	47,515	$160,455	55,229	$1,159,166	$58,900	431	$2,133	$195,123	$801,080
1896.....	39,880	169,693	99,696	2,149,503	53,370	372	1,230	88,855	647,997
1897.....	46,583	225,730	172,891	3,323,395	42,800	142	350	155,595	836,197
1898.....	51,839	248,639	137,911	2,588,296	40,791	(*q*)	.....	167,902	491,660
1899.....	51,796	234,520	95,762	1,834,871	33,406	408	1,960	220,258	487,271

(*a*) From *Reports Compiled by the Geological Survey of Canada.* (*b*) Estimated. (*c*) Includes building stone, brick, tile, lime, sand and gravel, estimated as for previous year. (*d*) Value of tile in 1895, $210,000, sewer pipe, $257,045; in 1896, tile, $225,000. sewer pipe, $153,375; in 1897, tile, $225,000, sewer pipe, $164,250. In 1898 and 1899 sewer pipe is alone reported. (*e*) Oven coke, all the production of Nova Scotia and British Columbia. (*f*) Copper contents of ore, matte, etc., at the following values per lb.: 1895, 10½c.; 1896, 10⅞c.; 1897, 11⅛c.; 1898, 12$\frac{1}{16}$c.; 1899, 17·61c. (*g*) Figures for 1896 includes quartz, 9 tons, $50. (*h*) Fine ounces calculated at $20·67 per oz., or $664·60 per kgm. (*i*) Export returns. (*j*) In 1895 there were 84,558 metric tons of native iron ore ($218,336) converted into pig iron. (*k*) Lead contents of ores, etc., at the following values per lb.: 1895, 3·25c.; 1896, 3c.; 1897, 3·6c.; 1898, 3·78c.; 1899, 4·47c. (*m*) Calculated from inspection returns at 100 gals. crude to 42 gals. refined. The value of the crude per bbl. of 35 imp. gals. was, in 1895, $1·49½; in 1896, $1·59; in 1897, $1·42; in 1898, $1·40; in 1899, $1·48½. (*n*) Silver contents of ores, values for production and exports per oz.: 1895, $0·653; 1896, $0·67; 1897, $0·598; 1898, $0·583; 1899, $0·5958. (*p*) In 1895, includes natural gas ($423,032), mineral waters, 739,382 gals. ($126,048), lithographic stone ($2,000); in 1896, natural gas ($276,301) mineral waters, 706,372 gals. ($111,736), tripoli, 602 metric tons ($9,960); in 1897, mercury, 312 kg. ($324), natural gas ($325,873); also sundry minerals, estimated in part and including actinolite and tripolite; in 1898, natural gas ($320,000), mineral waters ($155,000), tripolite, 922 metric tons ($16,660); in 1899, mineral waters ($100,000), natural gas ($387,271). (*q*) Not reported. (*r*) Nickel contents of ore, matte, etc., in 1899, at 36c. per lb. (*s*) Quantity and value included with Portland cement. (*t*) Includes building stone, brick, lime, sands and gravels and tiles.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year. (*h*)	Aluminum.		Asphalt.		Brass. (*f*)	Bricks and Tiles.	Buhrstones. *Number.*		Calcium Chloride.		Cement.	Chalk.
1895..	*k*33	$3,248	2,789	$41,817	$427,454	$164,132	1,612	$2,172	1,649	$63,486	$251,926	$10,267
1896..	*k*7	7,537	3,843	33,875	477,279	193,585	1,572	2,049	1,565	58,994	255,029	9,075
1897..	*k*8	5,717	842	9,012	457,842	165,073	1,499	1,827	1,361	49,368	260,842	11,826
1898..	(*i*)	7,102	6,006	55,164	560,014	168,012	889	1,813	1,765	55,987	376,315	14,362
1899..	(*i*)	9,275	8,196	95,800	747,557	255,287	1,116	1,759	1,857	50,801	480,414	14,544

Year. (*h*)	Cryolite.		Clays.	Coal, Anthracite. (*b*)		Coal, Bituminous.		Coal Tar. *Barrels.*		Coke.		Copper. (*f*)
1895..	3	$386	$62,730	1,274,019	$5,350,627	1,475,315	$3,373,608	19,200	$36,581	39,223	$149,434	$258,214
1896..	21	2,699	62,984	1,428,255	5,667,096	1,586,657	3,352,767	18,407	31,209	55,994	203,826	294,410
1897..	16	2,106	59,386	1,322,049	5,695,168	1,596,789	3,313,826	23,661	36,942	75,596	267,540	270,086
1898..	17	2,315	72,795	1,325,139	5,847,685	1,785,838	3,225,151	26,702	35,194	122,527	347,040	867,452
1899..	50	2,120	88,517	1,583,471	6,490,509	2,220,735	3,736,663	39,206	54,447	128,172	362,896	798,326

Year.	Copper Sulphate.		Earthenware and China.	Emery (Wheels and Ground).	Explosives.	Flint and Stones.		Fuller's Earth.		Glass.	Gold and Silver.	
											Coin and Bullion.	Manufactures.
1895..	877	$62,777	$547,935	$22,344	$142,305	235	$2,705	102	$1,904	$1,120,986	$4,576,620	$341,476
1896..	710	57,380	575,498	18,840	136,818	287	3,027	(*i*)	1,884	1,104,481	5,226,319	340,241
1897..	516	40,469	595,822	11,231	131,562	475	5,344	(*i*)	1,552	1,139,764	4,676,094	296,143
1898..	738	57,497	674,874	15,478	141,731	389	5,344	(*i*)	3,330	1,024,706	4,390,844	297,242
1899..	726	61,749	916,727	22,343	212,968	243	2,977	(*i*)	3,418	1,343,058	4,705,134	342,330

Year.	Gravel and Sand.		Iron Sand.		Iron and Steel. (f)	Lead. (f)	Lead Pigments and Zinc White.		Lime. No. of Bbls.		Litharge.	
1895..	17,789	$24,779	10	$1,300	$9,249,749	$192,046	3,984	$282,353	12,008	$5,748	542	$32,953
1896..	17,198	24,604	15	3,008	10,952,607	239,109	5,312	367,569	10,289	7,331	481	32,817
1897..	19,330	25,222	17	1,694	11,226,484	223,125	4,677	347,539	16,108	10,529	546	34,536
1898..	29,164	43,287	34	2,999	16,407,644	335,278	5,753	448,659	12,850	9,002	519	32,904
1899..	27,477	42,209	18	1,979	19,768,725	368,450	6,581	514,842	15,790	11,124	432	32,518

Year.	Litho-graphic Stones.	Mineral Oils. Gallons.		Ochers.		Ores of Metals.		Phos-phorus.	Potassium Compounds Except Saltpeter.		Precious Stones and Jewelry.	Pumice Stone.
1895..	$10,078	6,852,425	$442,402	73	$2,077	10	$261	8	$8,065	$34,525	$301,150	$3,609
1896..	4,964	7,180,061	626,371	158	2,995	.....	24,088	10	9,525	41,037	675,490	3,721
1897..	6,360	7,696,258	614,982	293	5,533	25,021	105,072	16	8,575	35,132	506,728	2,908
1898..	7,791	8,175,006	623,364	448	9,428	69,766	155,583	5	4,618	32,502	743,607	3,829
1899..	6,223	9,708,189	682,447	502	10,432	61,354	153,952	1	778	52,177	923,837	5,973

Year. (h)	Quicksilver.		Kainite.		Sal Ammonia. (c)		Salt.		Saltpeter.		Silex.		Slate.
1895..	29	$25,708	294	$4,459	68	$7,088	95,342	$362,592	350	$32,282	130	$1,861	$19,471
1896..	35	32,343	180	1,978	52	7,876	96,467	363,438	598	55,628	149	2,174	24,176
1897..	34	33,534	206	5,984	69	7,258	108,309	345,587	456	43,066	116	3,415	21,615
1898..	27	36,425	49	1,006	38	3,721	96,995	326,202	627	51,236	141	2,773	24,907
1899..	47	51,695	30	777	53	5,092	88,378	300,359	980	65,186	180	2,595	33,100

Year.	Sodium Compounds, Except Chloride.		Stone and Marble. (f)	Sulphur.		Tin. (f)	Ultramarine.		Whiting.	Zinc. (f)
1895.....	14,775	$347,808	$200,786	2,222	$56,965	$972,078	112	$15,252	$94,441	$99,199
1896.....	12,226	376,910	219,298	3,145	62,973	1,226,640	108	14,079	27,322	127,722
1897.....	13,988	375,651	173,822	3,982	87,712	1,272,600	124	15,044	22,541	96,708
1898.....	16,026	446,799	189,386	17,248	373,786	1,548,346	113	12,842	25,761	126,849
1899.....	20,742	476,706	237,543	11,121	265,799	1,870,239	113	12,500	34,310	154,440

MINERAL EXPORTS OF DOMESTIC PRODUCE FROM THE DOMINION OF CANADA. (a) (IN METRIC TONS AND DOLLARS.)

Year. (h)	Agric. Imple-ments.	Antimony Ore.		Asbestos.		Barytes	Brick. Thousands.		Ce-ment	Char-coal.	Cin-ders.	Clay, M'f's of.	Coal.	
1895..	$663,718	(i)	.....	7,796	$493,075	$900	1,417	$6,774	$443	$21,823	$3,163	$871	1,007,506	$3,578,195
1896..	593,464	(i)	.....	8,698	482,679	168	1,216	7,788	1,065	36,086	2,125	496	1,229,918	3,949,069
1897..	761,842	(i)	.....	9,951	510,916	(i)	906	5,296	1,332	26,980	922	796	999,749	3,330,017
1898..	1,443,140	1,118	$15,295	16,714	510,368	(i)	276	1,553	609	5,181	(i)	343	961,696	3,273,415
1899..	1,863,468	(i)	.....	13,172	453,176	(i)	98	707	2,789	6,840	50	339	1,034,963	3,562,794

Year.	Coin and Bullion.	Coke.		Copper. (e)		Explo-sives.		Fertil-izers.	Glass and Glass-ware.	Grind-stones	Gypsum.		Iron and Steel, M'f's of	Lime.
											Crude.	Ground		
1895..	256,571	24	$108	1,722	$222,657	178	$98,068	$32,174	$4,738	$12,042	145,875	$156,897	$24,556	$308,711	$63,598
1896..	207,532	17	47	1,788	194,771	120	92,763	36,137	6,881	18,853	189,217	205,641	28,332	506,946	76,451
1897..	327,298	1,692	3,802	2,933	540,439	93	76,578	36,584	7,208	15,760	163,785	183,376	18,710	522,968	55,730
1898..	1,045,923	3,275	7,341	4,203	832,546	92	74,905	46,864	7,494	18,785	163,612	193,515	2,587	606,082	43,307
1899..	1,101,245	4,024	15,684	3,812	917,429	151	115,065	51,224	11,788	18,619	148,525	166,222	7,611	706,411	64,122

Year. (h)	M'f'es of Metals Not Iron and Steel,	Mica, Crude and Cut.		Mica, Ground.	Mineral Oils. *Gallons.*		Nickel in Ore.	Ores.							
								Gold Quartz, etc.	Iron.		Lead. (e)		Manganese.	
1895..	$18,982	335	$46,894	$575	63,543	$3,572	3,647	$599,568	$612,729	4,290	$48,088	8,191	$333,763	116	$7,693
1896..	31,308	251	55,514	113	18,241	2,971	3,173	486,651	1,099,053	2,666	39,999	9,209	408,625	79	1,988
1897..	41,601	215	61,565	483	1,831	280	3,415	498,515	2,804,801	1,198	2,492	13,632	522,327	73	4,348
1898..	53,317	231	69,513	(i)	9,530	2,061	6,696	970,531	3,587,953	236	402	19,939	1,008,147	7	1,371
1899..	34,532	538	148,239	(i)	4,268	1,194	6,545	994,537	3,272,702	1,677	4,968	15,443	895,349	24	882

Year. (h)	Ores—*Continued.* Silver (d).		Phosphate of Lime.		Plumbago.		Pyrites.		Salt. Bushels.	Sand and Gravel.		Slate.		Stone and Marble.	Tin Man'f's
1895..	35	$651,737	3,800	$33,610	4	$291	7,664	$35,452	5,402	$1,186	278,294	$90,093	201	$3,964	$60,405	$9,531
1896..	78	1,595,548	479	5,280	94	10,771	6,706	32,538	3,007	767	259,585	119,847	275	8,983	49,679	5,777
1897..	127	2,613,173	21	275	78	3,584	14,219	29,408	4,702	1,168	171,057	75,134	(i)	.....	4,983	2,764
1898..	211	3,519,786	232	1,090	348	6,428	18,752	221,564	5,559	1,139	150,270	80,952	(i)	.....	61,080	5,578
1899..	187	2,630,281	781	9,290	662	12,749	11,707	27,377	5,209	1,518	162,509	93,492	(i)	.....	66,516	3,159

(a) From the tables of the *Trade and Navigation of the Dominion of Canada.* The imports figures are for home consumption only. The exports are those of domestic produce. (b) Including coal dust. (c) Represents fine copper contained in ore, matte, regulus, etc. (d) Silver contents of ores. (e) Represents lead contained in ore. (f) Includes manufactures. (g) Represents barrels. (h) Fiscal years ending June 30. (i) Not reported. (k) Includes alumina.

MINERAL PRODUCTION OF BRITISH COLUMBIA. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Coal.		Coke.		Copper.		Gold—Kg. Lode.		Gold—Kg. Placer.		Lead. (b)		Silver—Kg.	
1895	954,688	$2,818,062	459	$2,260	432	$47,642	1,221	$785,271	749	$481,688	7,473	$532,255	46,546	$977,229
1896	910,562	2,688,066	625	3,375	1,732	190,926	1,936	1,244,180	846	544,026	10,977	721,384	97,519	2,100,689
1897	896,980	2,648,562	18,116	89,155	2,415	266,258	3,301	2,122,820	796	513,520	17,618	1,390,517	170,227	3,272,836
1898	1,154,039	3,407,595	35,560	175,000	3,298	874,781	3,423	2,201,217	1,600	643,346	14,376	1,077,581	133,200	2,875,841
1899	1,314,888	3,882,396	34,799	171,255	3,503	1,351,453	4,302	2,857,573	2,092	1,344,900	9,917	878,870	91,425	1,663,708

(a) From the *Annual Reports of the Minister of Mines.* (b) Attention should be called to the discrepancy between certain of these figures and those of the Canadian Geological Survey; practically the total production of lead in the Dominion is from British Columbian ore.

MINERAL PRODUCTION OF NOVA SCOTIA. (a) (d) (IN METRIC TONS AND DOLLARS.)

Year. (c)	Coal.	Coke.	Copper	Gold. Kg.	Grindstones.	Gypsum.		Iron Ore.	Limestone.	Manganese Ore.		Moulding Sand.
1895.....	2,122,672	42,160	(b)	626	$17,189	120,929	(b)	72,226	27,375	99	(b)	68 .....
1896.....	2,271,140	59,681	10	812	30,317	131,577	$122,614	57,235	31,670	131	$3,798	....
1897.....	2,358,051	45,720	(b)	827	32,400	126,500	(b)	48,852	25,300	102	(b)	....
1898.....	2,317,957	42,672	(b)	967	38,000	133,096	.....	31,547	24,334	76	.....	....
1899.....	2,264,610	56,372	(b)	864	50,000	142,240	.....	16,428	32,512	102	.....	(b) .....

(a) From the *Annual Reports of the Department of Mines of Nova Scotia.* (b) Not stated in the official reports. (c) For the fiscal year ending September 30. (d) There was also produced, in 1896, 178 net tons of graphite; in 1899, 400 net tons of copper ore, 835 tons barytes and 898 tons of tripoli and silicia.

MINERAL PRODUCTION OF ONTARIO. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Brick—*Number.* Common.		Pressed. Plain.		Pressed. Fancy.		Building Stone. Dimension Stone. *Cubic Feet.*		Heads and Sills—*Cu. Ft.*		Coursing Stone—*Sq. Yds*	
1895....	126,245,000	$705,000	15,258,370	$115,695	2,312,497	$24,075	(d)	.....	(d)	.....	(d)	.....
1896....	105,000,000	577,000	10,774,400	88,945	1,256,600	9,900	(d)	.....	(d)	.....	(d)	.....
1897....	(g)	.....	7,148,908	53,727	895,000	9,350	(g)	.....	(g)	.....	(g)	.....
1898....	170,000,000	914,000	7,754,868	87,894	(b)	.....	(d)	.....	(d)	.....	(d)	.....
1899....	233,989,000	1,313,750	10,808,000	105,000	(b)	.....	(d)	.....	(d)	.....	(d)	.....

Year.	Building Stone. *Continued.*		Cement—*No. of Barrels.*				Cobalt. Metric Tons		Copper. Metric Tons.		Gold.		
	Rubble, etc. *Cubic Yards.*	Total Value.	Natural Rock.		Portland.						Ore. Met. Tons.	Metal.	
												Kg.	Value.
1895....	(*d*)	.....	$428,000	55,219	$45,145	58,699	$114,332	(*g*)	.....	2,146	$160,913	5,896	94·2	$50,281
1896....	(*d*)	.....	394,000	60,705	44,100	77,760	188,230	(*g*)	.....	1,695	130,660	12,059	222·5	121,848
1897....	(*g*)	.....	.....	84,670	76,123	96,695	170,308	(*g*)	.....	2,495	200,067	(*g*)	354·9	190,244
1898....	(*d*)	.....	750,000	91,528	74,222	153,348	302,096	(*g*)	.....	3,798	268,080	(*g*)	505·8	275,078
1899....	(*d*)	.....	1,041,350	139,487	117,089	222,550	444,227	(*g*)	.....	2,571	176,287	(*g*)	1,164·8	419,328

Year.	Gypsum, Crude and Calcined. Metric Tons.		Iron.				Lime—*Bushels.*		Mica. Metric Tons.		Natural Gas. *1,000 Cubic Feet.*		Nickel. Metric Tons.	
			Ore.		Pig.									
1895....	3,461	$20,566	*Nil.*	.....	*Nil.*	.....	2,000,000	$280,000	23·6	$2,900	3,320,000	$282,986	2,100	$404,961
1896....	3,810	20,750	(*g*)	.....	25,675	$253,780	1,890,000	220,000	2·3	2,495	(*g*)	276,710	1,768	357,000
1897....	1,568	17,905	2,507	$4,689	21,783	266,128	(*g*)	.....	(*g*)	.....	(*g*)	308,448	1,813	359,651
1898....	2,722	4,000	24,865	48,875	43,775	530,789	2,620,000	308,000	31	7,500	(*g*)	301,600	2,071	514,220
1899....	1,089	16,512	15,342	30,951	58,740	808,157	4,342,500	535,000	241	88,000	(*g*)	440,904	2,605	526,104

Year.	Petroleum.									
	Crude. *Imperial Gallons.* (*h*)		Refined Products. (*e*)							
			Illuminating Oils. *Imp. Gals.*		Lubricating Oils. *Imp. Gals.*		All Other Oils. *Imp. Gals.* (*c*)		Paraffine Wax. Metric Tons.	(*e*) Fuel Products
1895....	33,351,997	$1,403,960	10,994,826	$1,237,328	2,400,404	$305,591	7,061,717	$285,308	891·0	$36,608	$79,569
1896....	27,380,588	1,222,307	11,842,880	1,263,290	2,283,047	204,946	7,821,262	340,054	695·2	76,250	70,815
1897....	25,556,591	(*g*)	10,891,337	1,131,083	1,959,810	199,755	949,341	77,340	970·4	88,378	281,035
1898....	26,978,977	(*g*)	12,281,322	1,243,490	2,043,226	202,150	1,340,967	121,840	1,186·6	116,349	286,705
1899....	23,615,967	(*g*)	11,697,910	1,059,485	2,087,475	189,294	1,394,580	148,963	1,266·8	136,066	213,544

Year.	Pottery.	Salt. Metric Tons.		Sewer Pipe.	Silver.		Terra Cotta.	Tiles—*Thousands.*				Total Value. (*f*)
					Kg.	Value.		Drain.		Roofing.		
1895....	$108,000	46,275	$188,101	$183,159	*Nil.*	.....	$38,500	14,330	$157,000	375	$6,200	$5,170,138
1896....	104,000	40,657	204,910	49,875	*Nil.*	.....	24,190	18,200	144,000	170	6,800	5,235,003
1897....	(*g*)	49,611	249,880	73,551	*Nil.*	.....	25,800	(*g*)	.....	(*g*)	73,551	3,899,221
1898....	155,000	53,874	278,896	93,717	2,693·5	$51,960	(*b*)	22,668	225,000	(*g*)	.....	7,201,891
1899....	101,000	51,143	317,412	138,356	3,280·4	65,575	(*b*)	21,027	240,246	(*g*)	.....	8,785,251

(*a*) From the *Annual Reports of the Bureau of Mines of Ontario*, Toronto. In 1894 there was also produced sand and gravel 733,500 cubic yards, value $293,450. (*b*) Quantity and value included with that of plain pressed brick. (*c*) In 1897, 1898 and 1899 comprises benzine and naphtha. (*d*) Quantity not stated, and value included in the total of building stone. (*e*) In 1897, 1898 and 1899, comprises gas and fuel oils and tar. (*f*) In 1898 total includes 521 metric tons carbide of calcium ($34,440) and 272 metric tons graphite ($6,000). In 1899, 5,300,000 paving brick ($42,550); 965 metric tons carbide of calcium ($74,680); 91 metric tons of talc ($500); 1,107 metric tons of graphite ($16,179); 1,089 metric tons of zinc ($24,000); and 52 metric tons of arsenic ($4,842). (*g*) Not reported. (*h*) One barrel of crude oil is reckoned at 35 imperial gallons, and 1 barrel refined at 42 imperial gallons.

CHILE.

THERE are no official statistics of mineral production in Chile. The exports and imports are summarized in the subjoined table. With respect to the most important articles of mineral production, namely, copper and nitrate of soda, the exports practically represent the production. This is also the case with respect to iodine and borate of lime.

MINERAL EXPORTS OF CHILE. (*a*) (IN METRIC TONS AND CHILEAN DOLLARS.)

Year.	Borate of Lime.		Borax.		Clay.		Coal.		Cobalt Ore.		Copper Matte.	
1894....	6,700	$335,008	(*b*)	.....	(*b*)	.....	205,201	$1,641,608	4·6	$230	342	$34,220
1895....	4,425	221,263	107	$21,380	(*b*)	.....	196,115	1,560,920	13·4	915	417	41,673
1896....	7,486	374,294	(*b*)	.....	40	$600	204,858	1,434,253	(*b*)	.....	2,528	284,897
1897....	3,154	283,770	14	2,860	20	998	243,968	1,869,310	6·0	312	2,519	251,915
1898....	7,028	1,124,509	6	3,391	(*b*)	.....	282,663	4,239,943	19·2	1,817	3,079	861,999

Year.	Copper and Silver Matte.		Copper, Silver and Gold Matte		Copper Ore.		Copper and Silver Ore.		Copper, Silver and Gold Ore—Kg.		Copper, in Bars.	
1894....	1,508	$245,223	2·5	$503	11,106	$444,227	90·3	$11,027	460	$53	19,640	$4,372,238
1895....	664	134,424	15·3	3,060	6,963	278,515	84·4	8,849	2,012	402	20,042	4,602,114
1896....	1,059	213,115	7·6	1,520	6,159	307,943	62·3	9,462	29.542	2,954	20,592	5,148,015
1897....	904	378,340	(*b*)	.....	3,396	169,810	161·8	25,455	(*b*)	.....	19,011	5,226,199
1898....	419	167,556	17·8	29,133	20,301	2,022,730	418·9	167,556	5,733	1,300	20,600	13,759,498

Year.	Earth for Smelting Furnaces. Kg.		Gold Bullion. Kg.		Gold Ore.		Iodine.		Iron Ore.		Lead and Argentiferous Lead, in Bars.	
1894....	53,155	$1,066	1,475·4	$1,172,355	192	$39,243	223	$3,332,780	(*b*)	.....	87	$6,073
1895....	(*b*)	.....	1,184·5	247,749	270	54,068	144	1,442,580	305	$15,255	93	4,627
1896....	(*b*)	.....	1,061·3	849,036	367	73,475	206	2,063,950	(*b*)	.....	594	59,415
1897....	8,200	82	1,181·7	906,168	64	28,535	243	2,429,370	5·1	256	369	66,696
1898....	30,440	1,218	1,630·5	2,445,735	24	2,488	285	3,169,570	(*b*)	.....	13	13,596

Year.	Lime.		Manganese Ore.		Mineral Specimens.	Chile Saltpeter.		Silver Ore.		Silver and Gold Ore.	
1894....	7·4	$148	47,994	$479,989	$1,150	1,081,337	$44,297,513	370	$401,449	56	$12,383
1895....	1·2	26	24,075	241,388	2,800	1,220,427	45,528,510	2,137	1,051,034	113	28,255
1896....	4·0	80	26,151	261,515	700	1,111,757	43,931,667	2,750	1,257,870	666	130,182
1897....	0·8	24	23,529	1,411,648	20,300	1,057,640	37,461,559	984	434,231	260	46,631
1898....	0·7	21	20,851	447,028	1,400	1,294,227	90,675,297	284	205,386	269	57.635

Year.	Silver, in Bars, etc. Kg.		Silver, Old. Kg.		Silver-Lead Ore.		Silver Sulphuret Ore.		Tin.	
1894....	153,723	$3,857,481	2,175	$43,556	15	$1,000	127	$165,098	4·2	$1,266
1895....	148,747	3,867,412	375	7,494	21	2,050	99	123,155	(b)	.....
1896....	151,226	4,183,108	966	19,754	0·8	83	160	418,785	(b)	.....
1897....	143,541	3,781,125	443	12,518	6·0	636	183	338,062	48	54,613
1898....	139,756	6,987,816	55	2,219	11·8	1,557	290	434,740	8	4,900

(a) From the *Estadística Commercial de la República de Chile*, Valparaiso. It will be noticed that the values reported vary widely for the different years. This is probably to be referred to the fluctuations of the currency standard. (b) Not reported.

MINERAL IMPORTS OF CHILE. (a) (IN METRIC TONS AND CHILEAN DOLLARS.)

Year.	Bitumen for Pavements.		Brass.		Cement, Roman.		Coal.		Copper.			
									Sheets.		For Various Uses	
1894....	118	$25,600	86	$41,914	12,302	$252,314	595,639	$4,752,966	37	$20,151	31	$18,366
1895....	190	38,901	82	36,776	18,226	365,550	574,063	4,578,067	18	9,780	57	15,473
1896....	254	52,198	118	50,518	14,431	326,395	683,196	5,462,954	16	9,484	54	29,853
1897....	237	55,892	60	26,206	18,205	365,129	513,421	4,122,918	55	28,333	43	23,824
1898....	83	52,473	.....	.....	16,905	676,199	656,202	13,173,965	57	44,002	.....	.....

Year.	Copper, Sulphate.		Earth for Smelting Furnaces.		Gold Coin.	Iron.							
						Bars and Ingots		Hoops.		Sheets, Ungalv.		Sheets, Galv.	
1894....	253	$36,275	88	$2,422	$179,595	14,397	$678,305	959	$62,012	4,113	$333,032	10,451	$1,260,622
1895....	134	18,855	177	3,980	392,900	16,506	772,723	1,198	84,794	4,714	375,848	5,972	719,721
1896....	111	16,643	69	1,386	5,635	13,970	698,611	1,452	101,605	1,256	99,985	7,917	942,449
1897....	85	10,737	100	3,542	4,440	7,604	354,592	802	63,943	1,859	153,732	10,756	1,292,176
1898....	218	65,358	30	1,785	*Nil.*	10,223	1,256,190	404	54,268	2,283	282,353	9,947	1,377,343

Year.	Lead, Bars and Sheets.		Quicksilver.		Salt.				Silver.				
					Common.		Refined.		Bars—Kg.		Coin.	Ore.	
1894....	96	$15,117	142	$144,564	5,417	$39,789	109	$8,353	.....	.....	$89,941	66	$15,557
1895....	194	27,750	106	106,277	3,192	23,042	246	12,116	17·3	$954,255	112,430	58	21,383
1896....	525	73,519	54	54,350	7,311	53,496	353	19,077	.....	.....	89,853	40	14,640
1897....	261	37,832	40	39,758	13,179	90,742	320	16,191	.....	.....	36,930	9	3,030
1898....	118	28,389	15	44,435	1,543	30,865	101	10,111	.....	.....	*Nil.*	21	5,455

Year.	Soda, Caustic.		Steel, Bars and Sheets.		Sulphur.		Tin.		Whiting and Gypsum.		Zinc.			
											Bars.		Sheets.	
1894....	1,433	$85,475	3,198	$382,223	2,291	$136,596	447	$61,936	532	$11,973	74	$9,714	353	$45,714
1895....	2,111	123,102	7,142	857,400	1,679	100,961	804	113,046	796	20,674	7	944	309	55,830
1896....	2,570	149,222	3,480	422,078	1,775	110,742	506	71,648	1,492	47,831	3	426	315	45,451
1897....	2,604	155,026	2,225	268,181	2,655	157,671	608	82,149	492	13,668	24	3,314	252	50,936
1898....	2,796	341,760	1,369	166,268	3,317	331,725	51	58,173	466	22,028	73	18,213	282	68,398

(a) From the *Estadística Commercial de la República de Chile*, Valparaiso.

CHINA.

THERE are no official statistics of mineral production in China. The exports and imports as reported by the Imperial Maritime Customs officials are summarized in the following tables:

MINERAL IMPORTS OF CHINA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Brass Wire.		Cement.		Chinaware, Fine and Coarse.		Coal.		Colors.		Copper. Bar, Rod, Sheets, Plates, Nails.	
1894....	217·6	$50,080	9,950·5	$123,694	514·0	$70,586	494,075	$2,480,434	1,529·5	$124,010	811·7	$178,633
1895....	159·8	38,343	(*b*)	105,943	203·3	37,518	581,590	2,715,418	2,675·6	122,542	589·5	105,414
1896....	193·5	46,780	(*b*)	170,041	403·1	53,083	634,912	2,867,241	3,153·7	283,470	539·3	123,741
1897....	133·6	33,915	(*b*)	79,056	401·3	109,119	558,149	2,658,722	2,135·6	133,723	499·6	108,317
1898....	183·2	46,116	(*b*)	333,947	379·1	92,522	742,296	3,696,434	(*b*)	130,359	393·7	84,244

Year.	Copper—*Continued.* Slabs and Ore, Unmanufactured		Wire.		Wares, Unclassified.		Flint Stones.		Glass. Window—*Boxes.*		Wares.
1894....	1,596·9	$296,917	91·8	$21,994	73·5	$21,702	1,642·3	$13,011	103,470	$194,996	$102,117
1895....	990·3	224,287	69·9	18,915	102·4	27,980	2,261·7	17,897	80,881	165,335	185,748
1896....	2,285·4	511,674	113·8	29,985	99·4	40,589	2,585·4	33,123	111,586	281,988	278,652
1897....	2,525·8	606,602	79·9	20,743	34·4	13,975	2,652·9	30,707	138,553	335,582	238,784
1898....	1,817·7	429,702	80·6	21,718	23·0	8,382	2,868·7	32,339	93,409	222,611	247,538

Year.	Iron. Bar.		Hoop.		Nail-Rod.		Old.		Pig and Kentledge.		Sheets and Plates.	
1894....	7,565·4	$234,473	1,455·4	$53,775	20,856	$573,895	20,810	$540,846	5,453·4	$102,694	3,442·7	$157,716
1895....	8,745·9	309,540	1,401·2	43,544	16,358	480,478	25,009	419,298	7,378·5	138,497	3,664·4	184,394
1896....	9,881·5	314,737	1,949·4	81,797	26,677	761,351	52,435	946,887	8,752·3	155,980	4,861·1	229,416
1897....	6,867·9	209,254	1,480·8	54,396	11,611·4	353,935	32,881	587,278	1,456·5	36,678	3,489·6	164,309
1898....	11,309·6	352,061	1,923·3	70,663	21,253 6	643,671	43,018	798,206	10,179·2	280,540	4,709·2	215,454

Year.	Iron—*Continued.* Wire.		Wares, Unclassified	Jadestone.		Lead. Pig.		Tea and Sheet.		Machinery
1894....	2,078·1	$114,532	$123,185	152·8	$90,402	12,799	$720,873	120·5	$9,110	$862,228
1895....	2,214·3	133,506	115,700	272·0	225,497	10,858	646,258	122·3	8,199	1,908,164
1896....	2,991·3	173,474	421,083	264·2	216,728	10,114	622,690	248·1	17,328	1,672,197
1897....	2,529·1	163,455	299,645	227·5	174,467	7,709	494,721	102·2	7,271	1,956,051
1898....	2,968·0	184,461	334,550	249·5	135,311	8,832	587,556	101·2	7,500	1,231,081

Year.	Metals, Unclassed.	Nickel.		Paints.		Petroleum—*Liters.* (Refined.)		Quicksilver.		Spelter.	
1894....	$253,737	38·6	$29,855	2,025	$398,639	316,674,095	$6,164,092	67·5	$59,543	570·5	$42,231
1895....	322,425	120·3	93,220	(*b*)	420,138	295,319,349	5,220,238	73·7	69,723	190·3	14,095
1896....	273,153	104·1	67,634	(*b*)	624,690	304,190,319	7,357,490	91·0	101,680	1,116·0	97,020
1897....	197,659	37·0	23,334	(*b*)	457,608	451,387,582	9,575,378	55·8	54,023	1,819·5	169,636
1898....	247,245	67·9	43,393	(*b*)	513,659	366,728,870	8,340,289	70·4	67,481	623·1	71,684

Year.	Steel.				Tin.				White Metal. (German Silver)		Yellow Metal. Bar, Rod, Sheets, and Nails.	
	Common.		Mild, or Iron Ingots.		Plates.		Slabs.					
1894....	3,575·7	$212,699	3,115·9	$95,351	1,429·0	$81,105	4,435·2	$1,495,009	172·3	$83,125	1,537·4	$243,734
1895....	2,766·0	168,062	3,573·4	118,394	3,818·0	195,276	4,590·3	1,540,592	139·7	74,793	1,569·4	236,978
1896....	5,181·9	369,762	9,758·9	220,547	1,890·7	123,901	5,516·4	1,785,541	182·1	82,953	1,719·3	305,514
1897....	3,206·9	198,455	711·4	22,223	3,543·7	209,447	4,128·9	1,396,303	241·4	122,901	1,753·6	317,137
1898....	2,908·7	183,523	9,109·9	282,763	6,775·6	398,690	3,196·5	1,131,619	230·4	116,859	1,678·4	297,903

RE-EXPORT OF FOREIGN GOODS FROM CHINA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Cement.		Chinaware.		Coal.		Colors.		Copper, Bar, Rod Sheets, Plates and Nails.		Glass.		
											Window. *Bores.*		Wares.
1894....	(*b*)	.....	(*b*)	.....	47,950	$254,061	10·8	$2,906	3·06	$298	758	$1,354	(*b*)
1895....	(*b*)	.....	(*b*)	.....	48,889	284,380	143·7	18,265	3·75	921	1,046	2,018	$1,394
1896....	11·6	$156	1·3	$308	42,092	271,957	31·8	7,989	(*c*) 5·20	1,528	222	580	2,252
1897....	147·0	1,935	0·1	123	47,254	233,561	33·8	8,578	5·06	1,008	1,024	2,580	1,992
1898....	122·4	2,068	0·1	81	55,238	284,501	(*b*)	5,880	(*d*) 1·98	517	1,378	3,459	2,675

Year.	Iron.								Lead—In Pigs.		Metals, Unclassed.
	Bar, Hoop, Nail-Rod, Sheets, Plates, and Wire.		Old.		Pig and Kentledge.		Wares, Unclassed.				
1894....	704·0	$39,738	170·1	$2,604	$150·6	2,878	$3,351		18·7	$1,071	$3,351
1895....	1,698·8	64,041	210·1	3,743	152·0	2,815	484		80·5	4,806	14,790
1896....	1,412·8	58,733	334·5	7,197	154·4	2,524	4,544		98·4	6,890	10,385
1897....	1,601·5	69,555	258·5	5,098	1,181·6	33,774	6,340		68·6	4,961	5,650
1898....	1,155·9	57,249	342·5	7,467	707·1	19,650	11,000		72·7	5,384	12,693

Year.	Nickel.		Paints.		Petroleum—*Liters.* (Refined.)		Quicksilver.		Spelter.		Steel.	
1894....	(*b*)	.....	34·4	$11,925	500	$8	(*b*)	.....	(*b*)	.....	29·3	$1,761
1895....	1·269	$1,005	(*b*)	5,506	2,262,668	72,576	8·6	$7,709	(*b*)	.....	173·6	7,603
1896....	1·330	986	(*b*)	11,985	2,302,730	61,567	3·7	4,048	(*b*)	.....	377·7	15,733
1897....	3·443	1,874	(*b*)	15,996	1,992,607	42,623	2·6	2,639	12·08	$1,132	591·0	20,695
1898....	4·654	2,952	(*b*)	15,016	2,585,556	59,441	4·1	4,026	(*b*)	.....	780·6	28,179

RE-EXPORT OF FOREIGN GOODS FROM CHINA. *Concluded.*

Year.	White Metal. (German Silver)		Tin, in Slabs and Tinplates.		Yellow Metal, Bar, Rod, Sheets and Nails.	
1894....	(*b*)	.....	344·6	$43,240	24·6	$4,261
1895....	7·1	$3,596	135·4	19,362	13·2	2,946
1896....	(*b*)	.....	45·2	5,871	9·1	446
1897....	1·3	711	271·1	32,430	0·2	52
1898....	0·2	91	127·5	27,999	13·6	2,998

EXPORT OF NATIVE GOODS FROM CHINA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Chinaware, Earthenware, and Pottery.		Glassware, Bangles, etc.		Gold and Silver Ware.	
1894...	18,663	$947,975	1,347·8	$304,160	2·176	$74,340
1895...	32,788	1,282,906	1,510·6	353,929	3·143	95,697
1896...	21,786	1,318,955	1,111·9	316,891	3·063	120,149
1897...	20,453	996,722	1,066·9	252,573	2·597	93,575
1898...	20,381	1,053,015	1,261·9	300,919	3·325	118,580

(*a*) From the *Returns of Trade and Trade Reports of the Imperial Maritime Customs*, Shanghai, China. In these reports the unit of quantity is chiefly the "picul," and that of value the "Haikwan-tael." In converting the original data to metric tons and dollars, the following relations were used: 1 picul = 0·060453 metric ton; in 1894, 1 Haikwan-tael = 77c. American gold: in 1895, 80c.; in 1896, 81c.; in 1897, 72c.; and in 1898, 70c., at the average sight exchange on New York, London, Paris, Berlin, Calcutta, and Hong-Kong respectively. (*b*) Not reported. (*c*) In 1896 there was also exported 111·2 metric tons of copper (unmanufactured, slabs and ore) valued at $31,966. (*d*) In 1898 there was also exported 4·41 metric tons of unclassed copperware valued at $1,821.

FRANCE.

THE official statistics of mineral production in France and her Colonies are summarized in the subjoined tables. With respect to the most important substances, reference should be made to the respective captions elsewhere in this volume.

MINERAL PRODUCTION OF FRANCE. (*a*) (IN METRIC TONS; 5 f.—$1.)

Year.	Alunite.		Antimony Ore.		Asphaltum.		Bauxite.		Bituminous Substances. (*c*)		Cement.	
1894..	354	$3,540	6,144	$81,231	14,336	$129,550	26,032	$32,588	230,603	$351,561	(*b*)	.....
1895..	330	3,300	5,396	68,004	*e*15,705	138,055	17,958	24,282	266,660	368,632	838,739	$3,995,733
1896..	171	1,710	5,675	68,544	17,717	150,263	33,820	48,833	225,784	348,187	934,624	4,556,503
1897..	(*b*)	.....	4,685	64,915	17,982	146,272	41,740	55,211	233,328	339,898	976,813	4,716,471
1898..	(*b*)	.....	4,433	65,046	18,832	153,758	36,723	67,133	229,108	333,888	1,072,025	5,359,620

Year.	Copper Ore.		Gypsum.				Iron Ore.		Lead-Silver Ore.		Manganese Ore.	
			Crude.		Calcined.							
1894..	(*b*)	.....	(*b*)	.....	*g*1,537,052	$2,996,233	3,772,101	$2,459,461	29,055	$473,920	32,751	$200,275
1895..	(*b*)	.....	85,885	$129,771	1,888,236	3,386,051	3,679,767	2,341,711	21,503	448,835	30,871	184,143
1896..	106	$704	264,187	241,608	1,429,550	2,516,112	4,069,390	2,565,205	19,042	450,709	31,318	185,717
1897..	956	3,645	292,753	264,845	1,369,269	2,458,337	4,582,236	3,008,028	21,212	556,261	37,212	208,035
1898..	382	2,841	303,531	281,690	1,449,384	2,596,870	4,731,394	3,207,447	23,342	640,226	31,935	166,211

Year.	Millstones.		Mineral Fuel.						Petroleum, Crude.		Phosphate Rock.	
			Coal.		Lignite.		Peat.					
1894..	(*b*)	.....	26,964,125	$60,747,709	452,780	$804,321	131,717	$356,606	8,762	$192,573	539,000	$3,311,142
1895..	33,396	$233,762	27,582,819	60,930,220	437,074	779,554	131,547	366,429	9,446	215,002	526,784	3,260,599
1896..	28,337	224,347	28,750,452	62,516,853	439,448	776,938	130,207	379,047	9,292	209,184	582,667	3,502,028
1897..	32,175	244,063	30,337,207	66,020,171	460,422	781,517	98,067	253,851	10,283	234,811	595,390	2,854,887
1898..	*h*38,929	727,235	31,926,127	71,701,384	529,977	929,300	104,265	301,486	9,694	221,172	568,558	3,115,958

Year.	Pyrites.		Salt.		Slate.				Sulphur Ore. (*d*)		Zinc Ore.	
					Roofing.		Slabs.					
1894..	283,439	$684,906	890,607	$2,787,624	*h*563,608	$2,723,772	(*b*)	.....	851	$2,747	80,065	$825,867
1895..	253,416	699,414	871,312	2,406,155	291,444	3,191,646	2,103	$32,364	4,213	14,077	72,989	873,765
1896..	282,064	712,996	1,042,614	2,473,568	283,352	3,154,551	1,148	31,006	9,720	25,660	81,346	1,160,021
1897..	308,488	752,658	948,008	2,239,624	310,920	3,620,949	1,143	31,282	10,723	31,252	83,044	1,305,003
1898..	310,972	785,242	999,283	2,115,120	316,911	4,118,789	1,318	38,552	9,818	27,141	85,550	1,475,581

METALLURGICAL PRODUCTION OF FRANCE. (a) (IN METRIC TONS; 5 f.—$1.)

Year.	Aluminum.		Antimony.		Copper.		Gold—Kg.		Iron, Pig.		Iron, Wrought.	
1894....	270	$274,400	1,012	$136,024	6,415	$1,353,574	376	$240,640	2,069,714	$23,600,457	785,781	$25,794,576
1895....	360	360,000	779	98,734	8,245	1,872,740	380	261,744	2,008,988	22,264,312	756,798	23,443,660
1896....	370	259,000	969	130,217	6,544	1,641,876	327	225,537	2,339,537	26,476,584	828,758	25,343,590
1897....	470	285,500	1,083	141,857	7,376	1,845,243	276	190,109	2,484,191	29,121,027	584,540	24,406,664
1898....	565	301,800	1,226	163,200	7,834	2,270,200	267	183,900	2,525,100	31,858,400	766,000	25,221,800

Year.	Steel. (l)		Lead. (f)		Litharge.		Nickel.		Silver—Kg.		Zinc.	
1894.....	774,190	$33,895,860	8,696	$962,882	62	$2,994	1,545	$1,237,500	96,955	$2,133,010	23,387	$1,815,439
1895.....	714,523	36,645,671	7,627	895,422	(b)	.....	1,545	1,087,500	71,073	1,563,606	24,230	1,781,515
1896.....	916,817	41,896,067	8,232	444,371	(b)	.....	1,545	921,500	70,479	1,592,825	35,585	2,922,628
1897.....	994,891	45,276,658	9,916	630,928	(b)	.....	1,245	741,500	80,351	1,639,160	38,067	3,268,490
1898.....	1,174,000	55,042,800	10,920	788,400	(b)	.....	1,540	920,000	90,588	1,817,000	37,155	3,598,600

(a) From *Statistique de l'Industrie Minérale.* (b) Not reported. (c) Includes pure bitumen, bituminous schist, bituminous sand, and asphaltic limestone. (d) Sulphur and limestone impregnated with sulphur. (e) Including briquettes. (f) Lead produced from native ores only, and does not include the metal produced from foreign ores and bullion. (g) Cubic meters. (h) Thousands. (k) Finished product. (l) Wrought steel.

MINERAL IMPORTS OF FRANCE. (a) (IN METRIC TONS; 5 f.—$1.)

Year.	Bituminous Substances. (b)		Coal. (c)		Copper.		Gold.	Silver.	Iron and Steel.		Cast Iron, Crude.	
1895..	43,975	$879,600	10,261,069	$33,363,400	38,196	$8,859,400	$50,775,089	$28,280,595	66,240	$1,619,400	36,947	$561,200
1896..	30,954	495,200	10,180,449	34,733,600	46,830	12,992,400	60,167,745	45,951,637	48,423	1,712,000	18,323	368,800
1897..	29,931	419,000	10,457,255	37,894,400	54,460	14,375,400	58,143,077	42,468,793	60,804	1,954,800	35,633	944,400
1898..	20,385	244,600	10,445,090	41,438,200	52,976	14,998,000	39,881,575	38,274,504	47,395	1,941,900	.....	998,000
1899..	30,770	369,200	11,896,080	46,995,000	58,419	16,574,400	63,697,020	37,528,657	64,178	2,660,400	.....	1,487,800

Year.	Lead.		Nickel.		Nitrate of Soda.	Stone.	Sulphur.		Tin.		Zinc.	
1895..	66,241	$4,626,800	252	$300,600	$8,624,200	$3,144,200	110,989	$1,598,400	7,691	$2,645,800	25,652	$1,998,000
1896..	79,752	5,751,600	425	374,800	9,025,400	2,440,000	111,515	1,965,400	8,400	2,820,400	33,459	3,060,800
1897..	86,589	6,661,800	316	475,400	8,105,400	2,675,800	136,118	2,602,200	7,642	2,572,000	31,211	2,998,800
1898..	74,902	5,843,600	330	525,000	8,026,400	2,814,600	130,289	2,800,400	9,247	3,475,800	32,342	3,632,400
1899..	67,149	5,310,200	286	414,600	9,341,600	3,511,000	120,062	2,689,000	6,907	2,607,200	25,516	2,955,400

Year.	Alum.	Antimony Ore.	Borax.	Bromides.	Carbonate of Lead.	Cement	Chlorhydric Acid.	Chloride of Lime.	Chloride of Potash.	Chromate of Potash and of Soda.	Copper Ore.	Gold Ore. Kg.
1895..	109	.....	442	12	1,077	13,441	2,777	1,017	3,524	2,875	10,450	58,811
1896..	41	.....	255	18	892	14,395	2,468	2,083	11,499	2,838	8,584	e101,499
1897..	54	(d)	264	18	1,327	15,141	2,119	1,718	11,630	2,852	11,960	(d)
1898..	27	(d)	139	30	1,376	11,290	1,994	1,288	10,929	2,890	8,779	(d)
1899..	34	(d)	128	46	2,029	13,640	1,905	1,887	13,335	3,147	8,517	(d)

Year.	Iron.		Kaolin.	Lead Ore.	Lime.	Manganese Ore.	Mercury	Nitrate of Potash.	Nickel Ore.	Nitric Acid.
	Ore.	Pyrites.								
1895..	1,651,369	67,930	359	5,032	246,677	414	178	775	10,303	702
1896..	1,862,043	45,788	38,703	5,569	283,707	616	234	2,614	15,756	684
1897..	2,187,860	69,470	42,384	13,961	321,647	855	248	1,309	17,441	774
1898..	2,032,240	71,569	40,352	14,377	346,000	1,002	221	1,008	24,965	922
1899..	1,950,665	109,696	36,904	12,570	321,610	1,066	276	1,015	28,690	1,222

Year.	Oxides.						Petroleum.	Phosphates.	Plaster.	Platinum. Kg.	Potash and Carbonate of.	Sal Ammoniac.	Salt.
	Cobalt	Copper.	Iron.	Lead.	Uranium.	Zinc.							
1895..	5	24	855	1,931	2	802	258,700	139,600	2,412	926	796	9,923	17,528
1896..	5	22	897	1,984	4	988	272,693	256,888	1,774	2,117	1,526	15,256	17,191
1897..	9	29	1,125	1,759	5	1,114	288,671	313,608	1,869	1,069	1,769	27,454	32,917
1898..	9	52	1,021	1,874	15	1,256	291,961	336,842	2,040	505	2,418	20,426	35,863
1899..	9	36	1,037	1,807	44	1,765	306,078	242,021	2,260	817	2,779	12,210	37,970

Year.	Silver Ore. Kg.	Soda. Caustic	Sulphates.		Sulphide of Mercury.	Sulphuric Acid.	Superphosphate of Lime.	Tin Ore.	Zinc Ore.
			Copper.	Iron.					
1895.....	*f*524,394	1,021	24,404	3,882	23	3,461	150,758	104	41,622
1896.....	*f*1,818,650	1,109	33,803	3,086	25	3,995	185.602	7	50,899
1897.....	(*d*)	1,378	30,132	1,353	24	3,147	195,853	149	58,074
1898	(*d*)	1,772	30,897	896	19	4,666	178.569	357	60,481
1899.....	(*d*)	1,494	21,733	1,698	21	4,583	171,631	486	78,192

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (*a*) (IN METRIC TONS.)

Year.	Aluminum.	Antimony.		Cement	Coal.	Copper.		Gold. Kg. (*e*)	Iron.				Iron Pyrites
		Ore.	Metal			Ore.	Metal		Ore.	Pig.	Bars.	Steel.	
1895...	110	832	68·2	.....	.....	1,772	8,829	1,353	236,923	150,540	.....	29,074	37,968
1896....	793	736	73·8	242,247	1.044,820	1,261	10,494	2,193	238,430	195,212	24,721	44,795	41,222
1897....	234	623	61·4	244,504	1,142,195	2,000	12,667	3,335	299,589	108,645	39,894	45,809	54,367
1898....	192	616	100·6	241,150	1,320,616	1,788	14,850	1,812	236.169	102,991	27.424	47,278	60.406
1899....	256	304	254·7	244,480	1,229,090	2,078	17,949	2,622	291,846	153,792	29,112	33,584	53,395

Year.	Lead.		Manganese Ore.	Millstones and Pieces. *Number.*	Nickel. Refined	Phosphate Rock.	Plaster.	Silver. Kg. (*f*)	Tin.	Zinc.	
	Ore.	Metal.								Ore.	Metal.
1895....	8,670	8,087	16,198	.....	408	.....	.....	13,567	650	61,291	5,849
1896....	8,597	10,856	10,913	196,685	490	48,719	89,952	9,849	744	62.415	10,485
1897....	12,007	10,364	19,464	158,979	498	69,188	107,823	5,374	651	79,909	10,977
1898....	10,216	8,663	12,229	203,584	526	98,742	106,790	1,886	587	60,664	16,995
1899 ...	8,909	1,163	12,289	112,620	280	70,517	112,590	.....	666	76,104	14,958

(*a*) The figures are from *L'Economiste Français*, and represent the *Commerce Spécial* of France. (*b*) Includes bitumen, bituminous schist and sand, and asphaltic limestone. (*c*) Including coke. (*d*) Not reported. (*e*) Gold and platinum, in ore, sheets, leaves, or threads. (*f*) Silver in ore, sheets, leaves, or threads.

MINERAL PRODUCTION OF ALGERIA. (IN METRIC TONS AND DOLLARS; 5 f = $1.)

Year.	Antimony Ore.		Cement.		Clays.		Copper Matte.		Gypsum, Crude.		Gypsum, (Plaster).		Iron.	
													Ore.	
1894..	175	5,200	..	.....	.....	.....	702	$64,584	23,900	$114,900	.....	.....	343,880	$527,782
1895..	307	7,365	...	.....	.....	.....	364	30,800	2,800	1,248	34,136	$131,978	318,416	508,946
1896..	658	18,957	60	$300	48,297	$42,526	427	39,028	300	138	29,870	114,123	374,476	583,946
1897..	781	21,621	...	.....	67,180	59,607	289	28,380	350	175	29,120	109,473	441,467	663,282
1898..	138	4,416	...	.....	78,690	60,275	488	54,200	150	75	29,750	110,585	473,569	703,667

Year.	Iron—*Con.*		Lead-Silver Ore.		Lime.				Marble.		Onyx.		Phosphate Rock.	
	Cast, 2d Fusion.				Hydraulic.		White.							
1894..	538	$32,499	276	$4,045	.....	.....	.....	.....	.....	.....	.....	.....	64,260	$230,868
1895..	537	35,894	178	2,959	20,000	$120,000	11,890	$39,840	1,112	$32,248	1,764	$132,300	157,886	634,323
1896..	577	35,480	117	2.188	20,000	120,000	9,450	36.480	900	26,100	900	25,530	165,738	500,905
1897..	699	44,870	145	3,041	20,425	122,500	9,215	32,260	1,660	26,150	364	25,480	228,141	913,564
1898..	*Nil.*	.....	120	3,120	13,000	78,000	12,975	46,783	985	30,004	219	12,483	269,500	1,078,000

Year.	Quicksilver Ore.		Salt.		Sand and Gravel.		Zinc Ore.		
1894..	966	$11,602	17,830	$74,763	.....	.....	29,708	163,392	Algeria exported in 1894, 2,673 of lead and copper ore, 28,785 of zinc ore, and 71 tons of antimony ore.
1895..	85	6,210	25,758	102,990	20,400	$2,400	13,967	93,486	
1896..	8	500	19,658	85,122	41,400	8,280	17,567	169,766	
1897..	.....	.....	23,223	78,058	80,860	20,280	32,269	301,538	
1898..	.....	.....	21,300	86,000	72,185	17,717	29,800	281,400	

MINERAL PRODUCTION OF NEW CALEDONIA. (a) (IN METRIC TONS.)

Substances.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	1898.
Chrome iron ore	3,895	4,343	512	242	2,927	8,079	20,186	3,949	14,300
Cobalt ore	3,740	1,343	2,200	2,200	4,112	4,277	c4,823	3,200	21,000
Copper ore	(b)	199	(b)	(b)	(b)	(b)	(b)	2,200	5,300
Copper regulus	259	152	(b)	(b)	(b)	(b)	(b)	.....	(b)
Lead ore	881	236	(b)	(b)	(b)	(b)	(b)	200	(b)
Nickel ore	22,689	60,921	83,114	69,614	61,243	29,623	6,417	26,464	53,300

(a) From *Statistique de l'Industrie Minérale*. (b) Not stated in the reports. (c) Exports.

MINERAL PRODUCTION OF TUNIS. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Salt.		Galena.		Zinc Ore.		(a) From *Annual General Reports*, by C. Le Neve Foster, and *Statisque de l'Industrie Minérale*. Additional products in 1896: Gypsum, 4,700 metric tons; marble, 1,000 metric tons; phosphate of lime, 1,000 metric tons; building stone (dressed), 18,000 cu. meters; stone (broken), 135,000 cu. meters. (b) Not reported. (c) Value corresponding to only 10,800 tons.
1894....	7,000	(b)	(b)	.....	31,000	(b)	
1895....	8,000	$7,200	(b)	.....	14,800	c$112,600	
1896....	5,500	22,000	(b)	.....	12,100	118,600	
1897....	8,100	32,400	2,123	$26,960	11,880	147,200	
1898....	7,300	33,200	2,375	37,680	21,477	185,520	

GERMANY.

THE mineral statistics of the German Empire, together with those of Baden, Bavaria, Prussia and Saxony, are summarized in the subjoined tables:

MINERAL PRODUCTION OF GERMANY. (*a*) (*b*) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Antimony Ore.		Arsenic Ore.		Asphaltum.		Boracite.		Coal.		Lignite.	
1895....	24	$218	3,546	$35,311	59,563	$113,606	150	$8,781	79,169,276	$134,723,786	24,786,363	$14,502,891
1896....	*Nil.*	.....	3,691	46,960	61,552	113,348	184	10,620	85,690,233	148,244,000	26,797,880	15,238,278
1897....	*Nil.*	.....	3,777	56,021	61,645	94,634	198	10,222	91,054,992	162,234,685	29,419,508	16,562,642
1898....	*Nil.*	.....	3,540	52,414	67,649	104,064	230	10,746	96,279,992	177,564,243	31,648,498	18,339,869

Year.	Cobalt, Nickel and Bismuth Ores.		Copper Ore.		Graphite.		Iron Ore.		Kainite.	
1895....	5,180	$163,450	633,354	$3,844,961	3,751	$50,612	12,349,600	$10,268,985	690,174	$2,402,339
1896....	4,087	129,688	717,306	4,239,362	5,248	72,108	14,162,315	12,349,650	856,290	2,989,736
1897....	3,355	139,777	700,619	4,752,552	3,861	66,126	15,485,979	15,021,922	992,389	3,486,007
1898....	3,157	138,545	702,781	4,921,152	4,593	97,916	15,893,246	15,202,159	1,103,643	3,835,856

Year.	Lead Ore.		Magnesium Sulphate.		Manganese Ore		Petroleum.		Pyrites.		Salt, Rock.	
1895....	161,614	$3,234,902	7,328	$14,856	41,327	$124,534	17,051	$240,614	127,086	$243,878	686,940	$777,095
1896....	157,504	3,249,000	2,350	5,250	45,062	120,155	20,395	297,128	194,950	231,579	755,833	755,898
1897....	150,179	3,253,884	2,601	5,637	46,427	115,356	23,308	349,111	133,302	241,117	763,412	804,298
1898....	151,601	3,412,128	2,444	5,277	43,354	111,786	25,789	394,552	186,849	242,586	804,658	888,656

Year.	Silver and Gold Ore.		Tin Ore.		Uranium and Wolfram Ores.		Zinc Ore.		Potash Salts. Unspecified.		Vitriol and Alum Ores. Unspecified.	
1895....	10.845	$427,047	154	$15,197	29	$4,449	706,428	$2,644,814	841,748	$2,776,504	351	$535
1896....	18,487	596,604	88	5,881	41	7,081	729,942	4,255,473	902,707	2,964,750	369	512
1897....	9.708	363,271	55	5,992	38	8,367	663,850	4,220,339	953,798	3,080,143	225	338
1898....	12,413	336,657	51	3,550	50	11,527	641,706	5,511,819	1,105,212	3,576,628	188	232

(*a*) Including Luxemburg. (*b*) From *Vierteljahrs- und Monats-hefte zur Statistik des Deutschen Reichs.* (*c*) Not reported.

METALLURGICAL PRODUCTION OF GERMANY. (*a*) (IN METRIC TONS AND DOLLARS; 4 marks = 1)

Year.	Antimony and Manganese.		Arsenical Products.		Cadmium—Kg.		Copper Matte & Black Copper.		Copper, Ingot.		Gold—Kg.	
1894.....	424	$65,244	2,339	$166,798	6,472	$6,886	676	$26,921	25,722	$5,467,554	3,199	$2,228,947
1895.....	989	107,960	3,014	207,416	7,047	9,809	789	49,890	25,777	5,818,903	3,547	2,469,379
1896.....	1,326	156,111	2,637	221,397	10,000	19,117	598	39,083	29,819	7,298,387	2,487	1,728,988
1897.....	1,665	210,744	2,989	270,949	15,531	44,157	315	14,380	29,408	7,545,376	2,781	1,934,371
1898.....	2,711	302,892	2,679	253,623	14,943	31,162	62	2,170	30,695	8,182,102	2,847	1,978,358

Year.	Iron, Pig.		Lead.		Litharge.		Mineral Paints.		Nickel & Various By-Products. (*b*)		Silver—Kg.		Sulphur.	
1894	5,380,089	$57,892,412	100,751	$4,762,392	3,646	$192,043	2,884	$72,802	997	$1,154,757	442,822	$9,625,876	2,168	$50,209
1895	5,417,548	58,667,695	111,058	5,569,475	3,453	190,806	2,655	72,443	1,143	1,111,074	391,979	8,600,828	2,061	40,024
1896	6,295,272	73,930,749	113,792	6,258,097	3,980	235,698	2,668	75,318	1,391	1,365,330	428,429	9,718,000	2,263	47,181
1897	6,864,405	87,345,441	118,881	7,160,138	3,841	213,699	3,119	84,071	1,464	1,492,337	448,068	9,095,366	2,817	48,357
1898	7,215,927	93,575,641	132,742	8,555,606	3,857	265,496	3,031	91,153	1,691	1,465,254	480,578	9,539,162	1,954	43,167

Year.	Sulphuric Acid.		Sulphates. (d) Copper.		Iron.		Zinc.		Copper and Iron Mixed.		Tin		Zinc.	
1894	357,908	$4,029,659	4,809	$363,084	8,922	$39,658	4,249	$56,815	202	$7,436	806	$271,804	143,577	$10,453,272
1895	537,928	3,713,673	4,636	341,600	9,820	40,579	4,018	55,950	170	6,492	884	266,292	150,286	10,409,160
1896	590,888	3,781,092	6,046	466,478	9,788	39,072	4,811	67,171	165	6,218	826	195,734	153,100	11,777,005
1897	628,130	3,739,472	5,549	470,016	10,351	44,684	5,488	77,495	266	10,543	929	287,829	150,739	12,619,131
1898	664,865	4,095,734	4,852	356,540	10,422	42,707	6,104	91,152	176	6,991	908	372,252	154,181	14,708,459

EXTRACTION OF SALTS FROM AQUEOUS SOLUTION. (a)

Year.	Alum.		Aluminum Sulphate.		Magnesium Chloride		Magnesium Sulphate.		Potassium Chloride.	
1894.....	3,914	$107,092	26,804	$480,174	17,422	$50,913	28,628	$88,814	149,775	$4,722,049
1895.....	3,358	87,372	30,886	539,000	17,089	52,850	26,028	107,188	154,427	4,921,179
1896.....	3,430	98,250	34,370	595,000	17,525	57,396	27,161	107,835	174,515	5,718,559
1897.....	2,995	78,838	37,053	613,684	18,014	64,294	35,072	155,585	168,001	5,764,425
1898.....	4,089	91,086	35,366	514,952	19,919	96,973	30,295	122,667	191,847	6,280,220

Year.	Potassium Sulphate.		Sulphate of Potassium and Magnesium.		Common Salt.		Glauber Salt.	
1894.....	23,281	$958,736	14,156	$274,694	522,590	$3,574,655	71,929	$423,218
1895.....	19,452	805,098	9,877	193,939	525,396	3,563,296	71,411	406,572
1896.....	19,682	813,381	4,628	85,977	547,486	3,661,084	71,958	448,915
1897.....	13,774	565,720	7,812	149,079	543,272	3,084,198	68,822	434,198
1898.....	18,853	763,897	13,982	259,485	565,668	3,116,087	69,111	452,509

(a) From *Vierteljahrs- und Monats-hefte zur Statistik des Deutschen Reichs.* (b) Including metallic bismuth, cobalt products, and uranium salts. (d) There was also produced nickel sulphate and tin chloride as follows: 1894, 107 tons, $31,398; 1895, 131 tons, $34,511; 1896, 176 tons, $42,986; 1897, 209 tons, $51,333.

MINERAL IMPORTS OF GERMANY. (a) (IN METRIC TONS; unit of value, $1,000; $1 = 4 marks.)

Year.	Alabaster and Marble, Crude.		Aluminum, Nickel Wares, etc.		Ammonium Sulphate.		Cement.		Clay Products. Roofing Tile and Brick.		Porcelain.		Kaolin, Feldspar and Fire clay.	
1894..	24,784	$750	291	$300	36,635	$2,375	24,868	$175	118,010	$400	698	$200	153,208	$1,350
1895..	23,785	725	299	300	29,208	1,600	27,351	200	114,020	375	748	200	153,911	1,350
1896..	26,655	800	335	350	32,061	1,450	32,394	275	140,333	475	806	325	192,807	1,675
1897..	29,633	900	417	425	33,113	1,325	42,364	350	151,455	575	812	350	207,155	2,050
1898..	(f)	....	459	475	30,254	1,275	53,519	325	c186,183	600	788	325	208,186	2,100

Year.	Coal.		Coal, Lignite.		Cobalt and Nickel.		Coke.		Copper. Crude. (d)		Copper. Bars and Sheets.	
1894.....	4,805,971	$15,075	6,868,162	$6,175	431	$350	404,179	$1,425	37,082	$7,900	495	$125
1895.....	5,117,356	15,825	7,181,050	9,875	621	425	461,779	1,675	44,365	10,050	426	125
1896.....	5,476,753	15,275	7,637,508	11,275	b 951	600	393,881	1,625	56,115	13,675	401	125
1897.....	6,072,029	16,625	8,111,076	12,775	b 1,390	875	435,161	1,950	67,573	17,025	400	125
1898.....	5,820,332	17,250	8,450,149	13,950	b 1,467	850	332,579	1,675	73,291	19,700	450	150

Year.	Copper—Con. Scrap and Coin.		Copper and Brass Manufactures. Fine.		Wire.		Cartridge Cases, Coarse Wares, etc.		Glass Manufactures, All Kinds.		Gold. Coin. Kg.		Bullion. Kg.	
1894.....	1,228	$250	565	$450	131	$50	589	$225	3,008	$1,100	80,429	$50,350	37,056	$35,775
1895.....	1,708	350	625	525	171	50	563	250	3,119	1,100	19,949	12,475	17,075	11,875
1896.....	3,170	700	797	675	127	50	582	275	2,823	1,050	37,507	23,475	45,417	31,600
1897.....	4,199	950	846	700	80	25	612	300	2,057	475	22,914	14,350	34,991	24,325
1898.....	4,720	1,150	881	800	55	25	546	275	2,408	525	60,355	37,750	62,987	43,775

Year.	Gold and Silver. Scrap, Pagament. Kg.		Gold and Silver. Manufactures. Kg.		Gold, Silver, and Platinum Ores.		Graphite.		Iodine.		Iron. Ore.		Iron. Pig.	
1894.....	66,269	$5,800	36,947	$1,350	13,935	$3,000	13,000	$475	431	$2,850	2,093,007	$6,850	203,948	$2,375
1895.....	45,174	4,125	35,094	975	6,184	1,350	12,565	425	113	750	2,617,136	6,825	188,217	2,250
1896.....	39,395	3,700	34,310	825	6,024	3,950	13,718	450	94	625	2,586,706	8,700	322,502	4,000
1897.....	36,664	3,075	35,160	750	8,927	4,475	17,366	600	164	950	3,185,644	11,675	423,127	5,375
1898.....	54,793	4,525	31,549	700	7,481	2,650	20,269	950	216	1,275	3,516,577	13,625	384,561	4,900

Year.	Iron—*Continued.* Scrap.		Blooms, Bars, and Ingots.		Angle.		Wrought, in Bars.		Rails.		Wire.		Cast, Crude.		All Other Manufactures.		Lead, Pig and Scrap.	
1894..	7,900	$100	719	$25	245	(f)	19,966	$700	3,542	$75	4,888	$400	14,997	$1,800	9,798	$1,100	24,280	$1,200
1895..	11,389	125	757	25	124	(f)	19,777	700	1,881	25	5,588	450	13,749	1,675	11,654	1,225	28,449	1,500
1896..	14,679	175	1,054	25	176	(f)	23,770	875	140	(f)	6,398	450	19,600	2,450	12,983	1,650	33,016	1,925
1897..	37,957	450	1,088	25	1,081	$25	29,467	1,225	774	25	5,609	375	24,627	2,950	18,135	1,975	35,092	2,350
1898..	23,328	300	1,553	50	207	(f)	26,014	1,100	267	(f)	7,166	475	39,371	3,825	20,158	1,900	47,497	3,250

Year.	Lead and Copper Ores.		Lead, White.		Manganese Ore.		Mineral Oil. Petroleum.		Mineral Oil. Lubricating Oil.		Phosphate Rock.	
1894.....	51,304	$2,125	646	$50	14,255	$225	785,102	$11,375	65,708	$2,800	286,812	$3,575
1895.....	65,270	2,825	479	25	22,576	275	811,058	15,400	75,041	3,275	276,046	3,475
1896.....	84,341	2,025	579	50	33,870	750	853,642	14,950	81,256	3,425	216,950	2,700
1897.....	90,114	3,300	696	50	86,911	925	946,344	12,300	88,957	3,250	289,234	3,625
1898.....	64,727	2,800	822	75	130,711	1,325	954,646	16,200	97,028	3,750	270,988	2,700

Year.	Potash.		Other Potassium Compounds. Chloride.		Cyanide.		Iodide.		Nitrate.		Sulphate.		Pyrites. (e)	
1894..	2,296	$200	478	$25	1	(f)	24	$150	1,135	$125	1,306	$500	315,115	$1,550
1895..	1,634	125	1,334	50	5	(f)	22	125	1,005	100	895	25	293,446	1,450
1896..	1,420	100	1,058	50	3	(f)	29	150	1,380	125	703	25	343,852	1,650
1897..	1,734	125	715	25	7	(f)	18	25	2,889	275	912	25	356,869	1,900
1898..	1,486	100	422	25	2	(f)	16	50	1,895	175	999	25	376,817	2,175

Year.	Salt.		Silica, Sand, Marl, etc.		Silver. Bullion—Kg.		Silver. Coin—Kg.		Slag and Slag Wool.		Slag, Thomas Slag, Ground.		Slate.	
1894..	23,440	$100	216,860	$225	190,340	$4,075	19,811	$375	632,878	$2,200	98,443	$550	61,401	$1,050
1895..	24,286	100	214,537	225	97,924	2,150	17,764	350	537,542	1,825	92,251	550	52,418	950
1896..	22,908	100	253,905	250	127,753	2,900	20,785	425	680,251	2,525	83,765	525	53,588	1,050
1897..	(f)	.....	228,241	225	147,084	2,975	20,331	375	670,234	2,575	110,216	575	48,380	1,075
1898..	(f)	.....	239,708	250	104,770	2,075	40,132	725	685,118	2,625	88,374	500	57,571	1,400

Year.	Soda, Calcined.		Soda, Nitrate. (Chile Saltpeter.)		Stassfurt Salts.		Stone. Rough or Simply Hewn.		Stone. Limestone, Lime.		Stone. Grindstones, Polishing and Whetstones.		Sulphur.	
1894..	753	$25	404,561	$17,200	770	$50	859,459	$5,375	214,331	$1,075	2,578	$200	20,267	$450
1895..	1,008	25	459,514	17,800	910	50	760,433	3,800	221,685	1,100	2,734	350	18,836	375
1896..	1,295	25	449,028	16,850	143	(f)	767,908	3,850	216,641	1,200	2,869	275	21,864	500
1897..	916	25	465,493	16,875	6	(f)	853,825	4,275	235,715	1,300	(f)	.....	25,305	625
1898..	524	(f)	425,054	15,325	17	(f)	931,826	4,275	242,397	1,325	(f)	.....	30,269	750

Year.	Tin. Crude.		Tin. Manufactures		Zinc. Ore.		Zinc. Crude.		Zinc. Drawn and Rolled.		Zinc. Manufactures		Zinc-White, Zinc-Gray, and Lithophone.	
1894..	10,775	$3,750	64	$75	14,712	$175	17,988	$1,375	275	$25	112	$75	2,513	$200
1895..	10,581	3,375	79	75	25,818	300	17,542	1,250	128	.....	114	75	2,906	225
1896..	13,796	4,275	75	75	21,498	325	16,343	1,350	180	25	139	75	2,899	250
1897..	12,395	3,875	75	75	24,735	400	19,734	1,850	180	25	146	75	3,532	325
1898..	14,623	5,300	82	100	(f)	.....	24,116	2,425	53	(f)	128	75	3,653	375

MINERAL EXPORTS OF GERMANY. (*a*) (IN METRIC TONS; unit of value, $1,000; $1 = 4 marks.)

Year.	Alabaster and Marble, Crude.		Aluminum, Nickel Wares, etc.		Ammonium Sulphate.		Cement.		Clay Products.					
									Roofing Tile, and Building Stone.		Porcelain.		Kaolin, Feldspar and Fire clay.	
1894	1,042	$50	1,374	$1,350	332	$25	406,369	$2,575	169,110	$1,125	15,582	$8,32	108,324	$1,025
1895	1,645	75	1,669	1,675	1,336	75	471,124	3,000	188,046	1,350	19,441	4,550	111,277	1,125
1896	2,174	100	1,977	2,050	2,201	100	478,340	4,100	287,811	1,775	21,624	8,975	125,674	1,250
1897	2,727	75	1,899	1,950	2,623	100	524,557	4,500	233,229	1,550	21,687	8,775	121,535	475
1898	(*f*)	.....	2,045	2,150	4,083	175	551,744	5,000	*c*265,110	1,000	21,644	8,425	129,082	650

Year.	Coal.		Coal, Lignite.		Cobalt and Nickel.		Coke.		Copper.					
									Crude. (*d*)		Bars and Sheets, Unplated.		Scrap and Coin.	
1894.....	9,739,085	$25,300	20,444	$25	101	$100	2,261,924	$8,875	6,609	$1,450	5,007	$1,300	1,883	$375
1895.....	10,360,838	26,750	18,814	25	176	125	2,298,328	8,900	6,329	1,425	4,700	1,300	2,136	450
1896.....	11,598,757	30,475	15,708	25	*b* 143	100	2,216,395	9,000	5,996	1,500	5,406	1,700	2,968	650
1897.....	12,389,907	33,375	19,112	25	*b* 169	100	2,161,886	9,150	7,188	1,825	5,718	1,825	2,164	525
1898.....	13,989,223	39,925	22,155	25	*b* 203	125	2,133,179	10,200	6,972	1,875	5,369	1,800	3,636	900

Year.	Copper and Brass Manufactures.						Glass Manufactures, All Kinds.		Gold.				Gold and Silver Manufactures. Kg.	
	Fine.		Wire.		Cartridge Cases, Coarse Wares, etc.				Coin. Kg.		Bullion. Kg.			
1894..	4,117	$3,100	3,433	$1,050	5,870	$3,475	110,623	$9,225	17,345	$10,875	3,514	$2,450	88,560	$6,350
1895..	4,912	3,800	3,975	1,250	7,093	4,425	118,199	10,150	24,788	15,525	7,262	25,050	98,126	7,950
1896..	5,961	4,950	5,910	1,950	6,806	4,375	108,892	9,175	40,095	25,150	34,747	24,225	96,702	5,525
1897..	5,588	4,625	6,176	2,025	5,409	3,350	105,702	9,050	11,385	7,150	32,312	22,525	101,429	9,975
1898..	6,472	5,475	5,990	2,025	6,278	3,775	102,615	7,750	82,992	52,050	4,851	3,375	98,939	10,275

Year.	Gold, Silver, and Platinum Ores.		Graphite.		Iodine.		Iron.									
							Ore.		Pig.		Scrap.		Blooms, Bars and Ingots.		Angle.	
1894	12	.....	1,942	$100	13	$75	2,558,729	$1,925	154,647	$2,000	77,728	$1,025	41,998	$ 850	130,458	$3,225
1895	359	$50	2,207	100	23	150	2,480,136	1,850	135,289	1,775	84,814	1,150	61,808	1,200	172,863	4,075
1896	14	25	2,364	125	26	175	2,642,294	1,975	140,449	1,925	52,466	750	49,529	1,125	178,887	4,675
1897	53	25	2,422	125	26	150	3,230,391	2,400	90,865	1,250	38,102	550	39,792	900	169,287	4,400
1898	19	75	2,986	175	26	150	2,988,734	2,150	187,375	2,525	85,095	1,300	34,964	800	204,705	5,400

Year.	Iron—*Continued.*								Lead and Copper Ores.		Lead, Pig and Scrap.		Lead, White.	
	Wrought, in Bars.		Rails.		Wire.		Cast, Crude.		Man'factures, All Other.					
1894	300,559	$7,375	119,410	$2,475	209,818	$6,575	125,057	$17,450	379,862	$20,450	2,245	$100	24,355	$1,175	13,990	$950
1895	277,991	7,000	116,627	2,550	205,331	6,500	135,868	20,450	248,355	18,750	2,174	100	27,855	1,425	14,260	1,025
1896	259,461	7,175	129,418	3,100	207,116	6,875	158,652	24,325	309,882	23,200	31,675	200	24,828	1,425	16,350	1,275
1897	246,772	7,075	113,473	2,850	198,909	6,625	170,084	26,725	296,949	22,400	26,817	225	24,075	1,525	14,786	1,225
1898	263,698	7,050	123,889	3,250	188,713	6,625	193,039	30,600	312,948	22,875	24,163	225	24,867	1,625	16,473	1,400

Year.	Manganese Ore.		Phosphate Rock.		Potash.		Potassium Compounds							
							Chloride.		Cyanide.		Nitrate.		Iodide.		Sulphate.	
1894.....	2,787	$75	3,761	$75	12,105	$1,175	93,913	$3,375	36	$125	13,213	$1,325	107	$725	25,357	$850
1895.....	4,461	125	5,813	100	13,443	1,100	78,718	2,825	48	150	12,928	1,250	128	875	23,132	775
1896.....	7,178	150	5,548	100	12,673	1,050	85,862	3,075	55	125	11,323	1,750	142	950	16,019	575
1897.....	8,615	100	4,000	75	13,100	775	80,389	2,850	1,096	(*f*)	8,986	850	124	775	20,971	675
1898.....	4,810	75	5,100	75	13,456	800	96,236	3,375	1,907	975	10,969	1,000	135	850	27,105	850

Year.	Pyrites. (*e*)		Salt.		Silica, Sand, Marl, etc.		Silver.				Slag and Slag Wool.		Slag, Thomas Slag, Ground.	
							Crude and in Bars.		Coin.					
1894..	17,510	$100	237,058	$650	226,780	$200	405,970	$8,725	7,679	$150	21,504	$75	85,630	$850
1895..	14,223	75	196,344	600	261,195	225	258,925	5,750	4,303	75	20,482	50	80,056	525
1896..	16,833	75	214,060	625	267,173	225	305,826	6,975	31,648	650	17,214	50	134,257	875
1897..	15,387	75	(*f*)	.....	652,248	575	371,086	7,550	27,687	525	27,728	100	169,336	1,225
1898..	19,220	100	(*f*)	.....	910,354	800	348,733	6,950	46,932	975	29,931	100	187,598	1,400

Year.	Slate.		Soda. Calcined.		Sodium Nitrate. (Chile Saltpeter.)		Stassfurt Salts.		Stone.					
									Crude or Simply Hewn.		Limestone, Lime.		Grindstones, Polishing and Whetstones.	
1894..	4,056	$55	33,556	$875	14,217	$700	238,765	$1,425	578,451	$2,900	69,891	$350	10,909	$550
1895..	4,489	75	31,418	775	13,437	800	221,184	1,375	506,163	2,525	71,771	275	10,465	650
1896..	5,796	100	41,106	925	9,078	400	285,023	1,575	620,394	3,100	77,641	350	11,914	900
1897..	4,948	100	45,672	1,025	11,364	475	337,577	1,700	545,738	2,725	77,905	350	(f)	.....
1898..	4,434	100	37,106	825	12,684	525	370,829	1,750	556,561	3,225	84,892	375	(f)	.....

Year.	Tin.				Zinc.								Zinc-White, Zinc-Gray, and Lithophone.	
	Crude.		Manufactures		Ore.		Crude.		Drawn and Rolled.		Manufactures			
1894..	618	$225	819	$735	35,682	$425	61,800	$4,525	16,088	$1,350	842	$575	14,437	$1,750
1895..	700	225	975	825	31,031	350	56,933	4,025	15,921	1,275	1,032	700	15,539	1,125
1896..	868	275	1,010	850	37,959	575	58,082	4,750	16,227	1,500	1,206	825	16,969	1,325
1897..	861	275	967	800	30,047	500	51,341	4,400	17,459	1,650	1,168	800	17,631	1,925
1898..	874	325	1,119	1,000	(f)	.....	51,324	5,175	14,477	1,575	1,205	850	18,674	1,775

(a) From *Statistisches Jahrbuch für das Deutsche Reich.* (b) Nickel only. (c) In 1898 represents unglazed tile and brick only. (d) Previous to 1896 includes copper scrap. (e) Previous to 1896 includes alum ore. (f) Not reported.

MINERAL IMPORTS AND EXPORTS OF GERMANY FOR THE YEAR 1899. (IN METRIC TONS.)

The following figures are taken from *Chemiker Zeitung*, of Feb. 10, 1900, and while probably from official sources, are subject to correction.

Substances.	Imports.	Exports.
Alum, and aluminate of sodium and alumina, including hydrate.	142	25,485
Aluminum	922	230
Ammonium carbonate	1,409	2,781
Ammonium sulphate	28,868	1,553
Ammoniacal liquor	21,504	16,971
Antimony	1,486	665
Antimony and arsenic ore	1,129	274
Arsenic	.....	168
Arsenic, white	525	776
Arsenical compounds	32	926
Asbestos and asbestic mastic	2,709	358
Asphalt, pitch, and wood cement	61,534	37,771
Barium chloride	2,317	2,422
Barium salts, N. E. S.	59	2,274
Baryta-white	4	2,251
Barytes	2,379	46,500
Borax and boric acid	2,570	2,875
Brass and tombac	2,096	5,059
Bromide of potash and other bromine preparations	5	261
Bromine	2	109
Calcium chloride	197	739
Carbon bisulphide	112	157
Carbonic acid	19	3,618
Cement	63,388	580,255
Chloride of lime	443	101,045
Chloride of magnesium	29	14,238
Chrome alum	7·8	821
Chrome ore	14,915	112
Coal	6,220,489	13,943,174
Coal-tar oil, light	529	2,755
Coal-tar oil, heavy	2,543	8,640
Cobalt and nickel ore	13,469	8
Coke	462,577	2,137,995
Copper, crude	70,091	7,061
Copper, in bars and sheets	137	1,594
Copper alloys, in bars, sheets, etc.	473	3,277
Cinnabar	9	187
Cryolite and bauxite	12,300	373
Explosives	22	3,248
Fluorspar	130	11,416
Gold, silver, and platinum ores	7,597	5
Gold preparations and salts	1·8	59
Graphite, amorphous	23,399	2,708
Gunpowder	8	2,652
Gypsum	.....	.....
Iodine	191	26
Iron, crude, all kinds	675,793	235,193
Iron, wrought, in bars	37,179	193,933
Iron alum and iron mordant	777	472
Iron ores	4,165,372	3,119,878
Iron oxide, red	2,199	1,151
Kaolin, feldspar, and refractory clay	237,233	143,406
Lead and copper ores	65,821	35,359
Lead	55,635	24,491
Lead, white	703	16,360
Lignite	8,616,751	20,921
Litharge	406	3,760
Magnesium, artificial carbonate	67	89
Magnesium, natural carbonate	11,733	3,119
Manganese ore	196,825	7,040
Manganese preparations	144	728
Mineral oil for use in the arts	3,155	.....
Muriatic acid	.....	.....
Nickel	1,391	295
Nitric acid	1,342	1,374
Ozokerite, crude	3,851	1,738
Ozokerite, refined	264	1,882
Petroleum, crude	5,735	5
Petroleum, distilled	8,108	3,606
Petroleum, refined	897,175	1,092
Phosphorus	377	125
Pitch, except asphalt	64,692	4,629
Potash, all kinds	1,737	11,917
Potash, caustic	133	16,006
Potassium and sodium chlorate	1,297	680
Potassium chloride	443	101,045
Potassium chromate	510	659
Potassium cyanide	3	1,645
Potassium iodide and other iodine preparations	9	145
Potassium sulphate	533	32,845
Quicksilver	572	23
Salt	22,040	241,096
Saltpeter, Chile	526,944	13,910
Saltpeter, potassium	1,735	15,146
Soda, calcined	514	40,566
Soda, caustic	1,267	3,884
Soda, crude and crystallized	92	1,705
Soda chromate	.....	.....
Soda, double carbonate	.....	.....
Sodium and potassium sulphide	128	1,594
Stone, refractory, from clay	41,597	117,851
Strontia	13,886	399
Strontia preparations	40	311
Sulphur	31,196	620
Sulphate of potassium and sulphate of sodium	.....	.....
Superphosphate	86,857	79,059
Thomas slag, ground	70,557	199,982
Tin	12,253	1,121
Tin salts	136	272
Ultramarine	61	4,310
Vitriol, all kinds	2,087	7,786
Waterglass	210	6,126
Witherite	3,598	368
Zinc, ingot	23,691	46,234
Zinc ores	57,880	25,192
Zinc, sheet	95	18,981
Zinc white, zinc gray, and zinc sulphate	4,226	19,489

MINERAL PRODUCTION OF BADEN. (*a*) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Alumina Sulphate.		Barytes.		Coal.		Fire Clay.		Gypsum.		Iron. Cast, Second Fusion.	
1894	1,863	$31,671	(*b*)	.....	3,768	$10,726	8,772	$8,346	40,652	$21,407	39,988	$1,502,847
1895	2,136	31,779	(*b*)	.....	4,189	12,657	7,458	4,130	29,805	19,504	31,195	1,400,720
1896	1,824	27,133	180	$97	4,001	11,008	6,819	4,267	32,801	24,651	31,356	1,730,776
1897	1,824	27,134	400	300	4,752	11,880	11,450	6,654	40,702	26,972	36,235	1,876,962
1898	2,051	30,505	1,160	825	4,133	10,333	5,112	3,529	28,087	16,137	39,988	1,876,962

Year.	Iron—*Continued.* Ingot.		Iron—*Continued.* Wrought.		Lead and Copper Ores.		Limestone. (*d*)		Manganese Ore.		Porphyry.	
1894	2,391	$89,252	1,265	$58,539	*c* 2·0	$415	107,139	$16,218	(*b*)	.....	(*b*)	.....
1895	4,126	158,077	1,047	50,238	*c* 2·5	105	40,188	6,332	(*b*)	.....	(*b*)	.....
1896	3,418	143,371	1,118	51,954	(*b*)	108	116,913	16,496	180	$895	(*b*)	.....
1897	3,875	147,718	1,167	58,401	(*b*)	.....	137,670	19,502	(*b*)	.....	28,000	7,000
1898	3,875	147,718	1,167	58,401	(*b*)	.....	164,979	26,104	(*b*)	.....	7,650	1,912

Year.	Quartz Sand.		Salt.		Sandstone. (*e*)		Sulphuric Acid.		Tripoli.		Zinc Ore.	
1894	2,211	$1,382	27,867	$214,081	1,592	$1,775	11,955	$76,208	12·0	$976	723	$1,446
1895	3,319	1,901	28,188	191,252	443	1,000	11,944	65,692	8·5	650	21	198
1896	1,198	602	29,227	136,993	(*b*)	.....	14,226	78,243	9·0	1,125	(*b*)	.....
1897	1,648	1,172	31,445	136,993	(*b*)	.....	13,365	73,507	9·0	1,800	(*b*)	.....
1898	1,694	910	31,445	136,993	(*b*)	.....	13,365	73,507	6·0	575	(*b*)	.....

(*a*) Reported to THE MINERAL INDUSTRY by the *Grossherzogliche Badische Domänendirektion*, Carlsruhe. There was also produced in 1894, 3,975 tons, $9,937, of sodium sulphate. (*b*) Not reported. (*c*) Lead ore only. (*d*) Including cement stone and bituminous shale. (*e*) Including millstones.

MINERAL PRODUCTION OF BAVARIA. (*a*) (IN METRIC TONS; 4 marks = $1.)

Year.	Barytes.		Clay. Fire Clay.		Clay. Kaolin.		Coal.		Coal, (Lignite).	
1894	4,550	$5,425	121,950	$219,154	15,944	$22,121	806,389	$1,972,304	20,637	$17,079
1895	3,587	4,388	106,925	180,188	13,250	19,042	903,340	2,147,504	26,531	23,304
1896	3,397	3,404	110,174	251,899	19,080	26,208	900,080	2,129,069	35,934	33,861
1897	3,365	2,906	144,425	300,495	24,086	38,345	917,022	2,267,086	39,043	24,913
1898	4,339	6,465	282,994	671,465	29,196	35,426	964,611	2,449,264	38,663	32,842

Year.	Copperas and other Sulphates.		Emery.		Feldspar.		Fluorspar.		Graphite.		Gypsum.		Iron. Ore.	
1894	668	$34,526	148	$1,583	1,720	$3,540	3,616	$6,015	3,133	$45,732	25,267	$13,219	138,977	$140,944
1895	637	34,170	229	2,353	1,115	3,075	3,940	5,986	3,751	50,612	21,773	11,597	145,191	148,282
1896	601	32,209	249	2,527	1,315	3,145	5,218	8,280	5,248	72,108	28,799	15,334	161,279	164,282
1897	981	38,540	217	2,307	1,699	4,678	4,904	5,744	3,861	66,126	26,153	13,609	172,699	173,809
1898	866	42,663	280	2,156	1,949	8,052	4,440	5,210	4,593	97,916	25,698	13,166	171,987	178,130

Year.	Iron—*Continued.* Bar.		Cast, 1st Fusion.		Cast, 2d Fusion.		Pig.		Sheet.		Wire.	
1894	46,860	$1,351,169	196	$6,606	53,342	$2,490,670	75,669	$865,543	268	$10,067	279	$7,426
1895	48,596	1,368,013	293	9,898	62,131	2,868,070	77,114	884,615	150	5,250	100	2,551
1896	53,573	1,587,398	114	3,903	71,006	3,378,247	79,621	907,306	(*b*)	.....	243	6,232
1897	58,200	1,884,224	138	4,629	78,008	3,735,979	83,418	968,932	(*b*)	.....	252	6,895
1898	58,342	1,864,479	97	3,566	84,227	4,067,760	84,144	1,005,125	(*b*)	.....	323	9,638

Year.	Iron—*Continued.* Steel.		Manganese Ore.		Marl. (For Cement.)		Ocher and Mineral Paints.		Pyrites.		Salt, Rock.	
1894	86,594	$2,022,324	80	$270	81,723	$83,426	9,195	$22,721	1,923	$4,748	630	$4,122
1895	96,829	2,336,689	150	113	85,089	85,864	8,579	25,945	1,955	14,616	1,321	8,443
1896	101,954	2,497,814	70	35	94,481	99,929	8,667	26,496	1,997	14,930	708	4,573
1897	115,530	2,851,035	130	197	97,831	60,591	8,673	25,872	2,211	7,080	1,161	5,586
1898	120,623	3,284,299	(*b*)	.....	110,757	52,374	8,748	31,737	2,304	7,285	736	4,096

Year.	Salt, Brine.		Silica (Quartz Sand).		Slate.		Soapstone.		Sodium Sulphate.		Stone. Flagstones.	
1894	42,183	$459,146	29,775	$9,632	1,146	$10,562	1,900	$22,913	568	$3,250	18,083	$77,982
1895	41,106	446,628	33,457	10,513	1,278	11,766	2,421	37,979	494	2,875	18,909	84,685
1896	40,400	438,379	29,868	8,441	1,565	14,252	3,051	52,458	663	3,067	20,559	86,620
1897	41,533	388,029	31,678	8,812	1,496	14,001	2,464	35,360	2,318	12,749	14,647	57,343
1898	39,717	469,629	45,907	17,444	3,956	24,039	1,912	29,103	2,332	14,185	16,720	81,892

Year.	Stone. Granite and other Massive Rocks.		Limestone.		Lithographic.		Sandstone.		Whetstone.		Sulphuric Acid.	
1894	627,116	$844,769	229,784	$60,943	9,286	$185,730	221,642	$291,924	66	$1,456	6,979	$59,693
1895	612,969	854,209	271,196	69,596	8,788	175,770	210,450	281,893	215	9,000	6,515	52,800
1896	658,581	856,753	298,434	74,983	10,868	163,020	235,518	321,591	88	3,740	7,064	61,214
1897	663,749	711,029	224,550	76,385	13,941	217,821	242,112	315,117	95	3,800	7,041	63,709
1898	678,171	790,124	214,309	78,897	12,029	180,435	296,189	418,580	85	1,675	103,385	1,020,499

(*a*) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (*b*) Not reported.

MINERAL PRODUCTION OF PRUSSIA. (*a*) (METRIC TONS; 4 marks = $1.)

Year.	Alum Shale		Antimony Ore.		Arsenic Ore		Asphalt.		Boracite.		Coal.		Lignite.	
1894	126	$189	(*b*)	.....	2,222	$22,220	14,108	$44,537	164	$10,490	70,643,979	$113,518,107	17,791,062	$10,512,640
1895	126	188	24	$218	3,046	30,460	14,391	44,659	140	8,256	72,621,509	119,898,605	20,114,877	11,527,852
1896	154	231	(*b*)	.....	3,247	41,196	16,204	43,715	171	9,896	78,993,655	132,782,104	21,981,201	12,195,391
1897	129	193	(*b*)	.....	3,377	50,655	11,466	18,706	185	9,663	84,253,393	145,665,149	24,222,911	13,324,245
1898	107	161	*Nil.*	.....	3,298	49,470	12,822	20,402	216	10,162	89,593,528	160,465,335	26,035,814	14,781,896

Year.	Cobalt Ore.		Copper Ore.		Epsom Salt.		Iron Ore.		Kainite.		Lead Ore.	
1894	208	$5,741	579,132	$4,012,585	7,734	$16,833	4,012,446	$6,141,224	529,169	$1,814,585	144,724	$2,900,092
1895	120	6,208	623,420	3,799,452	6,789	13,701	3,726,725	5,700,009	513,097	1,804,950	140,991	2,911,948
1896	181	9,868	707,395	4,186,321	1,793	3,971	4,053,100	7,101,832	616,462	2,424,132	138,398	2,998,433
1897	121	6,256	690,388	4,095,679	2,248	4,882	4,183,536	8,432,766	716,348	2,529,349	133,158	3,122,880
1898	34	1,700	691,866	4,867,052	2,061	4,468	4,020,809	8,135,065	744,240	2,560,761	133,637	3,218,107

Year.	Manganese Ore		Nickel Ore.		Petroleum.		Iron Pyrites.		Salt, Rock.		Silver and Gold Ore.	
1894	42,526	$98,950	1,341	$13,413	1,600	$39,791	123,149	$215,597	305,810	$329,326	6	$9,281
1895	39,882	105,916	2,058	4,190	1,612	46,446	115,051	201,660	278,326	327,861	12	23,626
1896	43,614	102,884	738	4,540	1,512	46,867	117,545	203,745	305,227	346,952	15	18,748
1897	45,254	108,137	204	1,580	2,600	73,088	121,766	209,201	310,755	358,697	6	4,901
1898	42,232	96,691	79	698	2,545	70,513	128,077	220,411	329,959	377,162	43	2,866

Year.	MINERAL PRODUCTION—*Cont.* Zinc Ore.		Other Potash Salts.		METALLURGICAL PRODUCTION OF PRUSSIA. (*a*) Antimony and Alloys.		Arsenic Products.		Bismuth.		Cadmium. Kg.	
1894	727,645	$2,567,053	625,662	$1,897,079	876	$40,084	1,147	$57,377	3,092	$7,117	6,052	$6,335
1895	706,179	2,643,282	558,846	1,707,929	924	73,660	1,788	98,363	1,000	1,750	7,047	9,809
1896	729,725	4,254,351	587,306	1,786,786	1,239	113,886	1,750	130,539	(*b*)	.....	10,667	20,435
1897	663,739	4,219,761	640,236	1,898,647	1,552	152,744	1,924	143,775	(*b*)	.....	15,531	44,157
1898	641,671	5,511,655	718,957	2,161,101	2,612	251,642	1,624	121,818	*Nil.*	.....	14,943	31,162

Year.	Cobalt Works' Products.		Copper.		Copper Matte		Gold—Kg.		Iron.		Lead.	
1894	46	$142,675	21,966	$4,670,025	670	$26,223	687·86	$479,215	3,744,116	$43,048,291	92,379	$4,387,085
1895	45	144,108	22,069	4,980,829	766	48,548	653·15	455,159	3,778,775	43,961,347	99,586	4,983,733
1896	69	220,624	25,683	6,398,780	969	27,098	755·19	524,958	4,470,551	55,144,390	102,413	5,621,335
1897	51	166,015	25,997	6,696,564	274	13,157	1,067·07	756,184	4,892,059	64,780,061	108,880	6,556,526
1898	44	141,952	27,216	7,277,385	62	2,170	1,036·31	719,787	5,176,943	69,822,640	119,346	7,692,050

Year.	Litharge.		Manganese and Alloys.		Nickel.		Ocher and other Mineral Paints.		Quicksilver. Kg.		Salt, Common.		Selenium.	
1894..	2,647	$141,079	48	$25,210	522	$480,675	2,250	$39,375	(b)	.....	280,511	$1,777,592	(b)	.....
1895..	2,458	137,408	66	34,300	698	616,143	2,059	38,863	(b)	.....	278,902	1,757,837	(b)	.....
1896..	2,544	153,432	86	42,225	822	713,455	2,110	43,750	(b)	.....	288,300	1,848,166	(b)	.....
1897..	1,999	127,737	113	58,000	898	760,562	2,400	48,750	4,867	$4,684	274,888	1,643,473	23	$402
1898..	2,360	166,554	99	51,250	1,108	706,198	2,376	51,600	4,717	4,298	286,051	1,640,214	65	1,015

Year.	Silver—Kg.		Sulphur.		Sulphuric Acid.		Sulphates.					
							Copper.		Iron.		Nickel.	
1894..	279,781	$6,102,001	1,888	$42,912	420,965	$3,096,586	1,858	$137,956	7,600	$27,788	46	$13,975
1895..	245,527	5,376,046	1,819	34,085	416,289	3,894,871	1,922	138,905	8,627	30,185	91	24,325
1896..	288,467	6,534,058	2,011	41,041	456,781	2,934,098	2,568	196,592	8,618	29,883	128	31,360
1897..	289,960	5,907,572	2,091	42,860	484,289	2,890,702	2,689	225,877	9,064	33,087	167	42,327
1898..	291,969	5,776,802	1,757	36,386	531,888	3,223,613	1,701	139,109	9,144	31,007	127	21,690

Year.	Sulphates—*Continued.*				Tin.		Zinc.	
	Zinc.		Mixed.					
1894..	2,746	$38,914	148	$4,813	842	$252,600	143,577	$10,435,118
1895..	2,641	89,266	112	3,598	849	254,592	150,286	10,397,219
1896..	3,102	46,723	126	4,266	804	231,111	153,082	11,775,566
1897..	3,583	54,601	225	8,548	912	282,384	150,739	12,606,595
1898..	4,158	65,987	120	4,126	979	367,155	154,643	14,686,429

(a) From *Zeitschrift für das Berg-, Hütten-, und Salinenwesen.* (b) Not reported.

MINERAL PRODUCTION OF SAXONY. (a) (METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Arsenical, Sulphur, and Copper Pyrites.		Barytes.		Bismuth, and Bismuth, Cobalt and Nickel Ores.		Coal. (Not Including Lignite.)		Fluorspar.		Galena.	
1894....	13,045·2	$33,491	291·3	$815	2,980·4	$175,513	4,123,227	$9,435,934	1,880	$3,525	3,412·6	$175,745
1895....	8,900·3	26,578	284·5	1,045	3,001·1	153,056	4,435,398	10,294,100	995	1,866	3,346·7	172,617
1896....	8,834·9	27,080	574·2	1,554	3,168·7	141,735	4,536,603	10,778,005	805	1,509	2,844·5	155,868
1897....	9,402·0	29,112	218·4	616	3,080·6	131,952	4,571,695	11,563,214	592	1,110	(f)	.....
1898....	6,410·3	16,269	477·9	1,270	3,043·4	136,167	4,436,455	11,829,001	775	1,453	(f)	.....

Year.	Iron Ocher, Swabian and Colored Earths		Iron Ore.		Lignite.		Limestone and Various Products. (b)		Manganese Ore.		Quartz, Mica and Uranium Ore.	
1894....	213·9	$776	1,043·0	$1,838	918,589	$641,886	379·9	$5,917	15·0	$187	735·0	$2,653
1895....	415·0	628	20·2	107	1,018,486	670,274	(f)	6,051	8·4	105	26·5	571
1896....	333·4	1,054	3,499·4	4,314	1,035,825	663,590	10·0	6,967	10·4	130	20·1	827
1897....	92·7	643	13,181·1	12,837	1,073,289	666,358	65·1	7,662	250·0	637	39·7	629
1898....	440·4	1,317	5,671·2	6,197	1,180,928	728,156	(f)	8,110	(f)	.....	116·9	1,628

Year.	Silver Ores. (c)		Tin and Tinstone		Wolfram.		Zinc Blende.		Specimens	Totals.	
1894....	14,628·2	$499,778	211·4	$23,697	38·7	$5,747	155·1	$621	$512	5,080,845·7	$11,008,582
1895....	14,431·6	493,431	154·4	14,942	29·1	4,289	134·6	567	775	5,485,561·4	11,770,917
1896....	13,315·3	455,183	88·2	8,555	40·3	7,149	72·2	378	706	5,606,044·5	12,257,580
1897....	11,428·5	449,625	54·7	5,981	36·7	7,085	111·6	579	540	5,683,437·0	12,678,552
1898....	14,658·9	467,777	51·0	5,957	50·6	11,597	28·6	89	921	5,649,107·2	13,215,913

Year.	Manufactured Products.							
	From a Part of the Coal.				From a Part of the Lignite.			
	Briquettes—*M.*		Coke.		Briquettes—*M.*		Lignite Bricks—*M.*	
1894.....	1,166	$5,711	66,742	$227,555	64,500	$40,590	61,084	$107,466
1895.....	1,412	6,917	70,449	252,420	99,228	68,245	65,353	115,255
1896.....	1,981	7,997	77,086	299,248	123,571	81,933	58,374	104,506
1897.....	(k) 3,547	13,517	77,507	331,863	(k) 58,460	107,451	60,166	105,456
1898.....	(k) 5,321	18,363	72,245	340,665	(k) 71,576	143,438	62,944	112,804

METALLIC CONTENTS OF THE MIXED ORES INCLUDED IN THE PRECEDING TABLE WHICH WERE DELIVERED TO THE FISCAL SMELTING WORKS AT FREIBERG. (*a*)

(IN METRIC TONS; 4 marks = $1.)

Year.	Total Mixed Ores.		Arsenic. Tons.	Bismuth. Kg.	Copper. Tons.	Gold. Kg.	Lead. Tons.	Nickel and Cobalt. Kg.	Silver. Kg.	Sulphur. Tons.	Zinc. Tons.
1894....	31,099·1	$709,397	424·7	(*j*)	20·8	(*j*)	4,619·5	558·0	31,635·6	5,854·2	134·9
1895....	26,679·5	694,131	350·1	(*j*)	14 9	0·1314	4,739·8	60·8	29,383·3	3,905·0	81·9
1896....	24,923·3	638,237	308·2	(*j*)	20·0	0·1736	3,887·5	130·1	26,268·0	3,784·9	122·6
1897....	20,772·2	477,664	353·1	(*j*)	9·6	0·1396	5,016·2	(*j*)	31,974·5	7,398·3	124·6
1898....	31,724·2	658,756	85·4	(*j*)	3·5	(*j*)	5,769·7	(*j*)	21,404·9	5,445·9	305·3

PRODUCTS SOLD BY THE FISCAL SMELTING WORKS AT FREIBERG AND THE COBALT WORKS AT SCHNEEBERG, GERMANY. (*a*) (IN METRIC TONS AND DOLLARS; 4 marks = $1.)

Year.	Arsenical Products. (*d*)		Bismuth. Kg.		Cobalt Products.		Copper Sulphate.		Fine Gold. Kg.		Lead.			
											Products. (*e*)		Sheet.	
1894....	1,229	$109,113	2,402	$9,725	404	$442,737	2,140	$164,553	957	$667,141	4,603	$286,044	1,160	$63,414
1895....	1,217	108,826	2,529	4,788	406	353,768	1,833	137,557	850	592,497	6,603	345,129	592	32,632
1896....	882	90,626	1,112	1,765	599	431,318	2,547	197,421	847	590,227	5,810	333,744	880	50,958
1897....	2,127	127,122	1,624	3,801	1,255	581,559	3,756	162,881	889	619,881	10,841	332,652	1,268	41,626
1898....	2,106	131,710	1,375	3,774	1,262	620,217	3,554	146,164	851	579,519	14,912	486,095	1.294	45,710

Year.	Lead—*Continued.*				Nickel, Speiss.		Fine Silver—Kg.		Sulphuric Acid. (*g*)		Other Chemicals. (*h*)	
	Shot.		Other Manufactures of. (*f*)									
1894....	161	$10,055	198	$11,981	75	$3,656	81,328	$1,745,881	12,679	$132,171	699	$6,995
1895....	238	14,613	218	13,563	26	1,052	54,770	1,211,118	10,205	118.419	624	6,744
1896....	149	10,001	213	14,874	88	7,195	46,577	1,063,337	9,228	97,812	499	5,788
1897....	284	9,850	448	15,771	150	4,166	72,962	1,465,076	21,565	73,450	943	5,673
1898....	258	9.515	436	16,746	111	3,409	79,555	1,583,071	23,149	102,639	1,089	5,820

Year.	Zinc and Zinc Dust.		Clay and Chamotte Manufactures. (*i*)	Total Value.
1894....	222	$18,056	$13,507	$3,634,982
1895....	163	11,896	11,729	2,959,963
1896....	16	1,347	12,593	2,909,008
1897....	241	10,483	16,855	2,907,602
1898....	453	21,978	18,478	3,774,873

(*a*) From *Jahrbücher für das Berg- und Hüttenwesen im Königreiche Sachsen.* (*b*) Including arsenic powder, slags, washing-sand, granular ore, refuse stones and chippings. (*c*) Including silver-bearing lead, copper, arsenic, zinc and sulphur ores. (*d*) Including arsenious acid, red, yellow and white glass, and metallic arsenic. (*e*) Including assay lead, soft lead, antimonial lead, litharge, lead fume and tin-lead. (*f*) Including lead pipes, lead wire and various lead apparatus. (*g*) Including sulphuric acid of all kinds. (*h*) Including copperas and glauber salt. (*i*) Including tiles, plates, figured stone, muffles, clay and graphite crucibles, and assaying utensils. (*j*) Not reported. (*k*) Quantity stated in tons.

GREECE.

THE statistics of mineral production in Greece are summarized in the following tables:

MINERAL PRODUCTION OF GREECE. (*a*) (*c*) (METRIC TONS AND DOLLARS; 1 drachma—20 cents.)

Year.	Blende.		Calamine, Calcined.		Chrome Ore.		Emery.		Gypsum.		Iron Ore.		Iron Ore, Manganiferous.	
1894..	1,080	$16,700	19,800	$348,480	1,477	$18,400	3,570	$46,410	85	$1,540	121,570	$142,100	159,080	$314,120
1895..	2,710	45,230	21,321	395,250	2,740	34,120	3,055	45,825	113	2,030	150,210	179,250	152,123	342,670
1896..	1,750	31,520	20,950	301,680	1,600	22,400	3,650	77,745	120	2,150	225,600	314,700	166,850	417,129
1897..	3,118	58,826	22,817	489,357	563	8,087	3,024	64,411	51	918	260,828	375,592	182,850	499,180
1898..	1,139	22,320	30,906	562,490	1,367	18,000	3,982	83,754	83	1,500	267,100	413,230	218,938	641,840
1899...	1,137	30,986	21,770	589,990	4,386	55,310	4,360	92,868	81	1,465	331,080	499,430	294,320	896,500

Year.	Lead, Soft.		Lead Ore, Argentiferous.		Lead, Argentiferous.		Lead Fume.		Lignite.		Magnesite, Crude.	
1894..	479	$24,910	6,250	$147,500	13,955	$1,073,500	1,680	$13,700	14,320	$28,640	13,282	$59,340
1895..	7	490	1,580	147,400	19,883	1,656,500	1,406	12,020	17,748	55,500	11,096	33,288
1896..	480	28,900	3,200	137,600	14,700	1,325,095	1,550	10,510	14,000	28,000	11,600	34,800
1897..	520	33,904	2,815	125,174	15,946	1,524,977	2,785	26,388	20,018	40,000	11,311	39,500
1898..	305	21,655	(*b*)	.....	18,888	1,882,136	2,655	27,248	17,310	34,620	14,829	54,100
1899..	291	23,862	(*b*)	.....	18,768	2,156,443	2,584	28,424	12,150	24,300	17,184	63,580

Year.	Magnesite, Bricks.		Magnesite, Calcined.		Manganese Ore.		Millstones. *Number.*		Puzzolan.		Sea Salt.		Sulphur.	
1894..	873	$17,460	602	$6,020	9,319	$52,190	11,448	$5,535	26,550	$25,700	21,310	$332,496	1,528	$35,735
1895..	(*b*)	.....	(*b*)	.....	7,250	42,750	(*b*)	10,020	29,810	26,830	22,238	335,800	1,450	31,970
1896..	(*b*)	.....	(*b*)	.....	15,500	93,000	(*b*)	8,200	31,300	30,100	22,800	344,280	1,540	35,500
1897..	826	18,172	686	7,550	11,868	73,580	6,975	3,785	42,600	38,340	20,421	308,357	358	7,518
1898..	516	11,352	129	1,160	14,097	90,220	18,500	9,500	70,700	72,114	25,250	363,600	135	2,880
1899..	542	19,512	3,087	71,100	17,600	98,560	12,563	6,307	46,375	39,882	27,125	579,150	1,150	24,150

(*a*) Statistics communicated by E. Grohmann, Seriphos. (*b*) Not stated. (*c*) In 1896 there was produced also ocher, 40 metric tons, value, $104, and in 1899 speiss from lead smelting, 1,100 metric tons, value, $12,760.

INDIA.

THE official statistics of mineral production in British India, and the imports and exports, are summarized in the subjoined tables:

MINERAL PRODUCTION OF INDIA. (*a*) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year.	Alum.		Antimony Ore		Asbestos. Kg.		Borax.		Clays.	Coal.		Copper Ore.	
1894.....	418	(*b*)	418	.....	203	$808	367	$25,009	$1,150,366	2,818,488	$2,062,076	(*b*)	.....
1895.....	418	(*b*)	(*b*)	.....	205	910	400	27,880	814,945	4,441,681	2,904,167	(*b*)	.....
1896.....	418	(*b*)	(*b*)	.....	304	12	340	30,084	880,922	3,909,764	3,065,067	22	$210
1897.....	509	(*b*)	(*b*)	.....	254	10	280	23,622	(*c*)	4,128,330	3,116,647	33	69
1898.....	762	15,750	(*b*)	.....	254	10	(*b*)	.....	(*c*)	4,678,640	3,576,167	(*b*)	.....

Year.	Fullers Earth.		Gold—Kg.		Granite. (*c*)		Graphite.		Gypsum.		Iron Ore.	
1894.....	(*b*)	.....	6,587	c$2,868,040	817,482	d$89,498	1,623	.....	3,220	$407	39,004	$50,07
1895.....	40,300	$3,066	7,741	*c*3,511,607	1,188,840	*d*108,216	(*b*)	.....	6,816	776	46,552	55,689
1896.....	(*b*)	.....	10,662	5,314,617	835,697	*d*124,777	(*b*)	.....	7,605	1,017	13,996	22,228
1897.....	318	125	12,100	5,581,439	633,302	*d*129,381	61	.....	8,187	1,062	44,009	38,075
1898.....	(*b*)	.....	12,773	6,031,890	806,131	*d*123,815	22	.....	8,390	1,160	50,559	46,90[illegible]

Year.	Laterite.		Limestone.		Manganese Ore		Mica.		Ochers.		Petroleum—*Gallons.*	
1894.....	4,038,852	$54,698	860,086	$102,726	11,583	(*b*)	183	$29,077	5,678	.....	11,089,597	$220,141
1895.....	5,678,001	56,906	1,109,272	89,797	16,070	(*b*)	382	63,322	5,772	.....	13,013,990	308,518
1896.....	5,867,878	70,235	1,135,730	128,291	57,782	$113,738	538	85,617	(*b*)	.....	15,057,094	448,339
1897.....	5,399,794	94,074	1,861,309	195,673	74,862	147,500	708	49,751	(*b*)	.....	19,128,828	565,943
1898.....	6,173,887	(*c*)	1,924,577	(*c*)	61,469	120,900	271	56,265	(*b*)	.....	18,972,368	254,580

Year.	Salt.		Saltpeter.		Sandstone.		Slate. (*c*)		Soapstone. (*c*)		Tin Ore. (*c*)		Trap Rock. (*c*)	
1894.....	1,318,136	$1,586,335	12,678	$56,685	683,871	$40,720	11,111	$7,664	745	$3,378	100	$21,740	67,214	.....
1895.....	1,120,780	1,225,180	11,583	48,919	831,921	*c*32,811	60,165	.....	343	972	22	4,464	29,038	.....
1896.....	1,043,171	1,369,821	16,140	56,720	1,001,245	*c*80,521	22,840	9,708	789	2,719	82	20,361	107,956	$5,447
1897.....	987,982	1,219,074	12,819	44,063	956,532	*c*48,328	26,589	10,102	1,003	3,915	62	9,888	140,764	19,620
1898.....	1,043,898	1,485,702	11,702	40,256	*c*1,174,454	(*c*)	26,954	10,755	1,839	(*b*)	40	9,572	128,772	(*c*)

(*a*) From the *Review of the Mineral Production in India*; and the *Annual General Report upon the Mineral Industry of the United Kingdom of Great Britain*. (*b*) Not stated in the reports. (*c*) Incomplete. (*d*) Represents only a part of the product, about one-third in 1894 and 1895, and one-half in 1896 and 1897.

MINERAL IMPORTS OF BRITISH INDIA. (a) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year. (b)	Alum.		Arsenic.		Asphalt.		Brass.		Cement.		Chalk and Lime.		Clay Products. Clays.	
1893-94...	3,040	$69,745	78	$7,612	832	$8,847	614	$155,695	25,959	$265,885	1,709	$ 6,718	1,394	$11,495
1894-95...	3,632	91,855	76	6,980	1,055	13,937	445	119,238	28,048	279,702	2,688	12,975	1,328	12,242
1895-96...	4,050	99,550	122	12,705	1,387	19,502	543	150,182	36,408	356,302	1,607	6,055	2,910	30,670
1896-97...	2,738	62,372	88	7,870	883	12,002	499	144,675	29,196	278,617	1,745	4,875	2,326	22,720
1897-98...	5,248	115,980	98	9,520	1,082	13,307	670	187,542	37,354	371,690	1,040	5,177	2,788	26,950
1898-99...	3,607	70,637	119	11,490	1,090	12,725	443	136,800	36,170	369,797	1,735	7,757	2,914	24,332

Year.	Clay Products.—*Continued.* Brick and Tile. *Number.*		Earthenware and Porcelain.		Earthenware (Piping).		Coal.		Coke.		Copper and Copper Manufactures.		Glassware.
1893-94...	2,092,919	$37,522	$628,990		2,629	$51,239	553,710	$2,341,092	11,437	$90,315	39,028	$5,956,296	$1,971,200
1894-95...	2,337,410	41,275	482,108		2,268	41,017	521,926	3,580,238	12,624	98,247	17,988	3,779,540	1,549,990
1895-96...	2,328,741	45,270	578,555		1,011	21,163	758,799	3,390,590	14,626	112,535	c28,968	5,294,837	1,852,695
1896-97...	3,915,291	72,475	537,752		1,891	26,400	488,782	2,826,690	13,568	117,352	12,237	2,688,535	1,760,460
1897-98...	4,507,161	69,637	512,605		785	15,870	251,574	1,213,220	15,027	130,160	16,383	3,554,325	1,441,680
1898-99...	3,544,390	69,082	468,372		829	14,162	352,540	1,651,390	11,573	86,567	12,766	2,808,190	1,655,732

Year.	Iron, Pig.		Iron and Steel, Manufactures of.		Lead. Ore.		Lead. Pig.		Lead. Manufactures		Mineral Oils. *Gallons.*	
1893-94...	7,504	$93,278	233,765	$7,516,075	215	$12,370	1,025	$45,775	4,954	$382,672	68,608,538	$8,188,962
1894-95...	d8,163	108,995	208,046	7,077,455	176	10,392	712	33,860	4,828	360,577	53,441,425	4,854,915
1895-96...	10,213	142,490	279,472	9,599,708	137	8,712	1,050	54,400	4,993	372,178	66,647,921	6,949,358
1896-97...	13,278	190,152	262,562	9,131,382	309	17,810	709	36,865	5,252	368,720	68,421,148	7,720,697
1897-98...	11,045	155,355	281,253	9,847,455	160	9,855	547	28,302	5,098	351,545	87,325,026	8,494,467
1898-99...	12,407	158,200	239,637	8,169,988	211	13,140	706	38,242	4,204	291,610	76,625,406	7,464,680

Year.	Paints and Colors.		Precious Stones.	Quicksilver		Salt.		Stone and Marble.		Tin and Tin Manufactures.		Zinc and Zinc Manufactures.	
1893-94...	8,872	$621,120	$361,027	143	$110,885	419,482	$1,977,670	25,050	$68,652	2,332	$818,980	6,681	$492,187
1894-95...	8,888	556,977	1,422,748	54	46,897	516,509	2,105,482	41,548	91,502	2,459	801,750	4,344	307,680
1895-96...	9,685	704,942	1,630,878	160	152,038	413,787	1,683,065	43,094	72,250	2,556	823,527	4,564	312,750
1896-97...	9,696	674,395	1,321,700	118	102,992	388,557	1,571,360	24,880	62,980	1,849	551,342	2,277	175,580
1897-98...	9,288	670,680	1,126,225	143	119,050	494,503	2,171,795	36,745	96,485	1,962	554,078	3,185	244,312
1898-99...	10,159	643,257	(e)	115	99,895	418,455	1,653,067	31,200	78,552	1,483	416,930	3,795	291,012

(a) From the *Tables Relating to the Trade of British India.* (b) Fiscal years ending March 31. (c) Also imported 2 tons copper ore, value $287. (d) Also imported, 1893-94, 5 cwt. iron ore, value $7; 1896-97, 2 metric tons, $82; in 1897-98, 52 metric tons, $735, and in 1898-99, 597 metric tons of ore and old iron, $5,933. (e) Not reported.

MINERAL EXPORTS OF BRITISH INDIA. (a) (IN METRIC TONS AND DOLLARS; 4 rupees = $1.)

Year. (b)	Borax.		Brass.		Coal and Coke.		Copper, and Manufactures of.		Earthenware and Porcel'n.	Glassware.	Jadestone.		Lead.	
1893-94...	367	$31,368	139	$75,005	52,853	$126,627	107	$32,073	$11,460	$11,745	115	$108,662	34	$1,417
1894-95...	400	34,852	179	89,627	54,523	120,557	112	35,700	11,582	13,642	229	113,265	129	5,390
1895-96...	340	30,035	204	127,325	82,217	216,450	123	38,817	12,842	14,997	215	136,538	57	3,115
1896-97...	280	23,622	188	106,012	138,906	352,810	111	35,980	11,432	12,897	219	153,780	210	8,427
1897-98...	184	16,277	162	76,755	216,261	535,340	84	26,325	10,157	11,717	196	150,025	117	5,127
1898-99...	250	24,107	156	70,937	332,907	838,257	130	40,062	8,802	12,742	228	156,075	193	7,215

Year. (b)	Manganese Ore.		Mica.		Mineral Oils. *Gallons.*		Precious Stones.	Saltpeter.		Stone and Marble.		Tin.	
1893-94.....	1,676	$3,800	246	$195,460	67,377	$5,247	$34,183	16,354	$845,355	374	$10,712	69	$21,167
1894-95.....	6,518	12,832	261	159,437	48,391	2,522	21,162	17,931	1,028,362	226	9,050	71	19,972
1895-96.....	23,122	45,515	452	268,052	315,118	38,817	37,287	21,425	1,339,865	315	10,555	13	3,437
1896-97.....	48,067	94,660	652	288,340	191,424	28,225	34,670	26,845	1,430,410	499	14,072	86	20,362
1897-98.....	80,090	157,507	590	267,142	15,563	2,860	30,970	21,218	996,862	995	16,010	49	10,945
1898-99.....	62,875	125,830	556	202,087	722,686	90,960	29,012	18,263	873,360	688	14,215	43	10,982

(a) From the *Tables Relating to the Trade of British India.* (b) Fiscal years ending March 31. (c) Not reported.

ITALY.

THE official statistics of mineral and metal production in Italy, together with the imports and exports as reported in the official statistics of the Kingdom, are summarized in the following tables:

MINERAL PRODUCTION OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Alunite.		Antimony Ore.		Asphaltic Rock.		Boracic Acid, Crude.		Coal. (*c*)		Copper Ore.		Gold Ore.	
1894.....	6,000	$4,800	1,504	$45,059	60,493	$280,678	2,746	$236,156	271,295	$878,792	92,686	$445,629	7,748	$132,782
1895.....	7,000	5,600	2,241	40,254	46,713	204,750	2,633	184,310	305,321	433,555	83,670	367,516	7,099	129,887
1896.....	6,000	6,000	5,066	60,590	45,456	177,728	2,616	167,424	276,197	396,372	90,408	424,719	7,659	170,602
1897.....	6,500	6,500	2,150	34,864	55,339	189,655	2,704	173,056	314,222	467,111	93,377	431,229	10,723	178,010
1898.....	7,000	7,000	1,931	43,822	93,750	265,645	2,650	163,600	341,327	485,965	95,128	426,299	9,549	148,826

Year.	Graphite.		Iron Ore.		Lead Ore.		Manganese Ore.		Manganiferous Iron Ore.		Petroleum.		Pyrites. (Cupriferous in part.)	
1894.....	1,575	$2,520	187,728	$417,831	29,822	$758,422	760	$4,700	5,810	$9,296	2,854	$169,452	22,638	$59,269
1895.....	2,657	8,599	183,371	405,711	30,632	813,324	1,569	14,128	5,860	9,376	3,594	186,099	38,586	85,741
1896.....	3,148	10,193	203,986	507,973	33,545	992,865	1,890	20,450	10,000	20,000	2,524	128,894	45,728	108,825
1897.....	5,650	11,300	200,709	572,102	36,200	1,008,525	1,634	15,008	21,262	34,019	1,932	98,456	58,320	156,028
1898.....	6,435	17,423	190,110	549,248	33,930	1,044,248	3,002	18,707	11,150	26,760	2,015	117,826	67,191	165,610

Year.	Quicksilver Ore.		Salt, Rock.		Salt from Brine.		Silver Ore.		Sulphur, Crude.				Zinc Ore.	
									Ground.		Fused.			
1894	15,022	$203,443	19,467	$56,974	11,326	$57,861	1,103	$169,444	13,850	$96,950	391,981	$4,956,641	182,777	$1,839,791
1895	10,504	166,680	18,710	56,019	10,605	54,171	870	128,273	(*d*)	.....	370,766	4,134,588	121,197	1,542,905
1896	14,305	147,570	17,300	52,024	11,974	61,298	640	107,251	(*d*)	.....	426,353	6,134,253	118,171	1,498,129
1897	20,659	157,782	19,801	54,404	11,725	63,100	405	85,625	(*d*)	.....	496,658	8,995,647	122,214	1,656,065
1898	19,201	132,223	18,199	61,147	11,546	59,568	435	76,048	(*d*)	.....	502,351	9,708,050	132,099	2,412,333

REFINED PRODUCTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Alum.		Aluminum. Sulphate.		Antimony.		Asphalt, Mastic and Bitumen.		Boric Acid, Refined.		Borax, Refined.	
1894.....	1,065	$30,690	2,480	$45,960	351	$57,626	7,820	$49,826	170	$22,100	1,141	$136,872
1895.....	*b*1,115	21,000	2,950	53,760	423	56,986	14,491	75,920	253	29,807	944	101,930
1896.....	850	16,200	2,390	41,020	538	72,440	12,490	75,280	253	32,838	943	79,279
1897.....	1,030	21,180	2,310	40,160	404	57,072	18,644	89,125	260	31,200	990	83,160
1898.....	1,165	25,060	2,915	54,560	380	62,550	17,813	111,611	166	16,590	702	50,558

Year.	Coal. (Briquettes.)		Copper.		Gold—Kg.		Iron.					
							Pig.		Bar, Sheet, Pipe, Wire, etc.		Sheet, Tinned.	
1894.....	527,904	$3,207,265	2,670	$676,468	349·0	$252,057	10,329	$901,339	141,729	$6,643,515	5,750	$563,000
1895.....	451,470	2,527,564	2,375	577,352	280·0	199,453	9,213	207,948	163,824	6,962,837	5,860	555,360
1896.....	422,409	2,081,645	2,842	768,160	274·7	189,024	6,967	192,681	139,991	6,606,951	2,918	256,784
1897.....	549,050	2,768,350	2,960	749,954	316·0	218,862	8,393	181,763	149,944	7,176,951	6,500	546,000
1898.....	594,500	3,373,900	3,230	878,509	187·9	131,818	12,327	259,897	167,499	8,173,165	7,200	660,000

Year.	Iron—*Continu'd.* Steel.		Lead.		Petroleum, Benzine, etc.		Quicksilv'r.		Salt, Sea.		Silver—Kg.	
1894.....	54,614	$3,353,852	19,605	$968,446	1,640	$193,520	258	$227,040	402,515	$830,382	58,626	$1,289,772
1895.....	50,314	3,794,256	20,353	1,078,920	4,191	440,953	199	189,308	448,335	957,529	44,189	1,007,460
1896.....	65,955	4,874,539	20,786	1,162,469	2,734	296,147	186	174,840	422,555	856,072	38,075	882,228
1897.....	63,940	4,156,850	22,407	1,402,909	3,392	279,533	192	192,000	429,258	885,637	45,318	917,670
1898.....	87,467	5,417,096	24,543	1,646,865	5,040	395,821	173	173,000	451,426	710,501	43,437	894,157

Year.	Sulphur. Refined.		Sulphur. Ground.	
1894.....	71,295	$1,152,576	90,561	$1,654,465
1895.....	75,329	1,125,494	91,517	1,473,068
1896.....	71,072	1,198,447	89,292	1,602,797
1897.....	85,872	1,874,642	69,178	1,541,510
1898.....	99,494	2,273,053	146,001	3,259,999

(*a*) From *Rivista del Servizio Minerario.* (*b*) Including 120 tons of soda alum. (*c*) Includes anthracite lignite, fossil-wood, and bituminous schist. (*d*) Not reported.

MINERAL IMPORTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = £1.)

Year.	Antimony.		Arsenic—Kg.		Asbestos.		Asphaltic Prod. (Bitumen.)		Barium Sulphate.		Borax and Boric Acid.	
1894.....	30·4	$5,715	1,200	348	856·7	$102,804	9,380	$150,080	629·1	$16,357	182	$19,850
1895.....	41·6	7,238	1,200	348	530·8	47,772	6,075	97,205	507·9	13,205	177	19,475
1896.....	38·2	6,112	(*b*)	.....	850·8	76,572	11,892	190,277	549·4	14,284	166	18,249
1897.....	65·7	10,512	3,200	928	618·9	61,890	1,632	26,120	577·9	15,025	253	24,221
1898.....	58·3	11,077	700	203	1186·4	118,640	1,150	18,405	860·2	22,365	147	10,766

Year.	Cement and Hydraulic Lime.		Chalk.		Clay Products. Brick, Tile, etc.		Kaolin.		Majolica Wares.		Porcelain.	
1894.....	14,100	$126,900	14,866	(*b*)	15,336	$107,640	2,417	(*b*)	359·7	$16,532	496·1	$148,630
1895.....	15,120	136,080	17,102	$119,714	14,623	103,193	2,924	$23,392	416·3	18,109	484·4	132,181
1896.....	12,810	115,290	15,716	110,012	18,504	129,528	3,775	30,200	438·7	20,293	441·0	123,128
1897.....	16,680	150,120	28,937	202,559	19,086	133,602	5,719	45,752	476·1	24,637	471·7	126,548
1898.....	12,029	108,261	18,252	127,764	21,681	151,767	9,079	72,632	101·5	4,922	587·6	100,519

Year.	Clay Products—*Continued.* Potters' Clay and Manufactures.		Terra Cotta.		Coal.		Copper Ore.		Copper, Brass and Bronze.	
1894.....	715·9	$106,540	2,274	$71,922	4,696,258	$22,072,413	908	$72,240	6,446	$2,121,185
1895.....	748·5	117,060	2,583	79,194	4,304,787	17,219,148	1,600	160,000	6 661	2,431,779
1896.....	786·6	120,820	2,675	84,622	4,081,218	17,141,116	484	67,760	6,955	2,592,932
1897.....	743·4	116,980	2,167	69,621	4,259,643	19,594,358	1,611	225,540	7,999	2,678,833
1898.....	824·9	129,330	2,122	67,785	4,431,524	27,475,449	5,471	820,650	7,433	2,645,153

Year.	Copper and Iron Sulphates.		Glass and Manufactures.		Gold. Coin—Kg.		Unrefined—Kg.		Man'fact'res Kg.		Graphite.		Iron. Ore.	
1894.....	17,467	$1,397,392	12,274	$914,911	4,633	$2,872,460	1,609	$996,580	1,871	$628,050	152·6	$9,156	1,003	$1,205
1895.....	14,124	1,128,992	11,237	924,776	784	486,080	1,119	581,880	1,851	775,990	216·3	12,978	1,742	3,484
1896.....	24,255	2,037,462	9,322	813,134	1,004	622,480	2,517	1,308,840	1,515	570,446	204·4	12,264	594	1,425
1897.....	28,878	2,656,849	11,182	937,282	444	275,280	807	419,640	1,375	459,240	314·7	18,882	5,331	15,161
1898.....	25,560	2,351,538	10,399	863,916	154	95,480	507	263,640	1,844	641,785	381·8	26,726	8,723	23.552

Year.	Iron. Pig.		Iron. Wrought.		Iron and Steel. Plates, Rods and Manufactures.		Lead. Ore. (*c*)		Lead. And Alloys in Pigs.		Lead. Manufactures.	
1894.....	119,267	$1,908,272	6,263	$368,330	44,908	$6,704,733	13,379	$387,991	1,499	$80,924	131	$34,463
1895.....	131,870	2,109,920	5,398	326,064	63,672	6,880,541	8,901	284,832	947	53,043	233	42,639
1896.....	119,491	1,911,854	4,820	327,413	52,202	6,919,787	9,730	408,660	1,166	69,978	192	39,524
1897.....	156,019	2,496,310	3,801	269,378	88,895	5,179,932	14,854	653,576	1,178	75,424	247	54,481
1898.....	169,059	2,874,010	4,076	307,371	98,564	5,533,233	10,947	503,562	1,431	100,142	435	75,950

Year.	Nickel Alloys and Manufactures of.		Ocher & Other Colored Earths		Oxide and Carbonate.		Potash, Ammonia and Caustic Soda.		Potassium Sulphate.		Precious Stones, Manufactures. Kg.	
1894...	672·1	$742,215	1,149·6	$27,590	899	$65,356	9,665·7	$541,499	346	$19,398	467·5	$1,434,655
1895.....	248·6	304,826	933·6	22,406	848	65,356	9,906·7	499,140	262	14,688	454·1	1,897,948
1896.....	411·3	473,189	851·7	20,441	523	41,087	9,840·7	458,087	431	24,142	478·9	1,505,464
1897.....	432·4	473,386	887·7	21,305	580	43,211	11,012·0	528,537	562	30,375	490·8	2,033,714
1898.....	257·7	345,793	691·9	16,606	647	54,314	11,047·4	493,393	928	50,101	318·6	1,629,008

Year.	Silver.						Quicksilver.		Slag.		Sodium Carbonate.	
	Coin—Kg.		Unrefined in Bars—Kg.		Manufactures. Kg.							
1894.....	447,950	$17,918,000	3,529	$70,580	4,128	$66,061	11·4	$9,576	9,454	$34,539	19,129	$458,880
1895.....	9,768	390,720	2,845	56,900	6,070	102,887	10·3	9,373	11,316	113,156	19,949	359,078
1896.....	22,751	910,040	2,291	48,111	6,533	125,212	30·2	27,180	30,975	368,294	18,927	329,335
1897.....	26,008	1,040,320	2,434	43,812	5,286	91,453	30·3	27,270	27,201	409,208	20,721	393,699
1898.....	8,241	329,640	991	17,838	5,582	96,594	39·0	37,440	51,129	563,197	20,845	458,588

Year.	Sodium Nitrate.				Sulphur, Crude and Refined.		Tin.				Zinc.	
	Crude.		Refined, and Potassium Nitrate.				Bars.		Manufactures.		Ore.	
1894..	6,128	$306,425	601	$45,653	267·4	$3,851	1,297·6	$503,469	77·3	$34,558	18	$221
1895..	12,670	582,825	569	39,865	255·2	3,318	1,484·0	516,432	87·8	35,520	22	330
1896..	11,665	537,501	541	40,320	4·7	72	1,763·5	581,955	90·6	35,156	(*f*)	.....
1897..	16,400	705,204	917	69,946	236·4	4,492	1,520·4	501,732	81·4	31,560	(*f*)	.....
1898..	19,961	818,409	702	48,478	175·8	3,621	1,721·7	688,680	109·3	50,341	216	4,320

Year.	Zinc—*Continued.*						All Other Ores.		Metals Not Specified.			
	Oxide.		Pigs and Bars.		Manufactures.				Crude.		Manufactures.	
1894.	503·2	$59,320	2,229·9	$196,231	3,248·9	$385,067	108	$2,160	947·5	$70,115	4·2	$3,184
1895.....	576·4	57,640	2,378·0	190,240	3,136·4	344,390	297	5,940	933·3	69,004	17·9	9,308
1896.....	540·2	54,020	2,596·0	233,640	3,482·4	421,918	1,894	75,760	1,134·6	90,768	21·1	10,972
1897.....	570·0	57,000	3,278·2	295,038	3,556·5	421,153	380	15,200	1,508·7	114,661	30·3	15,756
1898.....	572·8	63,008	2,812·9	308,793	3,200·0	441,066	917	40,348	1,611·8	128,944	27·2	14,416

(*a*) From the *Movimento Commerciale del Regno d'Italia.* (*b*) Value included under limestone. (*c*) Includes argentiferous lead ore. (*d*) Including other stones and non-metallic minerals. (*e*) Representing the values of chalk, kaolin, limestone, and other stones and non-metallic minerals. (*f*) Not reported.

MINERAL EXPORTS OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Antimony.		Asbestos.		Asphaltic Product. (Bitumen.)		Barium Sulphate.		Borax and Boric Acid.		Cement and Hydraulic Lime.	
1894.....	278·6	$52,378	70·3	$8,436	6,091·3	$36,548	24·6	$639	2,015	$222,322	2,416	$19,328
1895.....	254·1	44,213	29·3	2,637	9,535·5	57,213	1,053·8	27,399	2,591	244,275	4,534	36,272
1896.....	361·0	57,760	129·7	11,673	13,729·2	92,375	65·6	1,705	2,719	252,785	3,871	30,968
1897.....	270·9	43,344	169·9	11,893	15,310·0	244,960	143·2	3,723	1,618	145,676	5,330	42,640
1898.....	338·5	64,315	208·4	16,672	19,465·0	311,440	70·3	1,828	2,167	143,925	5,192	41,536

Year.			Clay Products.									
	Chalk.		Brick, Tile, etc.		Kaolin.		Porcelain.		Potters' Clay and Manufactures.		Terra Cotta.	
1894.....	5,002	(*b*)	107,197	$610,273	18	(*b*)	102·0	$26,024	471·4	$72,700	1,857	$61,593
1895.....	3,719	$18,595	127,177	727,210	23	$184	161·1	35,566	590·7	90,010	2,509	79,737
1896.....	5,598	27,965	143,648	809,823	49	392	98·8	24,912	1,142·5	157,100	2,852	97,548
1897.....	7,556	37,780	125,925	710,155	98	744	137·7	33,164	1,292·1	174,910	2,472	85,653
1898.....	6,744	40,504	125,614	711,039	94	752	85·7	28,476	1,347·4	196,600	2,751	96,254

Year.	Coal.		Copper Ore.		Copper, Brass and Bronze.		Copper & Iron Sulphates.		Glass and Manufactures of.		Gold.	
											Coin. (Kg.)	
1894.....	13,692	$64,352	7,789	$112,162	149	$64,257	18	$1,464	5,551	$1,069,004	6,359	$3,942,580
1895.....	16,497	65,988	5,859	87,885	268	113 300	12	960	6,327	1,198,147	3,139	1,946,180
1896.....	18,924	71,481	3,608	61,251	643	225,211	71	5,998	5,867	1,066,612	2,483	1,539,460
1897.....	23,191	106,679	2,408	40,936	641	281,764	18	1,638	6,022	1,015,225	1,608	996,960
1898.....	17,749	110,044	2,356	42,408	857	169,986	25	2,300	5,209	849,328	2,566	1,590,920

Year.	Gold.—*Cont.* Unrefined. (Kg.)		Graphite.		Iron. Ore.		Iron. Pig.		Iron. Wrought.		Iron and Steel. Plates, Rods and Manufactures.	
1894.....	1,543	$802,360	2,285·1	$18,281	159,205	$302,489	1,052·9	$16,846	264·2	$13,241	1,345·9	$164,411
1895.....	2,759	1,434,680	3,153·3	25,226	164,367	328,734	965·9	15,454	430·1	27,510	2,467·6	266,465
1896.....	2,517	1,308,840	3,727·1	29,817	187,059	448,941	1,378·5	22,056	427·3	26,006	1,819·5	227,305
1897.....	1,381	718,120	4,164·2	45,806	207,619	539,809	498·3	7,973	1,434·3	67,10(	7,148·1	662,990
1898.....	1,739	904,280	5,145·4	56,599	217,556	587,401	840·0	14,280	699·3	33,870	4,606·5	533,233

Year.	Lead. Ore (*c*).		Lead. And Lead Alloys in Pigs.		Lead. Manufactures.		Lead. Oxide and Carbonate.		Nickel, Alloys, and Manufactures of.		Ocher and other Colored Earths.	
1894.....	6,354	$184,266	1,157	$62,505	1,762	$135,687	834	$75,645	14·0	$20,665	1,704·5	$37,499
1895.....	6,622	211,904	2,153	120,596	1,492	125,716	747	71,531	1·7	3,422	1,923·3	42,313
1896.....	4,731	151,392	1,419	85,140	1,441	141,019	489	47,823	34·7	34,229	2,411·6	53,055
1897.....	4,747	161,398	2,790	178,586	1,410	141,672	461	45,829	26·9	25,270	2,318·2	46,364
1898.....	4,492	157,220	5,870	410,886	1,764	189,362	414	41,413	29·4	28,596	2,884·2	57,684

Year.	Potash, Ammonia, and Caustic Soda.		Quicksilver.		Salt, Sea and Rock.		Silver. Coin. Kg.		Silver. Unref'd in Bars, etc.—Kg.		Slag.	
1894.....	72·4	$4,408	337·0	$283,080	172,319	$379,102	38,965	$1,558,600	134,305	$3,085,293	2,454	$2,454
1895.....	42·7	3,125	213·1	193,921	209,395	460,669	21,989	879,560	37,01[illegible]	836,494	2,469	2,469
1896.....	88·2	7,453	155·1	139,590	171,741	360,656	28,377	1,135,080	26,854	644,496	4,753	4,753
1897.....	65·6	5,492	236·3	212,670	176,522	247,131	72,605	2,904,200	50,503	1,050,462	8,847	8,847
1898.....	85·2	11,718	244·2	234,432	126,887	197,913	8,241	329,640	68,607	1,427,026	6,861	6,861

Year.	Sodium Carbonate.		Sodium Nitrate. Crude.		Sodium Nitrate. Refined, and Potassium Nitrate.		Stone. Alabaster, Crude.		Stone. Building Stone.		Stone. Limestone. (*d*)	
1894.....	233	$5,585	25·0	$1,230	292	$22,245	114·0	$2,964	39,288	$140,427	55,677	$308,485
1895.....	289	5,206	27·0	1,237	378	26,488	176·1	4,579	26,722	89,905	45,139	225,695
1896.....	279	4,855	51·0	2,351	306	23,751	289·0	7,514	23,580	93,699	37,676	188,380
1897.....	275	5,225	151·0	6,506	344	26,482	268·8	6,989	26,229	117,705	48,972	244,850
1898.....	391	8,599	79·4	3,255	256	17,323	457·0	11,882	35,945	178,024	74,523	367,615

Year.	Stone.—*Continued.* Marble, Crude.		Stone.—*Continued.* Marble and Alabaster.		Sulphur, Crude and Refined.		Tin. Pigs, Bars, etc.		Tin. Manufactures.	
1894.....	72,812	$873,744	51,211	$1,827,062	299,090	$4,306,890	8·5	$3,208	62·6	$27,592
1895.....	75,462	905,544	58,371	1,796,660	317,566	4,128,359	7·2	2,506	74·4	29,822
1896.....	80,750	969,000	68,639	1,937,828	356,370	5,452,461	9·9	3,267	89·1	33,994
1897.....	83,081	996,972	62,750	1,864,862	358,932	6,819,713	29·3	9,699	108·8	41,452
1898.....	88,404	1,060,848	68,150	2,154,092	405,823	8,359,962	34·4	13,760	177·3	80,169

Year.	Zinc. Ore.		Zinc. Oxide.		Zinc. Pigs and Bars.		Zinc. Manufactures.		Other Ores.	
1894.....	123 268	$2,095,556	34·3	$3,430	(*f*)	.....	5·9	$1,561	6,939	$69,390
1895.....	111,227	1,668,405	19·3	1,930	3·9	$312	9·4	2,165	2,072	20,720
1896.....	115,454	1,847,264	48·4	4,840	33·1	2,979	8·4	2,644	15,296	152,960
1897.....	133,125	2,130,000	189·1	18,910	308·7	27,783	63·0	13,143	32,494	194,964
1898.....	130,064	2,601,280	110·2	12,122	155·9	16,837	14·1	3,021	16,925	118,475

(*a*) From the *Movimento Commerciale del Regno d'Italia.* (*b*) Values included under limestone. (*c*) Includes argentiferous ore. (*d*) Includes chalk, kaolin, and certain other non-metallic minerals. (*e*) Included under building stone. (*f*) Not reported.

JAPAN.

THE mineral production of Japan is reported in the subjoined tables.

MINERAL PRODUCTION OF MINES IN JAPAN WORKED BY THE GOVERNMENT. (*a*)

(In metric tons.)

Fiscal Year.	Coal.	Copper.	Copper Sulphate.	Gold—Kg.	Lead.	Pig Iron.	Silver.
1890-91	6,699	24	347	260·49	(*f*)	2,192	7,026·1
1891-92	14,925	17	353	217·72	(*f*)	1,969	5,980·1
1892-93	21,945	275	68	273·77	70	1,417	8,584·1
1893-94	20,123	149	(*f*)	306·12	162	1,186	10,422·9
1894-95	22,506	277	(*f*)	372·15	50	1,210	10,991·1
1895-96	24,947	321	(*f*)	334·11	20	1,189	8,528·1

MINERAL PRODUCTION OF JAPANESE PRIVATE MINES. (*a*) (IN METRIC TONS). (*b*)

Year.	Alum, Refined	Antimony.		Arsenic Kg.	Asphalt, Refined.	Coal.	Copper.	Copperas	Gold.	Graphite.	Iron.
		Ore.	Metal.								
1892....	(*f*)	1,343·5	42·2	1,025·0	(*f*)	3,185,860	20,521	(*f*)	396·19	601	17,583
1893....	(*f*)	1,526·2	122·2	3,324·0	(*f*)	3,329,313	17,874	766·5	445·08	26	15,851
1894....	(*f*)	1,170·6	408·3	5,387·0	(*f*)	4,280,247	19,665	904·4	499·83	1,091	18,258
1895....	(*f*)	1,060·7	640·7	7,343·0	(*f*)	4,747,707	18,826	835·2	563·81	77	24,653
1896 (*g*).	(*f*)	827·6	517·3	6,043·4	(*f*)	5,100,005	20,114	868·6	963·69	215	27,420

Year.	Lead.	Lignite	Manganese Ore.	Mercury Ore. Kg.	Ocher, Red.	Petroleum, Refined.	Salt—*Hektoliters.*		Silver. Kg.	Sulphate of Copper	Sulphur, Refined	Tin.
1892....	819·7	(*f*)	5,027	14,855	(*f*)	2,018	10,202,488	$3,584,093	51,026·8	(*f*)	20,511	41·4
1893....	940·7	(*f*)	14,169	2,141	1·6	1,644	12,006,374	3,666,349	58,934·3	(*f*)	23,930	38·1
1894....	1,414·8	(*f*)	13,368	*e*1,547	(*f*)	*c*22,837	11,411,275	3,483,078	68,798·9	(*f*)	18,787	38·7
1895....	1,927·9	(*f*)	17,142	*e*481	(*f*)	*h*26,965,764	(*f*)	.....	63,824·1	(*f*)	15,557	48·3
1896....	1,957·7	(*f*)	17,967	*e*1,761·6	9·7	*c*31,314	(*f*)	.....	64,302·7	(*f*)	12,540	50·0

(*a*) From *Résumé Statistique de l'Empire du Japon*, Tokio, and special reports from the Japanese Government. The data contained in the table of production of the Government mines are nearly exact, but those contained in the table of private mines, being the figures furnished by the mine owners themselves, are only an approximation and considerably less than the actual figures. (*b*) In making the conversions from the Japanese units to metric tons, hektoliters and dollars, the following relations were employed: 1 kwan = 0·037565 metric tons; 1 koku = 1·8039 hektoliters; 1 yen = $1; 1 kg. = 32·151 troy ounces. (*c*) Crude product. (*d*) Number of liters. (*e*) Metal. (*f*) Not reported. (*g*) Includes production of Government mines. (*h*) Number of liters.

MINERAL IMPORTS OF JAPAN. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 kin = 0·00060104 metric tons; 1 yen = $1.)

Year.	Brass.				Coal.		Coke.		Copper.		
	Crude and Old.		Tubing.	Other M'f'res.					Sheet, Rod, and Old.		Tubes, and oth'r M'f'res.
1893.....	40	$28,071	$21,373	2,420	7,564	$81,707	5,106	$39,325	3	$1,479	$20,566
1894.....	46	33,872	34,990	6,729	37,843	472,757	2,876	22,880	24	11,819	26,373
1895.....	66	42,968	107,569	30,811	70,084	853,080	10,065	90,669	72	37,457	118,909
1896.....	47	20,188	92,300	41,243	50,815	519,380	6,179	42,537	91	46,507	94,306
1897.....	20	10,496	89,170	41,092	70,229	578,570	7,600	56,154	76	43,899	201,883

Year.	Iron.									
	Pig.		Bar and Rod.		Plate and Sheet.		Various Manufactures. (*b*)		Railway Materials	Pipe and Tubes, and other M'f'res (*c*)
1893.....	23,281	$446,477	23,630	$ 975,787	8,160	456,644	17,654	$1,255,639	$147,641	$659,516
1894.....	36,642	743,553	28,098	1,339,034	14,778	889,185	21,334	1,765,302	881,805	1,237,516
1895.....	35,309	673,796	44,175	2,085,684	17,580	1,038,794	23,561	1,853,160	1,253,343	1,332,730
1896.....	39,028	739,556	50,464	2,359,705	27,642	1,659,417	36,306	2,641,885	1,240,480	1,534,259
1897.....	43,688	984,010	55,236	3,046,132	23,883	1,724,913	33,944	2,663,807	2,001,119	1,734,987

Year.	Iron and Steel for Rails.		Steel, including Wire and Wire Rope.		Steel Manufact'res	German Silver.		Kerosene. *Hektoliters.*		Lead	
										Pig.	
1893.....	22,517	677,108	4,524	499,746	$202,301	45·6	38,290	2,055,788	$4,401,041	1,768	$149,328
1894.....	34,657	1,209,205	3,383	462,619	297,876	49·8	41,909	2,257,067	5,135,332	1,688	177,638
1895.....	25,977	925,531	4,619	639,419	377,396	34·2	33,476	1,826,844	4,308,929	3,189	313,632
1896.....	65,387	2,595,450	6,810	1,022,108	446,275	44·4	39,890	2,218,073	6,331,086	2,411	257,383
1897.....	87,076	3,335,005	4,227	577,260	358,255	44·4	41,790	2,472,669	7,667,350	2,298	257,805

Year.	Lead—*Continued.*				Mercury.		Nickel.		Plumbago. Kg.		Tin.	
	Sheet and Tea.		Pipe.									
1893.....	1,779	182,541	$5,428		76	$115,062	42	$60,761	1,615	$232	142	$90,578
1894.....	2,134	255,755	9,986		78	127,541	25	39,451	7,539	1,889	261	179,068
1895.....	1,664	224,546	40,310		75	141,302	29	37,731	10,718	2,160	328	191,936
1896.....	2,147	259,593	144,504		78	130,303	28	30,983	36,030	8,075	270	146,413
1897.....	1,972	269,638	76,910		80	156,527	44	50,321	155,595	15,413	264	145,174

Year.	Tin—Plate.		Yellow Metal.					Zinc.				All Other Metals and M'f'res
			Plates.		Rods.		M'nfactures.	Ingot.		Sheet and Old.		
1893.....	579	$56,267	120	$44,204	3	$1,196	$287	715	$95,106	2,350	$367,584	$188,147
1894.....	3,211	352,675	112	45,755	6	2,330	146	480	66,970	2,681	465,768	200,844
1895.....	3,009	314,044	308	124,461	14	5,851	1,494	998	134,614	3,441	555,695	441,186
1896.....	2,626	251,344	172	74,940	24	9,403	375	1,078	144,240	2,991	512,341	1,773,014
1897.....	5,677	559,654	116	51,419	13	6,334	4,703	836	127,703	4,164	799,930	1,053,682

(*a*) From data specially furnished THE MINERAL INDUSTRY by the Japanese Government. (*b*) Includes old iron, hoop and band, roofing or corrugated and galvanized, nails, and galvanized telegraph wire, iron wire and wire nails (old), and other manufactures. (*c*) Includes also anchors and chain cables, screws, bolts, nuts, etc., and other iron ware.

MINERAL EXPORTS OF JAPAN. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 kin = 0·00060104 metric tons; 1 yen = $1.)

Year.	Antimony.			Brass.			Bronze.			Coal.			
	Metal.		M'factures.	Wire.		M'factures.	Ingot.		M'factures.	For Ships' Use.		Dust.	
1893..	1,316	$183,854	$75,678	151	$52,029	$22,819	416	$91,115	$190,501	417,229	$1,529,070	269,328	$641,758
1894..	1,612	254,261	72,366	282	85,850	43,690	306	70,609	183,688	442,596	1,904,156	238,100	757,039
1895..	1,450	289,935	130,277	177	79,381	44,533	104	27,602	229,291	476,247	2,195,678	283,962	962,710
1896..	624	83,806	104,010	222	108,612	35,526	54	14,570	180,318	588,963	2,636,324	299,102	930,704
1897..	1,577	267,548	63,331	147	72,414	66,388	27	8,373	183,022	582,031	3,229,034	188,132	859,514

Year.	Coal—*Continued.*		Coke.		Copper.							
	Other Kinds.				Ingot.		Old.		Refined.		Slab.	
1893....	842,942	$2,647,084	1,143	$7,219	7,102	$2,033,820	22·0	$5,480	5,115	$1,597,926	3,110	$931,802
1894....	1,047,651	3,917,266	1,197	7,763	5,797	1,799,435	7·0	1,868	5,820	1,945,456	3,651	1,155,863
1895....	1,114,128	4,426,400	4,382	37,487	3,942	1,340,584	0·09	32	5,527	2,123,707	5,067	1,693,377
1896....	1,350,458	5,312,227	3,399	29,114	6,756	2,423,116	0·32	113	6,134	2,461,089	1,644	594,448
1897....	1,366,498	7,457,262	2,950	45,384	2,741	1,075,945	6·81	2,077	6,172	2,680,807	5,045	2,017,947

Year.	Copper—*Continued.*			Iron Wares.	Other Metal Wares.	Manganese Ore.		Sulphur.		Other Mineral Products.	Total.
	Wire.		M'factures.								
1893....	71	$26,169	$71,984	42,555	176,254	18,699	$212,884	10,441	$238,882	$4,202	10,804,023
1894....	69	26,975	134,970	62,142	179,894	17,634	198,810	12,083	244,542	2,245	13,059,614
1895....	69	31,145	101,342	96,875	314,530	16,496	196,599	15,984	296,136	660	14,651,701
1896....	72	33,717	135,079	105,481	358,126	20,986	274,454	12,525	308,588	1,920	16,161,052
1897....	98	46,947	223,234	178,037	485,862	14,665	205,318	9,209	321,341	1,396	19,525,094

(*a*) From data furnished to THE MINERAL INDUSTRY by the Japanese Government.

MEXICO.

THE Mexican Government collects no statistics of production; those compiled by THE MINERAL INDUSTRY, which represent all the important substances, will be found under the respective captions "Lead," "Copper," "Coal," etc.

MINERAL EXPORTS OF MEXICO (*a*) (IN METRIC TONS AND MEXICAN DOLLARS.) (*e*)

Year.	Antimony Ore.		Asphalt.		Building Material.		Coal.		Copper. Ore.		Copper. Ingot.	
1895	600	$14,281	(*b*)	.....	1,355	$5,774	61,666	$255,355	3,006	$523,650	20,429	$3,387,586
1896	3,231	32,093	17	$2.134	9,934	45,418	75,541	312,738	144	5,210	20,659	4,008,404
1897	5,873	71,835	10	282	1,808	7,858	105,298	434,624	1,094	176,391	16,858	3,329,331
1898	5,932	98,815	(*b*)	.....	597	3,561	118,553	486,596	13,146	3,738,738	10,369	2,314,790
1899	10,382	115,292	47	1,623	956	5,923	113,192	458,303	223	54,828	25,293	7,915,827

Year.	Gold. Coin.	Gold. Ore.	Gold. Bullion.	Gold. Cyanide	Gold. Sulphide.	Graphite.		Guano.		Gypsum.	
1895	$175,098	$103,773	$4,920,504	$31,231	$3,026	794	$9,121	1,179	$34,087	1,340	$4,270
1896	261,078	206,874	5,533,789	161,784	44,890	795	7,290	1,001	15,663	2,050	10,250
1897	202,223	365,226	6,220,765	226,986	33,916	759	6,698	1,098	14,615	2,095	7,775
1898	(*b*)	1,087,202	6,493,735	294,780	64,061	(*b*)	.....	2,267	28,983	1,650	8,250
1899	183,474	335,849	7,017,286	115,961	266,782	2,305	22,847	427	3,851	1,050	5,250

Year.	Iron Ore.		Jewels and Precious Stones. *Grams.*		Lead. Ore.		Lead. Base Bullion.		Marble.		Pearls—*Carats*	
1895	1,012	$2,572	81,396	$54,891	528·0	$30,165	50,122	$2,371,574	1,810	$229,495	304	$9,500
1896	1	30	1,357	1,118	167·0	4,820	48,663	1,340,785	2,258	210,836	225	5,500
1897	(*b*)	.....	3,880	5,628	1·9	87	60,029	3,006,821	2,176	185,004	1,420	6,000
1898	21	208	11,241	27,867	(*b*)	.....	60,918	3,291,014	469	46,724	*d*327	3,500
1899	(*b*)	.....	2,489	801	1·0	175	67,441	3,885,747	370	33,220	.....	17,000

Year.	Quicksilver. Kg.		Silver. Coin.	Silver. Ore.	Silver. Slag.	Silver. Bullion.	Silver. Sulphide.	Silver. Cyanide	Tin.	
1895	2,378	$3,910	$18,300,553	$10,977,079	$72,590	$22,178,294	$555,475	$14,649	39·0	$22,911
1896	450	770	18,737,331	9,971,053	64,721	28,565,843	1,495,306	38,049	5·0	2,265
1897	1,845	3,150	21,925,347	11,401,176	39,800	35,775,125	1,663,581	123,246	0·6	40
1898	(*b*)	.....	(*b*)	11,048,358	46,488	37,137,599	1,663,501	257,342	(*b*)	.....
1899	(*b*)	.....	5,580,884	10,766,099	4,810	37,585,911	1,929,085	76,942	.....	.....

(*a*) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. Additional exports in 1898: salt, 1,533 metric tons, $13,449; zinc ore, 1,001 metric tons, $10,016. (*b*) Not reported. (*d*) There was also exported pearls, 42 g., $1,000. (*e*) The average value of the Mexican dollar in New York in 1895, was $0·5218; in 1896, $0·5328; in 1897, $0·4671; in 1898, $0·4641; and in 1899, $0.473.

NORWAY.

THE official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone—27 cents.)

Year.	Apatite. (*b*)		Cobalt Ore.		Copper Ore.		Feldspar. (*b*)		Iron Ore.	
1893.....	1,513	$26,541	623	$12,150	21,907	$175,203	3,506	$17,087	800	$1,485
1894.....	2,086	36,730	89	8,100	20,226	192,240	7,836	35,910	(*c*)	.....
1895.....	1,601	28,760	45	4,050	21,869	218,835	9,780	44,901	1,250	2,349
1896.....	1,106	17,280	29	2,700	29,910	306,747	12,222	54,540	2,000	3,780
1897.....	872	12,960	24	2,700	27,606	303,907	17,392	79,650	3,627	5,670

Year.	Nickel Ore.		Pyrites, Iron and Copper.		Rutile. Kg.		Silver Ore and Native Silver.		Zinc and Lead Ore.	
1893.....	2,397	$6,480	53,574	$190,080	7,000	$1,620	890	$117,045	(*c*)	.....
1894.....	2,355	5,400	70,859	268,110	19,000	4,050	750	100,710	200	$1,890
1895.....	494	1,080	49,005	194,400	28,000	6,100	490	95,850	(*c*)	.....
1896.....	315	.....	60,507	261,900	30,000	9,720	527	108,000	450	3,645
1897.....	*Nil.*	.....	94,484	390,150	32,000	5,400	760	125,280	908	7,290

Metallurgical Production.

Year.	Cobalt—Kg.		Copper and Copper in Mattes.		Gold.	Iron, Pig and Cast.		Iron, Bar and Steel.		Nickel.		Silver—Kg.	
1893.....	5,000	$16,200	786	$168,750	$5,940	336	$3,510		$23,760	113	$70,605	4,773	$117,855
1894.....	(*c*)	.....	907	178,200	1,080	287	3,240	439	19,980	103	63,450	4,760	102,600
1895.....	2,500	8,100	958	184,950	2,160	348	3,942	433	19,440	17	10,590	5,000	105,300
1896.....	(*c*)	.....	1,067	(*c*)	9,450	335	3,780	379	21,060	16	8,100	4,664	(*c*)
1897.....	(*c*)	.....	1,054	(*c*)	675	417	4,590	400	23,760	*Nil.*	.....	5,372	(*c*)
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(*a*) From *Tabeller vedkommende Norges Bergvaerksdrift, Statistisk Aarbog for Kongeriget Norge*, 1896, and from *Meddeleser fra Det Statistiske Centralbureau*, No. 7, 1899, published by Det Statistike Centralbureau, Christiania. There were also produced in 1894, 7,000 kg. of molybdenite, value $3,780, and in 1895, molybdenite 4,000 kg., $2,160, and chrome ore 190 metric tons, $2,295. (*b*) Export returns, which approximately represent the production. (*c*) Not reported.

MINERAL IMPORTS OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone — 27 cents.)

Year.	Borax and Boric Acid. Kg.		Cement and Hydraulic Lime. *Hektoliters.*		Coal, Coke, and Cinders—*Hektoliters.*		Copper and Brass. Plates, Bolts, and Bars.		Copper and Brass. Wares.		Glass and Glassware.	
1894..	32,814	$4,428	81,013	$112,644	13,259,415	$3,938,058	1,362	$338,337	358	$257,877	3,729	$407,268
1895..	40,154	5,427	80,861	109,161	14,012,194	3,783,294	1,262	523,595	417	293,841	3,343	407,592
1896..	38,305	4,671	16,023	137,052	14,201,090	3,642,570	1,074	290,061	479	330,696	3,729	449,577
1897..	44,495	4,941	18,734	174,258	15,374,572	4,234,167	1,140	307,827	591	495,802	4,262	548,802
1898..	71,590	6,804	25,403	239,057	15,409,902	4,659,957	1,064	316,089	807	643,248	3,905	411,615

Year.	Iron. Pig and Cast.		Bars, Hoops, Shipe' Knees, etc.		Ships' Anchors and Cables.		Rails.		Nails, Spikes, Bolts, Screws, etc.		Other Manufactures of Iron.	
1894.....	20,876	$270,540	23,559	$742,500	1,246	$77,409	10,678	$293,706	1,482	$80,028	28,137	$2,195,575
1895.....	19,654	254,718	24,965	790,695	1,152	71,523	10,337	279,099	1,796	96,984	31,387	2,563,218
1896.....	20,201	252,801	26,552	890,433	1,090	67,689	4,315	116,505	1,760	104,544	34,323	2,762,298
1897.....	21,606	289,791	29,038	1,013,364	1,367	87,237	7,637	247,428	2,097	153,738	47,494	4,590,918
1898.....	23,106	339,579	26,208	951,345	1,485	105,057	10,327	362,475	2,087	159,921	62,197	5,450,868

Year.	Steel.		Locomotives and Machines.	Lead, in Pigs and Sheets.		Lead White and Zinc Oxide.		Paraffin Oil, Petroleum, etc.		Potash.	
1894.....	3,058	$165,105	$1,388,907	797	$45,225	931	$80,406	29,824	$791,748	780	$63,207
1895.....	3,654	197,289	1,263,438	657	38,988	1,068	89,424	27,978	962,044	562	47,084
1896.....	2,754	156,141	1,895,994	653	40,527	1,192	99,738	35,823	1,160,676	945	79,083
1897.....	4,350	246,645	2,058,885	848	54,972	1,119	93,636	39,810	1,021,113	919	76,923
1898.....	2,428	144,234	2,437,371	732	51,408	1,491	140,886	36,504	887,031	754	63,099

Year.	Salt. *Hektoliters.*		Saltpeter.		Soda.		Sulphur. (*b*)		Tin, in Blocks, etc.		Zinc, in Plates, Bars, etc.	
1894.....	1,611,769	$609,255	404	$32,697	5,002	$67,527	5,991	$145,584	85	$32,130	864	$81,675
1895.....	1,239,334	468,477	227	17,199	5,234	56,511	7,271	176,688	99	34,884	978	87,129
1896.....	1,179,198	429,813	308	23,247	5,156	55,674	9,847	227,124	142	49,738	1,101	107,055
1897.....	1,645,716	630,963	277	15,708	5,492	59,319	10,701	260,087	236	82,917	1,102	107,082
1898.....	1,273,405	488,214	477	34,776	4,823	52,083	9,589	258,903	177	66,744	1,370	162,729

MINERAL EXPORTS OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Apatite.		Clay Products.				Cobalt Ore—Kg.		Copper.			
			Bricks. *Thousands.*		Earthenware.				Ore.		Ingot.	
1894.....	2,086	$26,612	3,650	$12,825	225	$24,273	68,700	$18,549	14,811	$151,956	397	$80,325
1895.....	1,601	23,760	5,016	20,304	286	30,888	30,000	8,910	20,283	219,051	705	142,722
1896.....	1,160	17,226	10,008	48,654	365	39,420	33,800	8,208	30,367	409,941	565	129,573
1897.....	872	12,960	11,711	59,454	260	27,378	45,000	9,720	15,111	204,012	552	131,081
1898.....	3,593	53,352	15,534	69,677	2	243	25,000	5,400	13,587	183,411	444	113,886

Year.	Copper—*Con.*		Feldspar.		Glassware.		Iodine—Kg.		Iron.			
	Old Metal.								Ore.		Pig and Old.	
1894.....	814	$98,955	7,886	$35,964	1,159	$107,892	10,074	$57,182	1,607	$3,024	4,679	$53,055
1895.....	810	109,404	9,780	44,901	1,153	89,721	2,668	15,201	1,545	2,916	8,188	92,836
1896.....	712	86,454	12,223	54,459	1,231	94,581	1,959	11,097	2,051	3,888	5,493	62,289
1897.....	670	81,459	17,392	79,839	1,432	78,894	2,395	11,151	4,242	8,019	4,631	53,136
1898.....	1,206	156,249	11,355	49,059	841	31,212	5,474	29,555	4,601	8,694	3,844	44,118

Year.	Iron—*Continued.*				Steel.		Machinery.		Nickel.	
	Bars and Hoops.		Nails and Spikes.						Ore—Kg.	
1894.....	462	$17,469	8,081	$625,509	160	$9,504	1,578	$426,087	160	(*c*)
1895.....	19	702	10,408	781,758	133	7,884	1,100	297,055	(*c*)	.....
1896.....	12	459	10,664	747,819	132	7,830	714	192,915	(*c*)	.....
1897.....	56	2,376	9,097	636,552	167	9,986	992	267,840	(*c*)	.....
1898.....	25	1,134	7,270	489,402	158	9,369	507	136,971	30,000	$135

Year.	Nickel—*Con.*		Pyrites.		Silver Ore—Kg.		Stone, Ashlar.		Whetstones.	
	Metal—Kg.									
1894.....	146,384	$40,311	40,771	$132,111	202,200	$10,908	52,344	$247,320	189	$12,771
1895.....	(*c*)	.....	39,710	128,655	136,800	5,535	54,988	237,114	169	11,421
1896.....	(*c*)	.....	41,062	140,265	173,500	7,261	66,233	312,957	205	13,851
1897.....	(*c*)	.....	70,552	238,113	118,900	3,861	74,492	367,065	112	7,587
1898.....	(*c*)	.....	67,502	236,925	79,000	2,555	98,692	546,264	137	9,261

(*a*) From *Statistisk Aarbog for Kongeriget Norge.* (*b*) Inclusive of flowers of sulphur. (*c*) Not reported.

PORTUGAL.

THE mineral statistics of Portugal are summarized in the subjoined table, for which we are indebted to the courtesy of Sehnor Severiano Augusto da Fonseca Monteiro, chief of the Reparticao de Minas, Ministerio das Obras Publicas of Portugal.

The statistics of mineral production in Portugal in years prior to 1894 may be found in THE MINERAL INDUSTRY, Vol. II. and Vol. VI. It will be observed that the mineral industry of Portugal has not yet attained much importance, the output of copper and copper ore forming the chief part of the total. This is principally the production of Mason & Barry Co., Ltd.

MINERAL PRODUCTION OF PORTUGAL. (*a*) (IN METRIC TONS AND DOLLARS; 1 milreis = $1·08.)

Year.	Antimony Ore.		Coal. (Anthracite.) (*b*)		Coal. (Lignite.) (*b*)		Copper Ore.		Copper (Cement).	
1894.....	808	$48,952	10,017	$32,889	9,649	$25,002	321	$3,954	6,924	$498,675
1895.....	753	36,589	8,787	35,820	10,309	22,090	202	3,822	5,055	498,783
1896.....	595	28,917	8,743	17,944	8,000	18,806	436	10,535	3,453	467,897
1897.....	418	28,063	7,996	32,816	9,342	24,215	241	4,496	3,304	408,262
1898.....	245	6,736	10,250	37,032	12,291	52,656	290	12,799	3,149	533,561

Year.	Copper Pyrites.		Copper Pyrites. (Dressed.)		Gold Ore.		Iron Ore.		Lead Ore. (Galena.)		Lead Ore. (Argentiferous.)	
1894.....	44,005	$132,707	203,241	$245,621	(*c*)	.....	620	$335	1,713	$27,445	(*c*)	.....
1895.....	32,814	99,020	162,499	227,025	222·0	$3,159	(*c*)	.....	1,346	27,620	25	$513
1896.....	43,665	139,525	163,775	224,634	(*c*)	.....	(*c*)	.....	1,333	30,236	(*c*)	.....
1897.....	66,473	178,014	210,265	287,254	17·0	12,095	(*c*)	.....	2,180	56,300	(*c*)	.....
1898.....	54,368	133,899	248,218	366,258	6·8	4,687	2,519	3,265	3,242	100,480	(*c*)	.....

Year.	Manganese Ore.		Silver—Kg.		Tin Ore.		Tungsten Ore.	
1894.....	5,246	$38,866	76	$741	26	$6,270	18·9	$2,284
1895.....	1,240	8,250	(*c*)	.....	3	528	12·4	2,786
1896.....	1,494	14,679	(*c*)	.....	6	756	14·0	3,640
1897.....	1,652	11,596	79	2,042	*d* 9	2,550	29·4	7,263
1898.....	907	2,595	119	2,362	102	23,278	59·3	16,714

(*a*) From a report specially furnished THE MINERAL INDUSTRY by Senhor Severiano Augusto da Fonseca Monteiro, Chief of the Department of Mines of the Ministerio das Obras Publicas. (*b*) Consumed in the country. (*c*) No report, probably none. (*d*) Includes metallic tin and cassiterite.

In addition to above, there was reported a production of arsenic amounting to 594 met. tons ($20,360) in 1897, and 751 net tons ($44,764) in 1898.

RUSSIA.

THE official statistics of mineral production, imports and exports, are given in the subjoined tables. The Russian official statistics of production are somewhat tardy in appearance, the latest being those for 1897. Statistics of the production of gold, silver, copper, zinc, petroleum, quicksilver and pig iron for 1898 and 1899 have been reported, however, especially to THE MINERAL INDUSTRY, and will be found under their respective captions elsewhere in this volume.

MINERAL PRODUCTION OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

Year.	Asbestos.		Asphaltum.		Chrome Iron Ore.		Coal.		Cobalt Glance.	Copper.		Gold—Kg.	
1893.....	1,059	$6,836	14,925	$114,800	14,585	$35,101	7,613,724	$9,225,200	3	5,463	$1,586,526	44,865	$21,869,708
1894.....	570	3,400	16,054	176,422	7,537	18,760	8,762,840	11,473,600	9	5,419	1,320,800	42,932	20,985,200
1895.....	1,131	6,800	18,783	143,542	21,013	51,816	9,099,476	12,150,400	3	5,854	1,429,516	41,097	18,812,998
1896.....	1,274	7,666	22,192	145,428	6,682	16,816	9,377,551	13,198,424	4	5,882	1,424,076	37,199	16,845,122
1897.....	1,022	37,444	22,222	184,616	13,433	32,804	11,208,738	15,578,184	3	6,940	1,961,589	38,198	17,255,532

Year.	Graphite.		Iron, Cast.		Kaolin.		Lead.		Manganese Ore.		Petroleum.		Phosphorites.	
1893.....	311	$15,200	1,148,902	$19,100,000	3,197	$12,848	844	$46,922	268,621	$669,100	5,790,692	$2,781,851	13,706	$36,994
1894.....	...	.....	1,332,465	22,200,000	7,742	30,400	743	36,000	243,469	490,800	5,161,552	3,916,800	14,072	31,660
1895.....	...	.....	1,452,337	22,312,800	25,330	101,200	412	20,000	203,081	388,522	7,056,330	11,490,800	6,327	18,682
1896.....	...	.....	1,620,812	25,315,436	6,099	24,575	262	12,775	208,025	398,000	7,105,768	12,588,860	3,776	11,055
1897.....	98	.....	1,880,135	31,636,289	5,036	20,244	450	25,285	370,190	1,102,124	7,881,354	14,623,058	5,917	22,132

Year.	Platinum—Kg.		Pyrites.		Quicksilver		Salt.		Silver—Kg.		Sodium Sulphate.	
1893..	5,094	$917,261	16,271	$51,200	201	$220,484	1,351,056	$2,224,284	9,500	$276,696	5,037	$36,754
1894.....	5,202	890,400	19,801	48,000	196	153,200	1,354,218	1,723,200	7,813	227,200	4,014	28,800
1895.....	4,406	883,960	11,042	31,200	434	371,000	1,540,194	2,383,743	7,878	239,068	4,207	15,612
1896.....	4,930	1,083,600	11,550	32,435	491	432,058	1,346,247	2,079,388	7,813	235,172	5,198	9,600
1897.....	5,602	1,238,040	19,390	54,426	616	541,440	1,551,894	2,711,077	4,783	172,581	7,230	13,352

Year.	Sulphur.		Tin.		Zinc.	
1893.....	589	$7,200	7	$2,501	4,501	$428,604
1894.....	1	18	4	1,440	5,014	367,336
1895.....	190	2,316	8	5,068	5,029	360,000
1896.....	437	5,339	2	476	6,257	458,369
1897.....	574	11,917	2	476	5,868	549,114

(*a*) From the Russian official mining report "Sbornik Statisticheskekh Svedenie o Gornozavodskoi Promyshlennostye Rossie v Zavodskom Godu," St. Petersburg. (*b*) In the Russian reports the quantities are stated in poods and the values in silver rubles. In making the reductions the following relations were employed: 1 pood = ·0163 metric ton; 1 silver ruble = 40 cents.

MINERAL IMPORTS OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 pood = ·01638 metric ton; 1 ruble = 40 cents.)

Year.	Asbestos.		Asbestos Manufactures		Asphalt Rock.		Barytes and Witherite.		Clay, Bauxite and Talc.		Coal.		Coke.	
1893..	38	$1,060	108	$19,924	349	$2,600	4,083	$33,289	52,598	$322,717	1,715,258	$4,776,731	289,978	$968,914
1894..	49	5,819	197	34,192	458	5,654	6,355	40,797	64,428	418,048	1,973,516	5,361,537	290,822	925,762
1895..	107	8,186	241	58,888	566	3,465	7,708	65,297	55,212	327,838	1,934,289	5,298,302	310,113	961,475
1896..	137	5,510	378	85,432	947	8,183	7,053	56,528	e64,135	419,564	1,979,649	5,439,378	364,427	1,146,668
1897..	122	6,598	514	96,829	737	7,392	7,826	73,080	66,980	346,115	2,122,361	4,183,082	399,909	1,675,072

Year.	Copper and Alloys.		Copper and Brass Manufactures.		Glauber Salts.		Gold Bars and Coin—Kg.		Iron.			
									Castings.		Pig.	
1893..	13,814	$4,047,212	d1,480	$882,232	2,649	$88,728	17,019	$5,278,049	3,081	$326,122	160,507	$2,757,083
1894..	8,083	3,352,398	d1,558	935,508	3,644	99,052	(*b*)	.....	4,669	463,440	144,641	2,299,316
1895..	11,679	2,725,159	d1,598	1,045,372	3,303	79,893	(*b*)	.....	4,799	602,644	127,202	2,227,129
1896..	15,726	3,997,911	d1,601	1,100,247	3,619	78,067	(*b*)	.....	7,810	759,814	75,317	1,454,885
1897..	13,590	2,818,348	d1,947	1,159,106	4,550	90,664	(*b*)	.....	10,280	876,949	98,409	1,357,485

Year.	Iron—*Continued.*								Lead.		Magnesite.	
	Tin Plate.		All Other.		Iron and Steel Manufactures.		Steel, All Kinds.					
1893....	22,182	$2,309,751	87,449	$4,363,469	(*b*)	$2,121,317	35,130	$2,151,926	18,712	$1,481,455	720	$17,056
1894....	20,790	1,491,928	188,418	8,689,817	19,171	3,143,306	38,805	3,782,477	11,979	1,620,034	276	5,057
1895....	30,489	2,194,164	218,985	10,182,897	20,214	3,205,418	52,523	3,003,229	29,556	1,820,843	1,127	20,094
1896....	23,144	1,707,033	274,288	12,787,903	26,989	3,771,949	72,509	3,931,378	31,388	1,720,250	1,434	5,510
1897....	2,717	146,557	308,345	11,688,733	26,588	3,696,605	87,413	4,201,118	31,236	1,496,581	2,395	23,418

Year.	Mica—Kg.		Naphtha, Crude.	Oil, Kerosene, Benzine, etc.		Ores, Except Graphite.		Other Metals, Manufactures of.		Phosphorites.		Pyrites.	
1893..	377	$348	$11	362	$28,664	1,022	$2,261	479	$106,941	14,815	$156,082	11,594	$215,734
1894..	2,162	529	10	465	31,249	(*b*)	.....	1,573	982,444	27,006	356,372	36,950	279,076
1895..	2,407	917	(*b*)	537	44,494	1,534	19,800	12,370	3,374,097	(*b*)	.....	22,410	241,393
1896..	3,047	1,301	8	120	7,084	21,468	58,003	4,853	1,987,570	18,156	176,868	20,631	149,582
1897..	(*b*)	.....	48	1,811	57,961	21,887	59,619	.....	.....	(*b*)	.....	33,733	157,396

Year.	Quicksilver.		Salt.		Saltpeter, Chile.		Salts, Stassfurt.		Silver—Kg.		Strontianite and Celestite.		Sulphur.	
1893..	10	$7,312	6,914	$40,946	8,487	$183,537	4,746	$37,528	428,107	$7,249,674	404	$18,814	23,569	$407,818
1894..	11	5,332	10,433	65,692	14,083	357,504	5,190	56,308	(*b*)	.....	575	24,986	19,893	384,752
1895..	7	6,986	10,332	57,400	12,315	294,788	4,463	41,088	(*b*)	.....	34	1,118	19,525	233,233
1896..	15	10,499	10,213	62,492	18,337	414,770	5,553	51,250	(*b*)	.....	216	12,600	19,532	227,428
1897..	10	6,267	8,467	46,710	12,998	241,360	5,491	40,322	(*b*)	.....	881	10,676	20,715	258,160

Year.	Tar, Asphaltic Mastic, etc.		Tin.		Zinc. Ingot and Sheet.	
1893.....	52,598	$322,717	4,007	$868,763	8,146	$909,670
1894.....	3,021	74,075	3,386	688,388	8,289	879,164
1895.....	2,721	77,668	3,417	705,597	9,194	992,106
1896.....	3,011	89,810	4,481	841,436	7,516	766,164
1897.....	6,841	116,869	3,938	808,947	9,546	640,755

(*a*) From the Russian official report "Sbornik Statistícheskekh Svedenie o Gornozavodskoi Promyshlennostye Rossie v Zavodskom Godu," St. Petersburg. (*b*) Not reported. (*d*) Includes bronze manufactures. (*e*) Represents clay used for furnaces.

MINERAL EXPORTS OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS.)

(1 pood = ·01638 metric ton; 1 ruble = 40 cents.)

Year.	Bronze and Manufactures.		Clay, Bauxite and Talc.		Coal.		Copper and Alloys.		Copper and Brass Manufactures.		Gold Coin and Bullion—Kg.	
1893..	(*f*)	$7,042	409	$1,454	8,206	$27,035	49	$14,848	267	136,939	296·5	$92,579
1894..	(*f*)	.....	387	(*f*)	7,899	24,174	51	15,912	259	19,983	229·0	68,578
1895..	(*f*)	10,290	166	1,227	7,544	20,870	47	12,154	.....	86,146	(*f*)	.....
1896..	(*f*)	5,709	1,522	3,542	16,566	25,130	31	8,229	129	62,064	(*f*)	.....
1897..	(*f*)	.....	2,050	4,412	33,142	51,879	29	9,358	.....	120,460	(*f*)	.....

Year.	Iron.						Manganese Ore.		Metals, Other		Naphtha.			
	Castings.		Pig.		All Other.						Crude.		Residuum.	
1893..	481	$52,253	234	$6,340	5,189	$252,647	129,016	$1,512,538	(f)	.....	4,866	$12,745	41,759	$306,944
1894..	822	135,231	224	6,023	5,497	212,178	126,167	1,251,326	247	$36,689	9,574	22,840	53,254	392,590
1895..	298	30,070	192	7,040	2,312	117,207	146,840	1,194,085	202	30,808	9,266	22,856	54,658	403,076
1896..	439	39,011	184	7,950	4,534	239,499	165,600	1,215,399	127	12,150	2,942	40,514	54,451	449,878
1897..	547	60,019	238	6,708	4,150	184,139	187,396	1,138,824	(f)	.....	9,605	47,576	62,945	565,210

Year.	Oil, Kerosene, Benzine, etc.		Ores of Metals and Minerals, Except Graphite.		Paraffin.		Phosphorites.		Platinum, Crude—Kg.		Quicksilver.	
1893..	790,007	$8,416,610	66	$1,189	(f)	.....	7,329	$109,553	6,000	$447,685	266	$383,317
1894..	819,485	6,778,882	1,183	19,553	(f)	.....	8,892	28,800	2,000	270,190	122	153,111
1895..	709,406	5,642,097	432	10,260	b1	$370	10,918	134,896	(f)	.....	140	164,398
1896..	845,751	8,129,730	20,026	40,071	b93	3,746	8,840	107,898	4,700	777,900	430	383,629
1897..	917,216	9,507,619	16,890	40,874	b740	15,516	2,917	36,389	4,767	700,780	643	575,409

Year.	Salt.		Silver—Kg.		Slags.		Steel.		Zinc.			
									Ore.		Ingot and Sheet.	
1893..	9,774	$34,484	108,087	$1,754,856	3,967	$12,745	760	$59,534	248	$3,708	248·0	$1,219
1894..	6,457	21,817	167,010	2,858,582	8,994	15,776	283	24,029	40	1,414	27·0	2,963
1895..	7,023	17,790	(c)	.....	5,173	11,334	167	14,855	208	2,890	1·0	446
1896..	6,814	16,461	(c)	.....	5,606	11,274	173	15,338	194	4,390	3·5	264
1897..	9,022	21,668	(c)	.....	9,702	22,447	160	17,579	.....	.....	13·4	1,552

(a) From the Russian official report "Sbornik Statisticheskekh Svedenie o Gornozavodskoi Promyshlennosty Rossie v Zavodskom Godu," St. Petersburg. (b) Includes vaseline. (c) Included under precious metals. (f) Not reported.

SPAIN.

THE official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Aluminous Earths.		Antimony Ore.		Arsenical Pyrites.		Asphalt Rock.		Barytes.		Coal.					
											Anthracite.		Bituminous.		Lignite.	
1894....	310	$1,549	15	$300	68	$136	985	$2,009	541	$2,699	(*b*)	.....	1,659,274	$2,327,677	48,460	$63,266
1895....	240	1,198	44	700	(*b*)	.....	790	1,581	494	2,255	10	$10	1,739,075	2,648,367	44,708	58,333
1896....	320	1,600	54	1,385	(*b*)	.....	1,117	2,234	345	1,720	14,995	26,788	1,852,947	2,806,654	55,413	60,261
1897....	409	2,046	354	6,718	(*b*)	.....	1,656	3,312	429	2,449	8,758	15,745	2,010,960	3,409,686	54,232	54,027
1898....	505	2,525	130	2,149	230	575	2,383	4,772	364	2,074	20,105	36,190	2,414,127	4,147,333	66,422	93,310

Year.	Cobalt Ore.		Fluorspar.		Copper Ore.						Gold and Silver Ore.		Graphite.	
					Argentiferous.		Pyrites.		Copper and Cobalt.					
1894....	52	$624	18	$230	(*b*)	.....	2,445,242	$2,638,819	853	$20,472	(*b*)	.....	10	$30
1895....	7	84	27	405	(*b*)	.....	2,701,661	2,829,422	410	6,560	918	$4,593	(*b*)	.....
1896....	18	1,800	3	45	*c*157,365	$146,596	2,200,919	2,205,843	992	23,808	854	3,056	(*b*)	.....
1897....	13	3,400	2	36	*c*18,488	35,922	2,161,182	2,124,893	(*b*)	.....	2,456	9,873	(*b*)	.....
1898....	(*b*)	.....	5	75	208	1,831	2,299,444	2,735,261	(*b*)	.....	555	2,165	10	18

Year.	Iron Ore.				Iron Pyrites.		Kaolin. (China Clay.)		Lead Ore.			
	Non-Argentiferous.		Argentiferous.						Non-Argentiferous		Argentiferous.	
1894....	5,352,353	$3,995,912	796	$826	60,000	$30,000	247	$1,806	140,841	$2,208,839	181,715	$5,497,316
1895....	5,514,339	4,183,059	572	572	60,267	30,335	836	1,579	124,195	2,052,812	181,433	5,042,288
1896....	6,762,582	5,013,404	3,581	3,581	100,000	50,000	1,240	2,446	104,160	1,884,807	182,565	6,121,173
1897....	7,419,768	5,457,327	5,559	19,347	100,000	50,000	6,294	6,717	110,469	2,058,611	186,692	6,735,013
1898....	7,197,047	6,232,484	24,190	68,216	70,266	35,135	5,445	5,518	150,472	4,948,056	244,068	8,023,772

Year.	Manganese Ore.		Mineral Waters.		Nickel Ore.		Ocher.		Phosphorites.		Quicksilver Ore.	
1894....	340	$522	169,107	$65,862	7·2	$144	120	$240	40	$80	30,081	$1,200,069
1895....	10,162	17,043	15,193,865	75,538	(*b*)	.....	203	787	1,040	2,081	33,792	1,248,815
1896....	38,265	58,732	15,738,142	92,485	100	880	212	850	770	3,080	34,959	1,377,396
1897....	100,566	126,250	16,180,585	115,874	(*b*)	.....	200	800	2,084	16,672	32,378	1,328,443
1898....	102,228	343,045	15,610,222	122,265	(*b*)	.....	200	800	4,500	45,003	31,361	1,252,024

Year.	Salt.		Silver Ore.		Soapstone.		Sulphate of Soda.		Sulphur, Crude Rock.		Tin Ore.	
1894....	186,121	$228,374	7,411	$257,041	3,108	$13,603	312	$408	10,865	$28,248	26	$2,678
1895....	326,320	952,098	16,299	75,836	2,347	6,219	460	690	8,481	22,049	17	1,500
1896....	521,751	1,153,880	1,230	129,943	756	1,166	(*b*)	.....	26,204	45,433	*d*2,348	5,908
1897....	508,606	1,159,294	982	101,797	3,601	21,341	(*b*)	.....	18,845	32,598	*d*2,378	5,795
1898....	479,358	1,025,682	767	106,094	2,613	14,504	(*b*)	.....	*e*105,757	198,198	4	468

Year.	Topaz of Hinojosa. Kg.		Wolfram Ore		Zinc Ore.	
1894....	75	$1,549	21	$1,070	58,760	$395,790
1895....	67	1,385	14	710	54,109	370,007
1896....	80	12,860	31	1,560	64,828	407,670
1897....	44	751	10	309	73,848	363,846
1898....	90	1,454	37	5,325	99,836	991,386

(a) Figures for 1897 are from the *Reports of the Comisión Ejecutiva de Estadística Minera*. The figures for 1894, 1895, 1896 and 1898, are from the official *Reports of the Junta Superior Facultativa de Minas*, Madrid. There was also produced in 1896, lead and zinc ores, 400 metric tons, $400; in 1897, gold ore, 456 metric tons, $1,850; lead and zinc ores, 40 metric tons, $40; in 1898, lead and zinc ore, 38 tons, $38; gold, $2,338. (b) Not reported. (c) Represents non-argentiferous copper ore. (d) Undressed ore. (e) 70,814 tons, $152,309, of this product is the contents of copper pyrites from Huelva.

METALLURGICAL PRODUCTION OF SPAIN. (a) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Arsenic Sulphide.		Asphalt, Refined.		Cement, Hydraulic.		Coal, Briquettes.		Coke.	
1894.....	184	$13,224	905	$11,761	132,645	$376,204	209,776	$852,231	149,905	$748,028
1895.....	184	18,390	790	10,275	149,197	324,214	342,985	1,370,987	249,058	1,046,033
1896.....	271	27,100	1,285	16,705	130,738	281,144	343,432	1,373,061	288,523	1,302,716
1897.....	244	29,256	1,878	22,532	159,439	325,202	332,272	1,318,532	755,394	3,167,416
1898.....	111	13,320	2,354	29,626	164,862	358,501	369,418	1,277,425	768,151	3,211,900

Year.	Copper.						Iron, Pig.		Iron, Forged.	
	Fine.		In Matte.		In Precipitate.					
1894.....	785	$156,947	16,872	$1,012,295	29,548	$4,136,687	123,798	$1,699,682	54,214	$2,209,878
1895.....	7	1,520	5,756	345,385	31,725	4,439,889	179,752	2,575,065	48,462	2,053,822
1896.....	6	1,200	16,378	982,702	29,873	4,155,204	100,786	1,481,162	53,793	2,224,443
1897.....	7	1,400	16,120	967,210	29,652	4,138,725	146,940	2,267,403	80,894	3,190,139
1898.....	593	118,623	16,024	961,438	29,703	4,158,418	113,492	2,059,232	65,900	2,826,743

Year.	Steel.		Lead.				Quicksilver.		Silver—Kg.	
			Non-Argentiferous.		Argentiferous.					
1894.....	92,854	$3,584,400	64,189	$2,910,458	88,433	$7,385,797	1,610	$1,286,961	192,745	$4,195,184
1895.....	56,901	2,189,222	76,808	3,637,932	83,978	6,916,320	1,506	1,343,417	58,546	1,360,996
1896.....	68,126	2,666,843	82,215	4,396,544	84,802	5,808,318	1,524	1,287,144	64,554	1,507,441
1897.....	66,007	2,873,930	75,112	4,343,187	91,258	6,312,109	1,728	1,564,333	71,168	1,555,696
1898.....	50,362	2,332,019	78,370	5,188,394	88,981	6,026,450	1,691	1,593,921	76,295	1,865,156

Year.	Sulphate of Soda.		Sulphur.		Zinc.					
					Slabs.		Sheet.		Calamine, Calcined.	
1894.....	174	$3,059	3,417	$58,086	3,052	$360,276	2,421	$421,354	17,768	$11,514
1895.....	227	2,946	2,231	37,930	3,149	302,266	2,487	373,125	19,169	(b)
1896.....	(b)	.....	1,900	36,004	3,485	390,286	2,648	405,175	23,108	(b)
1897.....	(b)	.....	3,500	70,000	3,907	445,212	2,337	378,594	(b)	(b)
1898.....	(b)	....	3,100	68,400	4,300	602,000	1,731	322,040	(b)	(b)

(a) The figures for 1898, except those for briquettes, copper, coke, iron, steel, and zinc, are from the *Reports of the Comisión Ejecutiva de Estadística Minera;* all others are from the official *Reports of the Junta Superior Facultativa de Minas.* (b) Not reported.

MINERAL IMPORTS OF SPAIN. (a) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Asphalt and Pitch.		Alkaline Carbonates		Coal.		Coke. (b)		Gold Bars—Kg.	
1894.....	29,967	$599,335	28,372	$1,248,390	1,614,839	$8,720,129	225,902	$1,219 873	.....	.....
1895.....	33,766	675,316	27,591	1,219,755	1,515,506	8,190,270	255,043	1,377,232	.....	.....
1896.....	35,401	566,427	27,014	1,188,641	1,447,345	8,262,168	234,033	1,115,516	337	$235,900
1897.....	23,324	368,700	28,428	1,210,426	1,633,333	8,837,596	214,763	1,159,731	268	187,600
1898.....	28,352	526,058	26,099	1,148,371	1,215,554	6,482,757	231,467	1,057,171	2	1,440

Year.	Gold Coin	Iron, Pig.		Iron, Bar.		Iron and Steel, Forged.		Petroleum, Crude.		Silver Bars. Kg.	
1894.....	$634,394	25,308	$354,240	9,648	$419,706	23,142	$969,009	44,052	$1,145,362	1,935	$54,181
1895.....	119,945	12,395	173,389	7,768	354,289	18,233	753,267	46,592	1,211,397	4,286	120,008
1896.....	16,231	8,577	120,087	13,861	607,589	26,463	1,398,827	34,427	1,233,345	5,576	150,812
1897.....	12,149	1,855	25,972	13,558	571,043	24,037	1,472,690	34,972	1,259,011	243,540	6,341,894
1898.....	(c)	1,575	25,206	(c)	.....	2,583	113,640	32,468	1,168,843	152,364	3,961,484

Year.	Silver Coin.	Soda Nitrate.		Sulphur.		Tin Plate.		Tin Ingots.	
1894.....	$4,345,859	24,110	$1,446,690	5,668	$1,248,390	3,149	$264,543	1,027	$472,695
1895.....	4,776,929	26,385	1,583,129	6,813	1,219,755	1,241	104,223	825	379,625
1896.....	20,190,400	26,841	1,610,495	7,744	185,870	1,071	89,977	1,080	432,001
1897.....	22,325,312	34,862	2,091,729	5,310	128,883	565	46,098	923	369,359
1898.....	(c)	38,214	2,217,002	5,664	135,958	653	52,237	834	325,922

(a) The figures for 1894, 1895 and 1898 are from the *Anuario de la Mineria Metalurgia y Electricidad de España*, while those for the years 1896 and 1897 are from the *Revista Minera*, Madrid, Feb. 18, 1898. (b) In terms of coal. (c) Not reported.

MINERAL EXPORTS OF SPAIN (a) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Antimony Ore.		Coal.		Copper Ore.		Iron Ore.		Iron Pyrites.	
1894.....	46	$2,728	16,738	$80,341	541,321	$4,871,887	4,975,727	$8,956,309	510,129	$1,020,257
1895.....	26	1,566	8,320	44,927	530,075	4,796,335	5,175,260	9,315,468	490,709	981,419
1896.....	119	7,190	4,295	23,205	629,546	5,665,918	6,289,257	11,320,670	464,882	920,977
1897.....	20	1,176	2,539	13,300	822,570	5,613,482	6,884,244	13,180,064	217,545	569,287
1898.....	50	3,002	2,772	14,881	912,238	6,202,821	6,478,488	14,252,674	260,016	624,039

Year.	Lead Ore.				Manganese Ore.		Salt.		Silver—Coin, Jewelry, etc.	
	Argentiferous.		Non-Argentiferous.							
1894.....	11,457	$965,345	707	$35,327	7,321	$68,815	228,691	$686,071	22	$715,936
1895.....	8,349	534,352	1,979	94,516	30,007	282,269	253,391	760,174	277	8,397,770
1896.....	c6,270	392,668	(b)	.....	87,835	827,620	254,752	764,257	(b)	.....
1897.....	c8,267	462,595	(b)	.....	95,756	1,091,621	235,871	707,616	(b)	.....
1898.....	7,327	407,966	(b)	.....	138,062	1,573,907	219,301	657,703	(b)	.....

Year.	Steatite.		Sulphur.		Zinc Ore.		Cement, Hydraulic		Copper, in Matte and Precipitate.	
1894.....	325	$13,000	244	$6,345	34,119	$337,731	2,409	$14,451	32,061	$3,496,283
1895.....	1,415	45,272	83	2,152	29,360	267,957	2,807	16,841	36,210	4,218,024
1896.....	(b)	.....	(b)	.....	36,656	308,272	(b)	.....	46,417	4,885,213
1897.....	(b)	.....	(b)	.....	41,040	353,121	(b)	.....	51,487	7,373,928
1898.....	(b)	.....	(b)	.....	4,551	500,577	(b)	.....	48,884	6,846,854

Year.	Iron and Steel.		Lead, Pig.				Quicksilver.		Tin.		Zinc.	
			Argentiferous.		Non-Argentiferous.							
1894.....	57,257	$1,590,853	91,511	$5,490,639	68,806	$3,467,557	870	$974,117	16	$7,756	2,680	$267,997
1895.....	79,157	3,654,963	87,299	5,587,115	67,187	3,288,488	1,330	1,489,194	10	4,937	1,367	136,690
1896.....	23,085	283,270	d166,124	9,500,935	(b)	.....	1,559	1,746,154	(b)	.....	3,560	356,060
1897.....	43,612	610,776	d171,774	10,122,220	(b)	.....	1,742	1,436,293	(b)	.....	2,170	239,716
1898.....	46,129	738,058	d179,885	11,015,702	(b)	.....	1,742	1,881,352	(b)	.....	4,551	500,577

(a) From the *Estadistica Minera de España*, except 1896, 1897 and 1898, which are from the *Anuario de la Mineria Metalurgia y Electricidad de España*, Madrid. (b) Not reported. (c) Includes non-argentiferous lead ore. (d) Includes non-argentiferous.

SWEDEN.

THE official statistics of mineral production, imports and exports are summarized as follows:

MINERAL PRODUCTION OF SWEDEN. (*a*) (IN METRIC TONS.)

Year.	Alum.		Clay (Fire).		Coal.		Copper Ore.		Feldspar.		Gold Ore.		Graphite.	
1894.....	261	(*c*)	129,617	(*c*)	195,950	(*c*)	25,710	(*c*)	(*c*)	.....	(*c*)	.....	107	(*c*)
1895.....	286	(*c*)	120,385	(*c*)	223,652	(*c*)	26,009	(*c*)	(*c*)	.....	459	(*c*)	7	(*c*)
1896.....	334	$9,849	120,426	(*c*)	225,848	$435,891	24,351	$83,135	12,789	$39,417	736	$5,918	14	$491
1897.....	131	3,826	112,283	(*c*)	224,343	434,710	25,207	92,712	19,298	53,992	1,662	9,214	99	3,240
1898.....	153	4,543	131,891	(*c*)	236,277	465,936	23,335	99,749	20,737	64,572	2,136	6,186	50	1,620

Year.	Iron Ore.		Iron Ore. (Bog)		Manganese Ore.		Red Ocher.		Pyrites.		Silver and Lead Ore.		Zinc Ore	
1894.....	1,927,212	(*c*)	689	(*c*)	3,359	(*c*)	1,564	(*c*)	656	(*c*)	14,825	(*c*)	47,029	(*c*)
1895.....	1,904,662	(*c*)	2,691	(*c*)	3,117	(*c*)	1,291	(*c*)	221	(*c*)	12,045	(*c*)	31,349	(*c*)
1896.....	2,039,019	$2,518,175	925	(*c*)	2,056	$7,251	(*c*)	.....	1,009	$2,914	15,381	$75,252	44,041	$230,738
1897.....	2,080,119	2,700,639	1,047	(*c*)	2,749	12,810	(*c*)	.....	517	1,496	10,068	63,123	56,696	494,742
1898.....	2,302,546	2,969,986	968	(*c*)	2,358	11,142	(*c*)	.....	386	1,037	6,743	58,631	61,627	603,647

Year.	Cobalt Oxi. Kg.		Copper.		Copper Sulphate.		Copperas.		Gold.—Kg.		Iron.			
											Pig.		Bloom.	
1894.....	1,580	(*c*)	350	(*c*)	723	(*c*)	362	(*c*)	93·6	(*c*)]	462,809	(*c*)	204,517	(*c*)
1895.....	(*c*)	.....	216	(*c*)	1,195	(*c*)	94	(*c*)	85·2	(*c*)	462,920	(*c*)	188,736	(*c*)
1896.....	(*c*)	.....	249	$56,724	1,506	$105,300	191	$2,483	114·5	$70,806	494,418	$8,233,999	168,396	$4,772,077
1897.....	700	$2,457	289	68,645	1,315	99,860	282	3,235	113·3	75,517	538,197	9,467,855	189,633	5,045,705
1898.....	3,001	10,125	235	59,089	1,165	78,840	124	1,783	125·9	106,318	531,766	9,414,128	198,923	5,407,843

Year.	Steel.						Iron and Steel Bars, Rods, Sheets, etc.		Lead.		Silver. Kg.		Sulphur.	
	Bessemer.		Martin.		Crucible.									
1894....	83,322	(*c*)	84,003	(*c*)	510	(*c*)	261,492	(*c*)	330	(*c*)	2,870	(*c*)	36	(*c*)
1895....	97,320	(*c*)	99,259	(*c*)	598	(*c*)	284,504	(*c*)	1,256	(*c*)	1,188	(*c*)	(*c*)	.....
1896....	114,120	$3,091,384	142,301	$3,851,974	604	$53,318	308,132	$12,086,504	1,530	$76,033	2,082	$44,263	77	(*c*)
1897....	107,679	3,091,109	165,836	4,652,524	691	73,540	304,537	12,624,486	1,480	76,876	2,218	41,471	(*c*)	.....
1898....	102,254	2,836,079	160,706	4,512,212	1,013	106,318	299,846	11,943,458	1,559	102,464	2,033	38,887	50	$1,620

(*a*) From *Bidrag till Sveriges Officiela Statistik, Bergshandteringen.* In 1894 there was also an output of 0·08 ton of antimony ore, and in 1895, 1·5 tons of antimony ore. (*c*) Not reported.

MINERAL IMPORTS OF SWEDEN. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Alabaster (*b*)		Alum.		Aluminum Sulphate.		Ammonia, Caustic.		Ammonium.							
									Carbonate.		Chloride.		Nitrate.		Sulphate.	
1894..	0·555	$749	89·572	$2,660	266	$3,590	64	$5,145	65	314,013	57	$12,230	.....	$64	265	$21,484
1895..	0·567	765	92·916	2,760	348	4,704	76	6,147	74	16,045	84	18,181	11	4,258	39	3,133
1896..	0·178	58	74·808	2,223	620	8,494	81	7,279	79	17,114	88	18,963	11	4,609	88	7,122
1897..	3·616	4,882	103·442	3,072	733	9,896	59	5,605	109	20,516	110	20,873	42	9,013	67	4,536
1898..	5·973	8,064	135·887	4,036	968	13,074	105	9,886	99	18,723	101	19,168	12	2,565	81	5,477

Year.	Antimony, Crude.		Arsenious Acid.		Asbestos. (c)		Asphalt. (d)		Barytes.		Boric Acid.		Borax.	
1894....	60·480	$11,431	24	$2,604	68	$18,383	2,033	$28,436	172	$2,313	64	$10,318	108	$14,561
1895....	80·603	15,234	36	3,910	68	18,278	3,256	35,161	240	12,989	54	8,674	111	14,909
1896....	62·552	11,822	33	3,587	116	31.407	4,092	44,190	208	16,084	73	11,807	128	17.243
1897....	58·482	11,053	33	3,523	119	32,151	5,458	58,947	270	14,595	56	9,006	175	18,952
1898....	52·604	9,936	33	4,390	112	30,359	5,409	58,420	299	16,140	75	12,071	196	21,148

Year.	Bromine and Bromides of Potassium and Sodium.		Calcium Chloride.		Cement.		Chalk. White—Unground. *Hektoliters.*		Chalk. Other Kinds. (e)		Chemico-Technical Preparations N.E.S. (f)	Clay Products. Brick & Tiles.
1894....	3·977	$3,221	1,102	$47,603	2,425	$22,919	2,254	$244	100	$862	$64,261	$105,233
1895....	3·999	3,239	1,250	53,982	4,270	40,354	4,466	482	109	1,579	69,962	117,188
1896....	4·334	3,511	1,600	69,116	2,901	27,411	6,148	664	146	1,143	87,835	196,432
1897....	5·549	4,495	1,676	72,398	1,826	17,259	14,368	1,552	142	1,069	100,021	263,379
1898....	5·401	5,687	1,668	72,048	1,656	15,652	7,016	758	308	1,313	115,499	359,374

Year.	Clay Products.—*Cont.* Clay.		Clay Products.—*Cont.* Porcelain.		Coal—*Hektoliters.* (g)		Copper, and Copper in Alloys and Metals.		Emery.		Glass and Glassware. (h)		Gold Bars, and Gold M'f'res. Kg.		Gold Coin.
1894..	$38,844	235	$175,379	24,148,786	$8,802,232	3,147	$779,042	71	$9,543	1,427	$411,182	1,638	$1,099,048	$530,923
1895..	69,178	277	211,083	23,895,370	8,387,275	2,980	738,512	125	16,824	1,619	452,276	89	59,354	11,606
1896..	79,655	327	247,878	24,806,994	8,402,736	4,037	947,930	104	14,004	2,477	447,352	1,161	107,001	608
1897..	80,010	362	273,784	28,003,094	9,829,086	4,944	1,156,202	128	17,296	1,618	507,491	4,267	2,086,848	941
1898..	109,738	296	213,572	29,905,639	12,112,784	5,227	1,220,805	131	17,649	2,560	551,169	3,998	1,875,585	2,398

Year.	Graphite.		Gypsum. (j)		Iron and Steel. Crude.		Iron and Steel. Manufactures.		Lead.		Lime. *Hektoliters.* (k)		Litharge.	
1894....	133	$7,173	3,578	$27,655	34,201	$734,304	$2,616,178	1,750	$163,197	6,000	$2,155	93	$12.611
1895....	134	7,244	4,112	30,347	40.532	878,005	2,549,183	1,624	149,729	4,436	1.628	117	15,760
1896....	135	7,311	4,910	37,274	34,549	878,333	3,703,338	1,911	180,749	7,768	2,619	150	20,202
1897....	158	8,519	7,260	75,270	89,606	1,420,175	3,785,043	2,098	206,076	20,050	7,389	199	26,925
1898....	167	9,008	7,979	66,187	76,832	1,326,910	4,641,613	2,139	207,342	23,079	8,724	160	21,592

Year.	Nitric Acid.		Phosphorus.		Platinum. Kg.		Potash.		Potassium. Chloride.		Potassium. Cyanide.		Quicksilver.	
1894....	26	$2,109	56·658	$68,839	14	$2,646	1,744	$164,835	301	$40,595	1·315	$1,065	5·896	$6,686
1895....	36	2,933	71·407	86,760	42	7,938	1,979	187,019	561	113,699	1·457	1,180	6·318	7,166
1896....	32	2,574	52·482	63,766	34	6,426	1,933	182,688	211	45,508	2·122	1,719	5·194	5,890
1897....	41	3,285	57·972	70,436	63	11,907	1,432	135,370	363	58,817	2·922	2,367	3·125	3,544
1898....	34	3,252	66·466	80,756	49	9,261	1,112	105,119	259	28,006	2·604	1,757	2·631	2,983

Year.	Salt. Common. *Hektoliters.*		Salt. Refined.		Saltpeter. (l)		Sand.	Silver and Man'factures Kg.		Silver Coin.	Soda.	
1894....	953,960	$386,354	1,922	$77,847	8,436	$452,709	$14,062	2,535	$51,334	$14,803	12,565	$237,482
1895....	1,101,242	446,003	2,535	102,685	9,388	446,851	15,440	3,050	65,880	114,340	12,711	240,240
1896....	872,459	318,011	3,673	148,755	12,518	565,900	30,428	7,375	217,141	204,691	11,425	215,932
1897....	869,619	411,548	3,055	148,479	12,531	602,709	19,509	20,557	473,801	136,823	14,626	276,417
1898....	878,823	403,380	2,188	106,332	15,419	685,657	29,169	21,696	442,622	191,766	11,917	225,280

Year.	Soda, Caustic.		Sodium Sulphate. (i)		Stone. (m)	Sulphur.		Sulphuric Acid.		Tin.		Tin and Lead Ashes.		Tin Salts.
1894....	245	$13,222	4,642	$125,340	$36,601	3,142	$197,850	418	$8,472	413	$161,532	3	$3.335	3·982	$1,075
1895....	1,043	56,319	7,420	200,332	35,363	7,140	173,491	772	15,638	462	162,085	7	7.235	4 091	1,105
1896....	908	49,051	8,486	229,121	122,525	11,369	276,265	615	12,455	551	170,307	11	12,132	4·437	1,198
1897....	625	33,764	11,394	153,683	227,476	9,723	236,269	1,418	28,709	541	196,478	16	29.709	3·823	1,032
1898....	575	31,064	11,544	172,435	103,200	10.837	263,341	1,742	35,273	595	224,890	25	27,455	3·874	1,046

Year.	Vitriol, All Kinds.	Zinc.	
1894....	$12,212	1,783	$165,582
1895....	4,554	2,216	198,289
1896....	11,028	2,275	230,865
1897....	5,805	2,551	261,249
1898....	6,512	3,030	343,014

(*a*) From *Bidrag till Sveriges Officiela Statistik.* (*b*) Wrought stone. (*c*) Includes raw product and manufactured articles. (*d*) Natural and artificial asphalts. (*e*) Ground chalk, pastels, etc. (*f*) N. E. S.—*Not elsewhere specified.* (*g*) Includes coal-dust. (*h*) Exclusive of powdered glass. (*i*) Includes also sodium bisulphate. (*j*) Raw, ground and calcined gypsum. (*k*) Slaked and unslaked lime. (*l*) Includes both soda and potash salts. (*m*) Building stone, etc.

MINERAL EXPORTS OF SWEDEN. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Alum.		Aluminum Sulphate. Kg.		Ammonium Sulphate.		Antimony, Crude.		Asbestos. Kg.		Cement.		Chalk. Unground. *Hektoliters.*	Chalk. All Other.		
1894....	77	$2,294	10,000	$135	41	$3,343	1·2	$230	14,032	$3,789	29,806	$242,161	244	$26	3,355	$18,188
1895....	38	1,140	(*b*)	....	(*b*)	.....	2·3	440	28	7	31,029	251,333	111	12	3,458	18,815
1896....	40	1,184	(*b*)	....	100	8,138	0·8	144	2,040	551	22,901	186,228	164	18	4,382	23,747
1897....	54	1,604	(*b*)	....	180	12,150	0·8	156	1,348	364	27,112	219,611	1,188	128	4,422	24,032
1898....	32	947	(*b*)	...	36	2,491	4·7	896	1,055	285	28,676	232,276	365	39	4,844	26,209

Year.	Chemico-Technical Preparations.	Clay.	Clay Manufactures.	Coal.		Copper Ore.		Copper and Copper in Alloys and Metals.		Glass, All Kinds.		Graphite. Kg.	
1894....	$40,627	$17,360	$402,864	84	$153	(*b*)	.....	407	123,663	7,441	$1,131,823	4,336	$234
1895....	39,667	17,360	467,389	52	182	(*b*)	.....	2,283	269,066	7,150	1,221,319	9,749	525
1896....	34,532	28,809	589,159	141	594	1,094	$73,856	1,911	218,849	10,674	1,378,125	3,500	189
1897....	40,351	28,931	684,340	74	325	(*b*)	.....	983	425,089	9,399	1,134,495	7,215	390
1898....	80,377	43,005	641,730	496	2,513	1,102	7,441	1,346	293,159	9,838	1,143,037	9,108	492

Year.	Gypsum and Manufactures.		Iron Ore.		Iron and Steel. Unwrought.		Iron and Steel. Manufactures.		Lead and Manufactures.		Lime. *Hektoliters.*	
1894....	27·2	$21,308	831,395	$1,571,337	252,452	$7,009,258	$1,342,845		768	$42,158	34,172	$11,444
1895....	20·0	12,260	800,452	1,512,855	311,866	8,653,212	1,794,270		1,380	74,712	40,287	13,456
1896....	9·3	7,436	1,150,695	1,905,550	304,138	9,320,969	1,975,068		1,182	67,750	102,787	29,453
1897....	9·6	5,727	1,400,801	2,779,890	279,525	9,175,910	2,373,280		1,473	91,998	106,053	31,925
1898....	27·3	2,609	1,439,860	2,942,930	301,192	9,399,827	1,644,096		570	36,090	123,989	40,401

Year.	Peat.		Phosphorus. Kg.		Potassium Chlorate.		Salt, Refined. Kg.		Sand.	Silver. Bullion—Kg.		Silver. Manufactures—Kg.	
1894....	1,411	$3,810	1,585	$1,926	72·3	$19,522	1,781	$72	$619	2,579	$52,225	6	$324
1895....	1,143	3,087	885	1,075	436·4	88,377	1,661	67	511	751	15,918	10	540
1896....	1,452	3,920	1,510	6,795	258·7	47,958	890	34	2,232	819	17,359	14	756
1897....	1,816	4,902	1,627	1,977	462·5	74,927	1,424	69	969	329	6,396	119	6,435
1898....	1,616	4,364	4,065	4,968	506·4	82,041	216	11	8,336	180	2,492	238	12,744

Year.	Soda.		Sodium Sulphate.		Stone. (*c*)	Sulphur.		Sulphuric Acid.		Tin and Lead Ash.		Tin. Ingot.		Tin. Manufactures—Kg.	
1894....	872	$16,478	10·3	$279	$1,424,279	16	$891	2·6	$77	1·8	$1,992	18·4	$7,214	800	$808
1895....	959	18,128	12·9	348	1,420,508	11	271	3·1	85	7·6	8,245	13·3	4,663	736	459
1896....	772	14,599	6·8	184	(*b*)	9	214	6·6	178	7·6	8,197	18·9	6,367	2,996	1,636
1897....	686	12,970	3·4	46	2,062,011	11	258	7·6	205	1·7	1,896	25·6	8,905	7,113	4,326
1898....	509	9,618	5·1	69	2,885,736	11	272	2·8	76	15·8	17,013	20·8	7,591	1,263	724

Year.	Vitriol.	Zinc. Ore.		Zinc. Crude and M'frs.		Minerals, Not Specified.	
1894....	$44,876	24,822	$335,096	117	$7,341	6,823	$100,453
1895....	90,501	33,075	312,562	26	3,130	5,631	87,791
1896....	121,524	41,401	458,306	184	11,789	2,704	43,766
1897....	97,449	44,425	527,769	135	9,009	442	5,423
1898....	92,191	49,597	723,129	184	13,799	2,596	40,526

(*a*) From *Bidrag till Sveriges Officiela Statistik.* (*b*) Not stated in the reports. (*c*) Building stone, etc., wrought and unwrought.

UNITED KINGDOM.

THE statistics of the mineral production, imports and exports, are given in the subjoined tables. The statistics of the most important substances for 1899 will be found under the respective captions elsewhere in this volume.

MINERAL PRODUCTION OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1—$5.)

Year.	Alum Shale.		Arsenic.		Arsenical Pyrites.		Barytes.		Bauxite.		Chalk.	
1894.....	4,086	$2,480	4,878	$243,070	3,341	$19,115	22,700	$107,050	8,097	$28,090	(*c*)	.....
1895.....	2,096	1,290	4,875	260,990	2,998	13,925	21,509	115,295	10,574	12,530	2,971,023	$769,330
1896.....	*Nil.*	.....	3,674	237,415	8,949	40,035	24,117	127,050	7,365	9,590	3,616,177	785,850
1897.....	621	280	4,232	373,975	13,347	53,070	23,087	120,585	13,540	14,115	3,920 188	827,975
1898.....	13,895	8,510	4,241	268,935	11,272	40,720	22,581	116,265	12,600	14,490	4,366,782	903,255
1899.....	.....	.....	3,890	.....	13,644	.....	24,043	.....	8,137	.....	.....	.....

Year.	Clay. (*d*)		Coal.		Coal, Lignite.		Copper Ore.		Copper Precipitate.		Fluorspar.	
1894.....	3,315,988	$4,118,505	191,289,965	$313,650,895	339	$415	5,844	$69,545	245	$11,565	128	$345
1895.....	9,952,823	9,198,035	192,695,944	286,156,065	*Nil.*	.....	7,651	109,560	264	14,275	37	270
1896.....	11,523,250	7,210,345	198,487,049	285,950,735	*Nil.*	.....	9,113	107,930	201	10,620	400	2,390
1897.....	12,908,479	7,265,640	205,364,010	298,700,045	*Nil.*	.....	7,246	93,530	224	11,600	302	1,985
1898.....	14,974,290	8,081,790	205,287,388	320,846,910	*Nil.*	.....	9,145	129,245	132	6,500	57	245
1899.....	.....	.....	223,606,668	.....	.....	.....	27,911	.....	.....	.....	745	.....

Year.	Gold Ore.		Graphite.		Gravel and Sand.		Gypsum.		Iron Ore.		Iron Ore, Bog. (*b*)	
1894.....	6,708	$67,865	*Nil.*	.....	(*c*)	.....	155,905	$331,775	12,565,185	$15,953,285	7,928	$9,755
1895.....	13,478	82,920	41	$500	1,090,709	$405,535	180,738	359,175	12,817,261	14,328,545	5,742	7,065
1896.....	2,809	21,285	*Nil.*	.....	1,208,310	450,100	196,404	372,690	13,919,976	15,752,120	6,758	8,315
1897.....	4,589	31,410	*Nil.*	.....	1,378,496	556,660	184,287	334,890	14,008,484	16,068,975	7,238	8,905
1898.....	715	5,790	*Nil.*	.....	1,652,701	677,690	196,164	356,580	14,403,769	17,033,140	5,505	6,770
1899.....	3,096	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Jet—Kg.		Lead Ore.		Manganese Ore.		Ocher, Umber, etc.		Oil Shale.		Petroleum.		Phosphate, Lime.	
1894.....	217	$240	41,249	$1,334,975	1,838	$3,700	8,652	$70,200	2,018,167	$2,482,980	50	$460	711	$6,385
1895.....	76	80	39,026	1,366,960	1,293	3,405	7,747	944,945	2,282,815	2,808,580	15	140	2,540	21,875
1896.....	133	50	41,726	1,516,990	1,097	3,065	10,049	123,440	2,458,237	3,024,405	12	145	3,048	26,250
1897.....	31	40	35,903	1,377,045	609	1,755	14,653	64,985	2,259,325	2,779,680	12	145	2,032	17,500
1898.....	*Nil.*	.....	33,513	1,337,010	235	1,000	20,144	65,015	2,172,201	2,672,490	6	70	1,575	13,565
1899.....	.....	.....	31,446	.....	422	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Pyrites.		Salt.		Silica.				Slag.		Slates and Slabs.	
					Chert and Flint		Quartz.					
1894.....	15,771	$40,210	2,271,687	$3,818,145	(*c*)	.....	(*c*)	.....	(*c*)	.....	469,060	$5,856,830
1895.....	9,193	20,570	2,208,025	3,548,755	96,304	$83,305	736	$2,750	137,040	$29,440	591.068	6,370,730
1896.....	10,177	23,015	2,054,715	3,333,065	109,694	85,150	527	1,945	571,290	76,790	596,324	6,691,280
1897.....	10,752	22,625	1,933,949	3,104,490	95,209	82,475	459	1,695	(*c*)	.....	618,941	8,247,880
1898.....	12,302	24,020	1,908,723	3,100,575	83,370	72,565	(*c*)	.....	(*c*)	.....	679,461	9,501,140
1899.....	12,968	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Stone.								Strontium Sulphate.	
	Granite.		Limestone.		Sandstone.		Whinstone, Basalt, etc.			
1894.....	(*c*)	.....	(*c*)	.....	(*c*)	.....	(*c*)	.....	6,992	$9,810
1895.....	1,694,450	$2,739,995	9,677,440	$6,026,905	4,290,714	$6,832,980	1,756,004	$1,761,910	12,469	18,645
1896.....	1,784,925	2,490,370	11,187,532	6,078,020	4,579,869	7,089,925	2,323,591	2,127,935	18,331	25,940
1897.....	1,876,880	2,708 020	11,179,580	5,779,965	5,043,535	7,623,500	2,393,243	2,206,955	15,227	21,550
1898.....	1,905,830	2,882,286	12,172,207	6,280,770	5,325,988	8,163,930	2,644,131	2,327,715	13,148	18,370
1899.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Tin Ore (Black Tin).		Tungsten Ore.		Uranium Ore.		Zinc Ore.	
1894	13,117	$2,437,615	*Nil.*	.....	19	4,075	22,170	$336,555
1895	10,782	1,852,650	*Nil.*	.....	41	10,355	17,758	247,150
1896	7,786	1,299,640	44	$6,775	36	7,500	19,588	322,225
1897	7,234	1,271,090	127	10,040	30	6,895	18,586	345,770
1898	7,498	1,441,625	331	79,220	26	5,925	23,989	588,920
1899	5,752	.....	89	.....	7	.....	23,505	.....

METALS OBTAINABLE BY SMELTING FROM THE ORES IN THE ABOVE TABLE. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Copper, Fine.		Gold—Kg.		Iron.		Lead. (*g*)		Silver—Kg.		Tin.		Zinc.	
1894	454	$74,055	131·12	$74,055	4,417,082	$49,995,900	30,162	$1,423,120	7,575	$166,565	8,461	$3,022,500	8,260	$655,145
1895	589	136,315	205·28	92,600	4,465,307	52,671,625	29,465	1,543,670	8,722	174,540	6,754	2,233,900	6,780	508,475
1896	565	140,900	42·07	25,175	4,835,597	56,877,370	31,311	1,754,700	8,828	181,825	4,915	1,538,390	7,224	616,200
1897	526	135,480	63·20	35,925	4,942,679	56,973,895	26,988	1,662,890	7,750	148,070	4,524	1,456,630	7,162	634,115
1898	650	177,615	12·29	6,495	4,928,116	63,700,215	25,761	1,664,975	6,575	118,640	4,722	1,729,060	8,711	897,910

(*a*) From *Mineral Statistics of the United Kingdom.* (*b*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*d*) Including china clay, potters' clay, and fuller's earth; and for 1895 and 1896 comprises a large quantity of ordinary brick clay not included in the returns of previous years. (*e*) Not reported. (*f*) Not including chalk. (*g*) Smelted from native ores. (*h*) Includes copper precipitate.

Additional products: In 1897: Mica, 5,063 metric tons, $8,635; soapstone, 28 metric tons, $490; nickel ore, 305 metric tons, $1,500. Metals obtainable by smelting in 1897: Aluminum, 315 metric tons, $229,400; nickel, 7½ metric tons, $5,250; sodium, 86 tons, metric $63,750.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Alkali.		Asphaltum.		Brass and Bronze Manufactures.		Chemical Products.	Clay Products, Porcelain and Earthenware.		Coal, Culm and Cinders.		Copper. Manuf'res Unenumerated.
1894	7,907	$519,120	25,111	$404,965	1,133	$669,385	$6,877,445	10,246	$3,097,860	10,478	$155,460	$700,905
1895	9,353	572,930	27,399	364,880	1,637	854,905	6,383,350	12,062	3,432,875	16,525	215,440	910,500
1896	8,949	480,650	29,069	394,790	2,075	1,096,020	6,922,975	16,444	4,223,745	16,000	130,075	1,243,050
1897	11,557	545,010	44,541	519,080	2,129	1,045,985	6,781,900	17,847	4,523,940	9,605	87,555	1,892,425
1898	12,179	673,180	46,398	635,110	3,357	1,068,330	6,939,150	16,405	4,301,740	11,191	81,550	2,662,400
1899	12,078	669,390	.....	.....	.....	.....	7,478,170	.....	.....	.....	.....	.....

Year.	Copper—*Continued.* Ore.		Regulus and Precipitate.		Wrought, Unwrought & Old.		Diamonds—*Carats.*		Glass, All Kinds.	Gold Leaf. *Number.*	
1894	84,130	$2,468,125	80,107	$8,839,775	61,260	$12,325,000	2,485,352	$14,989,765	$13,425,310	84,614,380	$817,855
1895	101,353	2,873,015	92,727	11,104,775	46,493	9,598,485	3,607,750	23,770,425	12,375,880	66,138,410	641,105
1896	88,732	2,833,955	92,252	11,477,545	66,405	15,013,905	3,527,810	22,995,890	12,537,895	70,483,026	694,365
1897	83,916	2,383,130	90,008	11,378,900	62,055	15,200,790	3,472,916	22,176,300	15,084,555	68,173,400	666,595
1898	91,141	3,291,150	76,201	9,779,790	70,018	17,968,325	3,476,509	22,619,075	16,423,915	63,632,700	624,960
1899	130,611	5,460,735	84,015	12,562,945	60,502	21,572,275	.....	.....	16,044,400	.....	.....

Year.	Gold Ore.		Graphite.		Iron. Bar, Angle, Bolt and Rod.		Iron and Steel Mfres	Iron and Steel, Old and Scrap.		Ore.	
1894	488	$421,090	14,361	$1,059,230	64,259	$2,777,790	$14,230,955	17,108	$261,450	4,484,270	$14,892,985
1895	865	1,001,345	10,526	887,100	68,833	2,747,670	15,494,510	15,966	246,230	4,521,516	14,980,760
1896	2,263	604,015	13,861	1,121,195	72,195	2,852,060	22,055,335	15,578	295,555	5,525,320	18,893,945
1897	1,074	736,415	13,472	1,104,490	69,222	2,697,545	27,894,295	20,785	344,610	6,064,179	22,179,670
1898	1,019	1,393,175	15,113	1,506,805	70,331	2,782,745	33,979,160	24,619	349,230	5,555,899	20,173,210
1899	.....	.....	.....	.....	74,326	2,880,790	39,527,075	.....	.....	7,168,061	26,875,190

Year.	Iron—*Continued.* Pig and Puddled.		Steel, Unwrought.		Lead. Ore.		Pig and Sheet.		Manufactures.		Manganese Ore.	
1894	62,967	$1,498,695	8,724	$384,050	15,850	$366,625	164,451	$7,578,925	132	$14,495	130,029	$1,692,000
1895	94,609	2,156,360	11,036	475,015	32,234	797,400	165,581	8,270,365	416	60,180	133,623	1,484,740
1896	108,152	2,206,220	17,771	788,780	57,172	1,448,245	170,484	9,298,715	218	51,895	162,542	1,844,740
1897	160,581	3,588,490	40,628	1,330,655	32,818	1,001,140	170,121	10,166,500	298	61,150	158,825	1,735,775
1898	162,075	2,688,445	40,875	1,263,025	44,457	1,512,535	197,591	12,606,780	419	66,880	156,390	1,700,555
1899	174,159	3,104,255	78,257	2,170,100	.....	.....	201,551	14,419,185	.....	.....	.....	.....

Year.	Mica, Sheet.		Mica and Talc.		Paraffin.		Petroleum. *Liters.*		Phosphate Rock.		Painters' Colors and Pigments.
1894....	408	$456,900	.....	.....	31,397	$3,166,045	740,519,277	$12,424,890	386,351	$3,616,025	$4,544,005
1895....	373	439,860	918	$241,355	36,612	3,778,950	804,777,131	16,844,520	365,414	3,166,570	4,369,555
1896....	431	528,050	1,753	520,425	36,850	3,555,250	862,960,772	18,660,280	295,975	2,329,655	4,950,635
1897....	412	554,480	1,688	570,595	39,284	3,875,055	842,920,307	16,676,355	330,335	2,466,060	5,049,225
1898....	517	711,855	1,398	419,095	48,104	4,045,160	829,995,751	18,668,150	334,884	2,499,360	5,619,895
1899....	.....	.....	.....	.....	.....	.....	908,107,248	22,883,945	.....	.....	.....

Year.	Platinum, Wrought and Unwrought. Kg.		Potassium Nitrate.		Pyrites, Iron and Copper.		Quicksilver.		Silver Ore.	Slate.	Sodium Nitrate.	
1894....	2,701	$560,870	14,700	$1,327,820	625,907	$5,246,555	1,743	$1,516,810	$12,199,775	.....	127,305	$5,834,000
1895....	1,883	395,955	11,607	1,068,265	591,782	4,935,825	1,689	1,672,790	8,782,220	$358,445	124,650	4,994,485
1896....	1,748	368,725	17,310	1,392,420	508,480	4,998,905	1,604	1,638,645	6,483,680	940,890	108,148	4,182,760
1897....	2,257	437,385	16,744	1,233,496	633,009	5,104,480	1,862	1,954,405	7,149,210	1,538,660	107,525	4,059,325
1898....	3,389	734,205	13,325	1,016,215	605,544	5,404,415	1,856	1,940,890	5,739,525	1,646,005	132,412	4,864,005
1899....	.....	.....	12,635	1,022,425	712,363	5,821,335	1,759	2,077,150	5,162,750	.....	.....	.....

Year.	Stone, Marble. Hewn or Manufactured.		Sulphur.		Tin in Blocks, Ingots, Bars, or Slabs.		Tin Ore.		Zinc, Crude, in Cakes.	
1894....	594,844	$3,622,280	23,771	$513,140	39,774	$13,155,190	4,508	$742,760	53,743	$4,099,305
1895....	584,082	3,404,885	25,318	496,835	42,267	13,155,190	4,780	746,105	63,525	4,669,470
1896....	689,977	4,141,135	22,990	470,060	38,989	11,448,440	4,950	725,915	77,861	6,225,065
1897....	752,345	4,592,345	22,811	511,045	27,214	8,118,990	5,028	841,890	70,929	5,946,945
1898....	883,699	5,179,950	19,642	450,155	20,665	6,948,005	5,477	407,685	78,761	7,211,345
1899....	.....	.....	21,906	508,075	27,608	16,290,910	.....	.....	71,068	8,316,905

Year.	Zinc Manufactures.		Zinc Ore.		Ores, Unclassified.		Metals, Unclassified, Wrought, Unwrought, and Old.	
1894.....	16,843	$1,875,075	46,341	$892,030	45,391	$1,822,915	9,016	$2,512,145
1895.....	19,670	1,875,960	38,096	739,450	46,104	1,570,045	8,417	2,377,890
1896.....	21,339	2,097,185	21,421	498,170	45,782	1,289,665	10,695	2,887,750
1897.....	21,395	2,194,365	25,238	583,345	72,675	1,959,095	16,000	3,678,235
1898.....	21,618	2,290,440	49,238	1,100,705	51,660	1,476,625	14,459	3,545,690
1899.....	21,521	2,969,650	.....	.....	.....	.....	.....	.....

(*a*) *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported.

MINERAL EXPORTS OF THE UNITED KINGDOM. (*a*) (IN METRIC TONS AND DOLLARS. £1 —$5.)

EXPORTS OF DOMESTIC PRODUCE.

Year.	Alkali.		Brass and Manufactures.		Cement.		Chemical Products.	Clay Products, Porcelain and Earth'nw're	Coal and Culm.	
1895....	317,430	$7,787,665	5,445	$2,093,005	401,812	$3,207,755	$10,829,365	$10,851,610	32,222,344	$73,001,630
1896....	245,988	6,210,865	6,070	2,463,115	359,429	2,902,085	11,532,200	10,847,690	33,474,843	71,898,560
1897....	252,786	6,398,450	5,713	2,458,370	398,023	3,239,595	11,969,165	8,629,425	35,919,965	77,922,710
1898....	191,578	5,033,025	5,418	2,346,205	331,648	3,052,615	12,577,815	8,253,640	35,619,365	85,779,115
1899....	193,492	5,158,115	5,797	2,815,835	359,273	3,466,755	13,750,635	9,314,255	(*m*)	.....

Year.	Coke and Cinders.		(*b*) Coal, etc., for Steam'rs	Coal Products (*c*)	Copper Sulphate.		Copper.			
							Unwrought, In Ingots, etc.		Mixed or Yellow Metal.	
1895....	711,265	$2,324,215	9,558,314	$7,868,655	.....	.....	30,019	$6,724,565	15,150	$3,149,530
1896....	687,640	2,191,980	10,096,302	9,168,630	53,464	$4,903,315	23,698	5,697,555	11,428	2,576,510
1897....	993,980	3,372,525	10,623,050	8,340,420	60,326	5,113,900	21,282	5,381,155	11,192	2,640,740
1898....	782,053	2,755,580	11,444,431	7,624,740	35,150	4,224,085	27,102	7,154,225	10,452	2,542,915
1899....	(*m*)	.....	12,422,429	7,712,965	40,822	4,261,185	32,449	11,994,300	7,088	2,186,940

Year.	Copper—*Con.*		Glass, All Kinds	Iron.						
	Wrought or Manufactured. N. E. S.			Pig and Puddled.		Bar (Except Railroad), Angle, Bolt and Rod.		Railroad, of All Sorts.		Iron and Steel Wire and Manufactures. (Except Telegraph Wire.)	
1895....	15,931	$4,220,155	$3,948,540	880,433	$10,385,365	146,204	$4,270,085	464,873	$9,485,180	42,896	$3,555,940
1896....	15,598	4,445,120	4,462,445	1,077,128	12,669,415	180,973	5,471,445	759,625	17,802,050	57,008	4,519,975
1897....	15,275	4,629,430	4,359,535	1,219,958	14,461,865	170,285	5,421,865	795,983	19,293,670	52,471	4,332,965
1898....	13,765	4,296,590	4,410,205	1,058,973	13,684,020	152,911	4,978,125	619,976	15,081,290	44,954	3,964,220
1899....	11,231	4,562,590	4,578,415	1,401,365	23,921,785	161,679	6,136,680	601,266	15,652,560	50,041	4,441,095

Year.	Iron—*Continued.* Hoops, Sheets (Ungalvanized) and Boiler and Armor Plates.		Galvanized Sheets.		Black Plates for Tinning. (Iron and Steel.)		Tinned Plates.		Cast and Wrought, and Manufactures. N. E. S.	
1895....	104,574	$3,815,570	207,472	$11,256,870	34,918	$1,691,730	371,978	$21,195,965	293,486	$18,638,035
1896....	122,660	4,226,140	248,348	14,218,110	49,179	2,389,995	271,234	15,180,075	372,090	23,598,635
1897...	120,868	4,483,370	231,319	12,810,420	59,663	2,844,645	276,260	15,186,395	382,179	24,426,375
1898....	102,638	3,916,335	230,219	12,653,350	59,289	2,706,435	255,797	13,768,540	361,927	23,630,065
1899....	111,773	4,661,900	242,167	15,607,050	86,936	4,139,790	260,735	15,838,415	363,855	26,160,420

Year.	Iron—*Concluded.* Old.		Steel, Unwrought.		Manufactures of Steel and Iron.		Total of Iron and Steel.	
1895.....	98,654	$1,262,700	211,616	$9,743,975	23,718	$3,103,200	2,880,910	$98,404,615
1896.....	129,463	1,694,535	302,198	12,587,775	37,290	4,650,350	3,607,204	119,008,500
1897.....	99,259	1,195,925	304,249	13,093,895	47,626	5,706,825	3,750,122	123,198,215
1898.....	86,602	1,201,050	290,182	13,203,560	35,857	4,513,470	3 299,326	113,200,460
1899.....	118,262	1,952,620	333,837	16,842,310	45,181	5,109,295	3,777,098	140,463,900

Year.	Lead. Pig.		Lead. Sheet, Pipe and Other Manufactures.		Potassium Nitrate.		Mica and Talc.		Salt, Rock and Brine.	
1895.....	22,920	$1,175,390	19,452	$1,290,590	38,543	$220,815	46	$42,375	752,397	$2,731,095
1896.....	18,976	1,067,590	22,898	1,549,245	29,994	165,590	30	33,125	671,510	2,351,650
1897.....	20,749	1,254,985	20,162	1,437,585	23,372	117,995	39	55,100	680,477	2,332,620
1898.....	18,964	1,231,360	19,720	1,499,410	26,859	134,610	89	69,065	608,882	2,300,075
1899.....	(*m*)	.....	(*m*)	.....	(*m*)	.....	(*m*)	.....	638,213	2,237,090

Year.	Slate.	Tin, Unwrought.		Zinc. Ore.		Zinc. Crude.		Zinc. Manufactures.		Metals, Unclassified Wrought & Unwrought.
1895.....	$880,115	5,761	$1,918,020	.....	.....	9,102	$547,460	935	$104,465	$2,291,190
1896.....	954,920	6,299	1,987,390	168	$42,575	8,605	545,195	1,034	126,456	2,688,340
1897.....	1,043,110	5,050	1,616,000	308	78,660	6,951	484,055	1,047	134,440	2,968,185
1898.....	938,905	5,557	1,974,430	324	93,365	7,577	618,875	1,227	168,695	3,164,995
1899.....	(*m*)	4,785	2,937,655	(*m*)	.....	(*m*)	.....	(*m*)	.....	(*m*)

EXPORTS OF FOREIGN AND COLONIAL PRODUCE. (*a*) (IN METRIC TONS AND DOLLARS; £1 — $5)

Year.	Chemical Products.	Copper, Unwrought, Partly Wrought, and Old.		Glass. Bottles, *Gross.*		Glass. All Other Kinds		Glass. Total.		Iron, Bar, Angle, Bolt, and Rod.		Steel, Unwrought.	
1895....	$1,050,315	8,093	$1,788,920	10,813	$24,935	4,334	$380,280	$405,215		23,184	$874,945	5,291	$234,090
1896	1,093,585	9,809	2,338,295	12,335	24,895	3,324	292,575	317,470		16,882	638,705	4,000	167,345
1897....	1,295,400	10,207	2,526,150	11,666	26,670	4,320	376,305	402,975		21,842	962,905	4,244	194,980
1898....	1,509,195	13,454	3,578,710	12,694	29,875	3,961	353,905	383,780		26,891	1,135,450	2,619	154,480
1899....	1,085,185	24,515	9,160,530	13,768	29,915	3,172	257,180	287,095		19,058	879,680	3,289	149,165

Year.	Iron and Steel, N. E. S., Wrought and Manufactured. (*h*)		Petroleum. *Liters.*		Potassium Nitrate.		Quicksilver.		Tin, in Blocks, Ingots, Bars, or Slabs.	
1895.....	34,790	$2,499,670	8,001,341	$268,690	1,459	$130,020	1,214	$1,241,465	21,042	$6,593,535
1896.....	37,292	3,712,450	11,709,283	354,420	1,913	152,765	1,198	1,150,020	19,007	5,669,945
1897.....	29,305	4,324,980	6,753,750	270,735	2,949	218,610	1,047	1,050,330	14,994	4,545,610
1898.....	29,509	5,658,860	6,479,451	217,200	2,775	208,030	1,157	1,199,500	15,885	5,310,350
1899.....	28,874	5,094,390	6,699,181	279,505	1,578	122,240	1,097	1,316,130	17,120	9,953,050

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Number of tons shipped for the use of steamers engaged in the foreign trade. This not being an export in the ordinary acceptation of the term, the value thereof is not given in the trade returns. (*c*) Including naphtha, paraffine, paraffine oil and petroleum. (*h*) N. E. S. signifies "not elsewhere specified." (*m*) Statistics not available at time of publication.

UNITED STATES.

STATISTICS and full particulars of the mineral production of the United States will be found in the introduction and the articles on the different substances. We give below the mineral imports and exports for five years:

MINERAL IMPORTS OF THE UNITED STATES. (*a*)

Year.	Aluminum.							Ammonium Sulphate.			
	Crude.				Leaf.		Mfd.				
	Lb.	Kg.	Value.	Value per Kg.	Pkgs.	Value.	Value.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895.....	25,294	11,475	$7,814	$0·68	6,610	$646	$1,841	24,253,291	11,001	$498,011	$44·82
1896.....	698	316	591	1·87	4,657	523	2,305	21,254,639	9,641	384,693	39·90
1897.....	1,822	854	1,082	1·26	4,260	370	3,279	19,351,119	8,778	317,121	36·13
1898.....	60	27	30	1·11	2,000	174	4,675	11,057,762	5,016	210,073	41·88
1899.....	53,622	24,323	9,425	0·39	693	112	5,303	17,121,988	7,766	406,573	52·35

Year.	Antimony.				Antimony Ore.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1895.....	3,399,903	1,587	$223,968	$141·12	668,610	303	$14,718	$48·57
1896.....	2,402,562	1,090	148,700	136·42	1,180,828	535	21,392	39·98
1897.....	1,146,696	520	46,955	90·30	5,502,132	2,496	167,373	67·06
1898.....	2,025,183	919	143,909	156·59	3,725,222	1,690	50,256	29·74
1899.....	3,100,697	1,434	240,988	168·09	3,982,133	1,806	47,841	26·49

Year.	Asbestos.			Asphaltum.			
	Crude. Value.	Manufactured. Value.	Total. Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895.....	$225,147	$19,731	$244,878	79,194	80,461	$212,407	$2·63
1896.....	229,884	15,654	244,738	95,233	96,757	286,099	2·95
1897.....	264,220	10,570	274,790	127,325	129,362	427,390	3·30
1898.....	287,636	12,899	300,535	67,711	68,794	202,452	2·94
1899.....	303,119	8,949	312,068	100,168	101,771	319,089	3·14

Year.	Barium Sulphate.								Bauxite.			
	Manufactured.				Unmanufactured.							
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895....	1,629	1,655	$17,112	$10·34	2,551	2,592	$7,561	$2·92	12,985,525	5,890	$29,194	$4·95
1896....	2,467	2,506	23,345	9·32	509	517	1,274	2·46	4,765,974	2,161	10,477	4·84
1897....	1,300	1,321	13,822	10·46	502	510	579	1·14	5,924,639	2,685	10,515	3·90
1898....	687	698	8,678	12·43	1,022	1,038	2,678	2·58	2,690,240	1,220	4,238	3·47
1899....	2,111	2,145	22,919	10·69	1,739	1,767	5,488	3·11	14,931,840	6,773	23,708	3·51

Year.	Brass and Manufactures of. Value.	Chloride of Lime, or Bleach. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Cement. Barrels. (b)	Metric Tons.	Value.	Value per Met. Ton.
1895.....	$158,322	103,317,968	46,855	$1,628,877	$34·76	2,975,082	543,844	$3,873,123	$7·12
1896.....	157,798	99,491,385	45,129	1,441,600	31·94	2,668,624	484,192	3,394,426	7·01
1897.....	85,341	109,176,451	49,522	1,460,799	29·50	2,090,894	379,357	2,688,122	7·09
1898.....	24,611	108,462,828	49,198	1,229,978	25·00	2,013,819	365,385	2,624,228	7·18
1899.....	58,916	123,583,061	56,056	1,203,344	21·47	2,108,213	382,512	2,858,256	7·47

Year	Chrome Ore. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Chromic Acid. Lb.	Kg.	Value.	Value per Kg.	Clays or Earths, Including Kaolin. Long Tons.	Metric Tons	Value.	Value per Met. Ton.
1895..	5,220	5,314	$82,845	$15·59	2,083	945	$414	$0·43	98,862	100,450	$716,590	$7·13
1896..	8,669	8,808	187,400	21·27	2,429	1,092	387	0·39	98,243	99,815	707,826	7·09
1897..	11,566	11,751	186,313	15·85	71,220	32,305	5,477	0·17	102,391	104,029	696,935	6·70
1898..	16,304	16,565	272,234	16·43	5,329	2,417	1,758	0·73	113,280	115,092	779,491	6·77
1899..	15,793	16,046	284,825	17·75	33,134	15,029	6,360	0·42	122,035	123,988	807,792	6·52

Year.	Coal. Anthracite. Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.	Bituminous. Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.	Total. Metric Tons.	Value.	Value Per Met. Ton.
1895	141,337	143,598	$328,706	$2·29	1,229,109	1,248,775	$3,642,055	$2·92	1,392,373	$3,970,761	$2·85
1896	101,689	103,316	237,717	2·30	1,246,991	1,266,943	3,488,115	2·75	1,370,259	3,725,832	2·72
1897	3,281	3,333	8,720	2·62	1,276,963	1,297,394	3,423,434	2·64	1,300,728	3,432,154	2·64
1898	3,149	3,199	8,609	2·69	1,270,557	1,290,886	3,569,572	2·77	1,294,085	3,578,181	2·77
1899	61	62	245	3·95	1,393,579	1,415,876	3,858,299	2·73	1,415,938	3,858,544	2·73

Year.	Coke. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Cobalt Oxide. Lb.	Kg.	Value.	Value per Kg.	Copper, Ore and Regulus. Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895	26,448	26,871	$71,366	$2·66	36,155	16,400	$39,889	$2·42	21,580,160	9,789	$404,439	$41·32
1896	43,372	44,066	114,713	2·60	27,189	12,333	36,212	2·93	6,117,440	2,775	348,629	125·63
1897	35,198	35,756	98,558	2·76	24,771	11,236	34,773	3·09	11,587,520	5,256	816,590	155·36
1898	41,185	41,844	142,334	3·40	33,731	15,300	49,245	3·22	6,861,120	3,112	654,355	210·26
1899	27,855	28,301	142,504	5·04	46,791	21,224	68,847	3·24	70,866,880	32,145	2,243,583	65·56

Year.	Copper, Ingots, Old, etc. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Copper, Manufactures. Value.	Copper. Total Value.	Cryolite. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895....	9,381,800	4,255	$839,603	$197·32	$60,344	$1,304,386	9,500	9,652	$127,165	$13·17
1896....	11,397,273	5,170	961,920	186·05	85,123	1,395,672	8,009	8,057	40,056	13·10
1897....	16,578,420	7,520	1,454,016	193·35	58,897	2,329,503	10,115	10,277	135,114	13·24
1898....	54,166,467	24,570	4,120,680	167·71	39,467	4,814,502	6,201	6,300	88,501	14·05
1899....	71,465,406	32,416	10,067,630	310·57	42,000	12,353,213	5,879	5,973	78,676	13·17

Year.	Earthen, Stone, and China Ware Value.	Emery Grains. Lb.	Metric Tons.	Value.	Value per Met. Ton	Emery Rock. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Emery Manufactures. Value.	Emery. Total Value.
1895....	$10,523,925	678,761	308	$25,066	$81·38	6,803	6,912	$80,386	$11·62	$27,596	$133,048
1896....	9,690,142	751,464	341	26,520	77·77	6,289	6,390	119,667	18·72	1,971	148,158
1897....	8,409,678	520,095	238	20,022	84·84	5,209	5,292	107,649	20·36	2,211	129,882
1898....	7,273,471	577,655	262	23,320	89·01	5,547	5,636	106,269	18·86	3,810	133,399
1899....	8,101,393	728,299	330	29,124	88·17	7,435	7,554	116,498	15·42	11,514	157,131

Year.	Fertilizers.								All Other Value.	Gold and Silver in Coin and Bullion.	
	Guano.				Phosphates, Crude.						
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.		Gold.	Silver.
1895....	4,066	4,131	$48,917	$11·84	18,140	18,430	$122,698	$6·67	806,872	$32,538,736	$11,296,007
1896....	6,888	6,947	88,231	12·70	14,908	15,147	116,415	7·69	897,227	102,761,282	12,504,577
1897....	4,931	5,019	55 709	11·12	8,019	8,147	65,137	8·00	1.205,804	29,081,760	12,146,750
1898....	5,259	5,343	56,933	10·66	66,129	67,187	308,635	4·52	1.045,113	153,146,159	9,572,310
1899....	2,700	2,743	26,787	9·76	115,113	116,955	518,458	4·43	966,127	40,111,932	9,542,628

Year.	Gold and Silver in Ores.		Iron Ore.				Pig Iron.			
	Gold.	Silver.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895.....	$1,857,656	$13,087,340	524,158	532,571	$786,207	$1·47	53,232	54,084	$1,337,978	$24·73
1896.....	1,969,977	17,775,1..	682,806	692,999	1,036,917	1·49	56,272	57,172	1,207,890	22·87
1897.....	4,941,052	20,935,552	489,970	497,809	678,912	1·39	19,212	19,519	484,655	24·83
1898.....	5,017,793	19,559,070	187,093	190,086	253,243	1·33	25,152	25,554	704,431	27·56
1899.....	11,083,082	21,302,358	674,098	684,894	1,083,045	1·58	40,372	41,018	1,339,206	32·65

Year.	Scrap Iron and Steel.				Bar Iron, Rolled and Hammered.				Bars, Railway, of Iron or Steel.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	$6,066	6,163	$103,523	$16·79	44,910,023	20,381	$772,829	$37·91	1,447	1,470	$27,076	$18·41
1896	8,250	8,382	130,012	15·51	36,458,336	16,538	686,733	41·52	7,796	7,921	207,648	26·21
1897	1,549	1,573	12,433	7·26	31.349,707	14,220	614,318	43·20	415	422	15,989	37·77
1898	1,788	1,812	33,330	18·39	42,627,276	19,426	844,363	43·47	200	208	5,181	25·52
1899	11,675	11,862	165,745	13·97	44,333,256	20,109	942,347	46·86	2,134	2,168	70,751	32·65

Year.	Hoop, Band, or Scroll.				Ingots, Blooms, Slabs, Billets, etc.				Sheet, Plate, and Taggers Iron or Steel			
	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895	33,168	15	$2,925	$195·00	58,811,651	26,677	$1,610,889	$60·38	32,549,127	14,764	$727,638	$49·28
1896	59,105	27	5,664	209·78	45,598,270	20,683	1,657,805	80·15	15,201,124	6,896	328,421	47·62
1897	61,916	28	2,873	102·61	38,600,934	17,509	1,523,410	87·00	5,854,990	2,656	170,385	64·15
1898	7,527	3	224	74·66	23,868,688	10,827	1,008,360	93·13	5,085.287	2,307	181,021	78·47
1899	1.485,720	674	33,492	49·69	28,225,647	12,803	1,287,725	100·58	15,777,163	7,156	464,297	64·88

Year.	Ties for Baling Cotton.				Tin Plates, Terne Plates, and Taggers Tin.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1895.....	7,806,304	3,541	$102,309	$28·89	491,880,513	223,100	$11,482,380	$51·46
1896.....	16,081,468	7,272	235,830	32·43	266.943,267	120,803	6,140,161	50·82
1897.....	235,154	107	3,686	34·45	187,825,880	85,198	4.366,828	51·25
1898.....	*Nil.*	.....	.....	.....	149,576,525	67.847	3,311,658	48·81
1899.....	*Nil.*	.....	.....	.....	131,970,441	59,861	3,788,567	62·45

Year.	Wire Rods.				Wire, and Articles made from.				Manufactures. Value.
	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	
1895.....	60,107,521	27,264	$1,090,707	$40·00	12,981,695	5,866	$716.901	$122·21	$7,796,988
1896.....	42,509,008	19,281	811,224	42·07	8,230,248	3.733	492.961	132·05	7,602,238
1897.....	36,768,536	16,679	772,950	46·34	5,720,322	2,596	344.855	132·84	5,523,678
1898.....	35,807,368	16,242	767,909	47·28	4,516,761	2,049	318,553	155·46	5,299,542
1899.....	40.239,108	18,252	873,398	47·85	5,292,909	2,401	400,952	166·99	6,482,726

Year.	Total Value of Iron Imports. (e)	Lead, Pig, Bars, Scrap, and in Ore.				Lead, Sheet, Pipe, Shot, etc.	
		Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.
1895.....	$25,772,136	201,617 855	91,453	$3,910,547	$31·32	128,008	58
1896.....	19,506,587	160,318,517	72,720	1,780,265	24·48	216,820	98
1897.....	13,835,950	185,318,412	84,060	2,480,471	29·51	95,891	43
1898.....	12,474,572	179,291,290	81,326	2,535,362	31·18	242,759	110
1899.....	15,799,206	189,632,749	86,016	2,947,866	34·27	110,372	59

Year.	Lead, Sheet, Pipe, Shot, etc.—*Continued.*		Lead, Other Manufactures. Value.	Lead, Total Value.	Manganese Ore.			
	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895.....	$5,080	$86·72	$7,981	$2,928,538	86,111	87,489	$747,910	$8·54
1896.....	8,513	86·87	*Nil.*	1,794,365	31,489	31,993	250,468	7·82
1897.....	4,042	94·00	814	2,480,471	39,574	40,207	340,945	8·48
1898.....	9,389	85·35	*Nil.*	2,535,362	114,885	116,723	831,967	7·13
1899.....	4,402	74·61	8,581	2,947,866	188,349	191,363	1,584,528	8·28

Year.	Marble and Stone, and M'f'res of.			Metals, Metal Compositions, and Manufactures of, N.E.S.			Mica, Mineral Substances, and Nickel.		
	Marble and Manufactures of.	Stone and Manufactures of.	Total.	Bronze Manufactures.	All Other.	Total.	Mica.	Mineral Substances N. E. S.	Nickel.
	Value.	Value. (d)	Value.	Value.	Value.	Value.	Value.	Value.	Value.
1895	$865,243	$411,122	$1,276,365	$521,505	$4,180,892	$4,702,397	$174,886	$111,447	$56,890
1896	808,030	387,979	1,196,009	468,884	3,692,937	4,161,821	169,086	74,395	32,532
1897	860,659	263,381	1,124,040	526,741	3,543,827	4,070,568	159,616	73,373	53,013
1898	726,343	229,909	956,252	508,814	3,479,221	3,988,035	150,082	19,160	(f)
1899	726,070	200,192	926,262	710,086	4,008,883	4,718,969	275,984	78,453	(f)

Year.	Oil, Mineral.				Paints, Mineral. Zinc Oxide in Oil.				Paints and Colors.	Platinum, Man'f'ur'd.
	Gallons.	Liters.	Value.	Value per Liter.	Lb.	Metric Tons.	Value.	Value pr. Met. Ton.	Value.	Value.
1895.....	1,917,823	7,259,804	$54,963	$·007	129,343	59	$7,272	$123·25	$1,363,657	$171
1896.....	68,803	260,419	14,062	·054	311,023	141	14,793	104·90	1,189,781	52
1897.....	949,178	3,592,638	77,789	·021	502,357	228	19,608	86·00	1,319,894	121
1898.....	2,024,090	7,661,808	146,713	·019	27,050	12	2,279	189·92	1,142,670	271
1899.....	1,285,857	4,867,367	120,986	·025	41,699	19	4,154	219·79	1,412,874	876

Year.	Platinum, Unmanufactured.			Plat'm Vases, Retorts	Potash, Chlorate of.				Potash, Chromate & Bichromate.			
	Kg.	Value.	Val. per Kg.	Value.	Lb.	Metric Tons.	Value.	Val. per Metric Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.
1895	2,320·15	$690,584	$297·65	$27,354	4,917,636	2,231	$456,342	$204·55	2,045,910	928	$181,242	$195·30
1896	2,584·14	926,678	358·60	109,286	4,659,793	2,114	366,300	173·27	952,794	432	80,538	186·43
1897	2,584·14	960,299	371·61	43,800	6,508,359	2,950	461,620	156·38	1,329,473	603	108,497	179·93
1898	3,142·07	1,178,142	374·96	52,012	4,805,332	2,180	288,965	132·55	1,160,710	526	96,134	183·75
1899	3,027·81	1,482,157	489·59	54,877	1,534,657	696	102,189	146·82	1,130,965	513	73,510	143·29

Year.	Potash, Muriate of.				Potash, Nitrate of.				Potash, all other.	
	Lb.	Metric Tons.	Value.	Val. per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.
1895.....	81,833,581	37,119	$1,296,184	$34·92	11,419,090	5,180	$305,207	$58·91	24,862,765	11,278
1896.....	88,525,983	40,155	1,392,504	34·68	20,085,327	9,111	479,999	52·67	21,383,884	9,677
1897.....	108,839,049	49,369	1,683,472	34·10	16,276,352	7,383	306,696	41·54	30,311,863	13,749
1898.....	103,368,701	46,888	1,620,720	34·57	11,461,823	5,199	281,504	54·15	37,736,761	17,117
1899.....	117,449,708	53,275	1,819,448	34 15	18,961,270	8,601	377,371	43·88	45,314,597	20,555

Year.	Potash, all other. *Continued.*		Precious Stones.		Pyrites.				
			Rough & Uncut.	Cut.					
	Value.	Value per Metric Ton.	Value.	Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Average Sulphur Contents.
1895	$337,416	$29·92	$111,083	$6,623,669	190,436	193,483	$673,812	$3·48	.....
1896	289,628	29·98	78,815	4,854,358	199,678	202,873	1,140,571	5·62	.....
1897	667,209	48·53	1,425,614	5,090,125	259,546	263,699	874,419	3·32	25%
1898	827,556	48·35	2,550,076	7,388,086	171,879	174,629	544,165	3·12	.....
1899	942,352	45·85	4,955,587	11,653,917	310,615	315,585	1,084,451	3·44	.....

Year.	Quicksilver.	Salt.				Soda, Nitrate of.			
	Value.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value Per Met. Ton.
1895	$2,503	566,869,046	257,180	$760,811	$2·95	118,477	120,373	$3,778,360	$31·38
1896	2,037	527,635,090	239,334	696,197	2·91	115,504	117,352	3,566,744	30·39
1897	20,147	461,813,843	209,479	611,166	2·44	94,965	96,484	2,810,187	29·12
1898	51	374,810,225	169,786	587,348	3·46	147,494	149,854	2,298,240	15·34
1899	83	386,378,938	175,260	587,103	3·35	146,492	148,886	3,486,313	23·42

Year.	Soda, Bicarbonate.				Soda, Caustic.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1895	8,862,028	4,020	$123,425	$30·70	72,019,114	32,668	$1,211,090	$37·13
1896	4,343,786	1,970	60,878	30·90	47,827,247	21,694	859,809	39·63
1897	965,669	438	13,982	31·92	57,742,392	26,192	988,318	37·73
1898	280,988	127	5,794	45·62	24,981,873	11,332	354,270	31·26
1899	162,398	74	5,219	70·61	13,363,539	6,062	186,008	30·68

Year.	Soda Ash and Sal Soda.				All other Soda Salts.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Metric Ton.
1895	307,026,094	139,266	$2,321,612	$16·67	1,081,887	491	$31,581	$64·32
1896	191,780,537	86,991	1,444,417	16·60	1,279,285	580	40,039	69·03
1897	154,856,060	70,152	1,122,867	16·00	7,927,145	3,596	100,989	28·08
1898	73,064,707	33,142	447,119	13·49	23,354,802	10,594	256,962	24·26
1899	56,058,837	25,426	425,205	16·72	26,340,840	11,948	355,502	29·75

Year.	Sulphur, Crude.				Sulphur, Flowers.				Sulphur, Refined.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	115,959	127,974	$1,593,148	$12·45	581	590	$12,868	$21·84	229	233	$4,378	$18·79
1896	145,318	147,643	2,085,076	14·12	665	675	13,266	19·65	447	454	8,226	18·11
1897	138,846	141,067	2,442,420	17·31	319	324	7,950	24·54	148	150	3,387	22·58
1898	159,790	162,347	3,081,974	18·98	507	515	14,548	28·25	163	166	4,391	26·45
1899	143,234	145,526	2,484,791	17·07	336	341	9,917	29·08	184	187	4,519	24·17

Year.	Talc.				Tin.			
	Lb.	Metric Tons.	Value.	Value per Metric Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895	6,329,268	2,871	$25,843	$9·00	54,252,045	24,609	$7,405,619	$300·98
1896	3,901,314	1,770	18,693	10·56	44,639,324	20,248	5,848,933	288·86
1897	1,597,849	725	8,423	11·61	55,172,571	25,026	7,415,933	296·33
1898	889,645	404	5,526	13·68	62,748,399	28,462	8,770,221	308·49
1899	508,553	231	3,534	15·32	71,248,407	32,318	16,746,117	518·17

Year.	Zinc. Block, Pig, and Old. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Manufactures of. Net Value.	Total Value.
1895.....	864,113	392	$29,352	$74·87	$12,183	$41,535
1896.....	856,044	388	25,904	66·76	15,728	41,632
1897.....	2,557,341	1,160	95,883	82·66	19,431	115,314
1898.....	2,742,357	1,244	109,624	88·12	13,448	123,072
1899.....	2,985,463	1,354	151,956	112·28	14,804	166,760

(a) From *Summary of Commerce and Finance of the United States.* (b) Custom-house returns for these years are given in pounds, which are reduced to barrels of 400 lb. for convenience of comparison. (d) Including slate. (e) Not including ore. (f) Not reported.

MINERAL EXPORTS OF DOMESTIC PRODUCTION OF THE UNITED STATES. (a)

Year.	Aluminum and Manufactures of. Value.	Asbestos. Value.	Brass & Manufactures of. Value.	Cement. Bbls.	Metric Tons(m)	Value.	Value per Met. Ton.	Chemicals, Drugs and Medicines. Value.	Coal. Anthracite. Long Tons.	Metric Tons.
1895....	(n)	(n)	$730,424	(b) 83,632	15,174	$117,646	$7·75	$8,749,090	1,470,710	1,494,341
1896....	(n)	(n)	1,026,191	54,839	9,950	86,757	8·72	9,361,354	1,350,000	1,371,600
1897....	(n)	$73,736	1,346,802	53,466	9,701	93,684	9·66	9,633,881	1,298,768	1,319,548
1898....	$239,997	54,660	1,237,027	36,732	6,665	73,898	11·08	9,732,734	1,350,948	1,372,563
1899....	291,515	77,409	1,607,072	81,090	14,713	166,078	11·29	11,949,884	1,707,796	1,735,121

Year.	Coal—*Continued.* Anthracite.—*Cont.* Value.	Val. per Met. Ton.	Bituminous. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Total Long Tons.	Total Metric Tons.	Total Value.	Copper Ore. (c) Lb.	Metric Tons.
1895	$5,937,130	$3·96	2,211,983	2,247,375	$4,816,847	$2·14	3,682,693	3,741,616	$10,753,977	30,965,760	14,045
1896	5,925,506	4·32	2,276,202	2,312,621	5,072,818	2·19	3,626,202	3,684,221	10,998,324	41,426,560	18,790
1897	5,836,730	4·42	2,399,263	2,437,651	5,326,761	2·19	3,698,031	3,757,199	11,163,491	20,903,360	9,309
1898	5,712,985	4·16	3,152,457	3,202,896	6,699,248	2·09	4,503,405	4,575,459	12,412,233	20,928,320	9,492
1899	7,140,100	4·12	4,044,354	4,109,064	8,573,276	2·09	5,752,150	5,844,184	15,713,376	7,848,960	3,560

Year.	Copper Ore—*Con.* (c) Value.	Val. per Metric Ton.	Copper, Pig, Sheet, and Old. Lb.	Metric Tons.	Value.	Val. per Met. Ton.	Copper. Manufactures. Value.	Copper. Total Value.	Earthen. and China Ware. Value.
1895.....	$1,631,251	$116·13	d121,328,390	55,084	$12,222,769	$222·09	$1,084,289	$14,938,309	$139,297
1896.....	2,393,914	127·40	259,223,924	111,758	27,822,280	248·95	819,017	31,035,211	169,343
1897.....	1,199,029	130·20	277,255,742	125,763	30,627,945	243·54	958,379	32,785,353	207,603
1898.....	755,443	79·59	291,955,905	132,430	33,598,869	253·71	1,190,939	35,545,251	251,821
1899.....	382,989	107·58	247,370,681	112,207	41,250,166	367·62	1,952,499	43,485,654	511,752

Year.	Fertilizers. Phosphates, Crude. Long Tons.	Metric Tons.	Value.	Value per Met. Tons	All Other. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Glassware. Value.	Gold and Silver in Coin and Bullion. (e) Gold.	Silver.
1895....	596,600	606,146	$5,182,965	$8·55	(o)	.....	.....	.....	$1,002,328	$104,605,023	$53,853,153
1896....	496,168	504,107	4,537,544	9·00	(o)	.....	.....	.....	1,115,252	58,047,969	63,063,396
1897....	551,046	559,863	5,136,546	9·17	(o)	.....	.....	.....	1,234,558	34,174,182	58,352,274
1898....	570,948	580,083	4,672,462	8·05	16,714	16,981	$442,977	$26·09	1,289,919	16,113,258	53,672,605
1899....	867,790	881,675	6,770,102	7·68	49,082	49,867	1,031,882	20·69	1,716,843	45,317,461	53,422,242

Year.	Gold and Silver in Ores. (f) Gold.	Silver.	Iron Ore. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Iron, Pig. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895.....	$362,379	$377,983	1,549	1,573	$7,654	$4·86	26,164	26,584	$371,297	$13·96
1896.....	209,621	993,405	11,016	11,192	38,916	3·48	62,071	63,064	943,022	14·95
1897.....	102,219	309,018	7,583	7,704	24,612	3·19	232,686	236,399	3,969,010	13·25
1898.....	81,606	123,499	31,579	32,084	67,548	2·11	252,057	257,106	2,702,551	10·51
1899.....	61,950	39,495	40,690	41,341	76,687	1·85	228,653	232,311	3,282,241	14·13

Year.	Iron, Bar.				Iron, Band, Hoop and Scroll.				Billets, Ingots, and Blooms.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	3,329	3,383	$147,798	$43·68	198	201	$8,169	$40·64	(*n*)	.....	.....	.....
1896	3,505	3,561	168,560	47·37	268	272	12,303	45·23	(*n*)	.....	.....	.....
1897	4,493	4,566	150,897	33·05	1,424	1,446	44,754	30·95	6,356	6,458	$108,333	$16·78
1898	7,074	7,188	241,499	33·60	1,593	1,619	58,731	36·28	28,600	29,058	544,771	18·75
1899	10,603	10,773	407,161	37·79	2,869	2,915	117,002	40·14	25,605	26,015	546,487	21·01

Year.	Iron, Nails and Spikes, Cut.				Iron Nails & Spikes; Wire, Wrought, Horseshoe and all other, inc. Tacks.				Plates and Sheets of Iron.			
	Lb.	Metric Tons.	Value.	Val. per Metric Ton.	Lb.	Metric Tons.	Value.	Val. per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Metric Ton.
1895	17,689,414	8,001	$339,825	$42·47	5,301,208	2,304	$239,610	$104·00	946,470	429	$32,170	74·98
1896	23,708,847	10,754	458,758	43·59	9,563,812	4,338	362,291	83·51	1,725,779	784	48,905	62·37
1897	33,773,216	15,319	670,709	43·78	20,070,024	9,104	550,654	60·49	9,061,447	4,110	175,799	42·77
1898	35,247,266	15,988	641,779	40·20	35,409,789	16,062	839,299	52·25	10,003,903	4,538	204,170	44·99
1899	22,342,543	10,135	482,882	47·64	79,767,846	36,182	1,957,773	54·11	13,880,092	6,296	356,791	56·67

Year.	Plates and Sheets of Steel.				Rails or Bars of Iron.				Rails or Bars of Steel.			
	Lb.	Metric Tons.	Value.	Val. per Metric Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	1,818,319	825	$60,889	$73·80	6,792	6,900	$189,470	$20·21	8,807	8,948	$222,661	$24·88
1896	4,144,127	1,880	96,568	51·37	628	638	18,581	28·18	72,508	73,663	1,712,716	23·30
1897	11,364,821	5,155	173,567	33·66	5,413	5,499	95,520	17·37	142,808	145,098	2,949,901	20·33
1898	60,647,662	27,510	787,245	28·62	8,311	8,444	101,109	11·97	293,592	298,290	5,838,464	19·57
1899	118,425,114	51,449	1,690,570	32·86	6,442	6,545	96,185	14·69	171,272	174,012	6,122,382	35·18

Year.	Structural Iron and Steel.				Wire.				Wire Rods. (Steel.)			
	Long Tons.	Metric Tons.	Value.	Val. per Met. Ton	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Met. Ton
1895	(*n*)	.....	.....	.....	66,024,358	29,948	$1,375,195	$45·90	(*n*)	.....	.....	.....
1896	(*n*)	.....	.....	.....	85,216,157	38,655	1,788,095	46·23	(*n*)	.....	.....	.....
1897	15,072	15,313	$604,339	$39·47	118,887,872	53,927	2,353,829	43·65	23,484,969	10,653	$240,787	$22·60
1898	34 088	34,583	1,255,451	36·30	167,252,882	75,865	3,036,818	40·03	41,462,598	18,807	390,144	20·74
1899	54,244	55,112	2,059,289	37·37	200,589,627	90,964	5,526,930	60·76	38,062,488	17,265	594,466	30·38

Year.	Lead & Manufactures.	Marble, Stone, & Manufactures of.	Mica.	Nickel. (*g*)	Petroleum (1 — 1,000 in Quantities and total Values).							
					Crude.				Naphtha.			
	Value.	Value. (*k*)	Value.	Value.	Gallons.	Liters.	Value.	Value per Liter.	Gallons.	Liters.	Value.	Value per Liter.
1895	$214,856	$959,871	*Nil.*	$239,897	116,108	439,469	$6,286	$0·014	12,922	48,910	$1,000	$0·024
1896	601,373	1,156,051	*Nil.*	606,833	118,133	447,133	6,032	·013	13,641	51,631	1,123	·021
1897	433,319	1,682,044	*Nil.*	997,391	121,864	461,255	5,044	·011	13,704	51,869	1,020	·020
1898	215,239	1,842,220	$6,272	1,359,609	120,436	455,888	5,016	·011	17,258	65,326	1,071	·016
1899	273,919	1,900,763	4,655	1,151,923	117,690	445,495	5,958	·013	18,210	68,930	1,597	·023

Year.	Petroleum—*Continued.*											
	Illuminating.				Lubricating.				Residue, etc.			
	Gallons	Liters.	Value.	Value per Liter.	Gallons.	Liters.	Value.	Value per Liter.	Gallons (*j*)	Liters.	Value.	Value per Liter.
1895	686,006	2,596,583	$43,540	$0·017	47,877	180,214	$6,289	$0·035	170	643	$14	$0·022
1896	758,076	3,869,317	49,704	·013	51,705	195,719	6,771	·035	521	1,972	28	·014
1897	804,446	3,044,828	46,876	·015	52,659	199,332	6,732	·034	12,247	46,355	395	·007
1898	764,823	2,895,093	38,895	·013	65,526	248,038	7,626	·031	30,436	115,209	815	·007
1899	733,393	2,776,120	49,174	·018	71,105	269,153	8,656	·032	21,609	81,798	658	·008

Year.	Petroleum—*Continued.* Paraffine. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Quicksilver. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Tin Manufactures. Value.
1895	114,234	51·9	$4,505	$86·79	1,188,955	504	$482,085	$956·91	$252,223
1896	112,517	51·0	4,563	89·41	1,525,726	692	618,437	895·02	268,561
1897	136,069	61·7	5,284	85·61	1,007,770	457	394,549	863·35	284,020
1898	166,317	75·4	6,363	84·34	981,497	445	440,587	990·08	281,794
1899	181,861	82·5	7,650	92·74	1,268,572	573	609,586	1,063·84	401,217

Year.	Zinc Ore and Oxide. Lb.	Metric Tons.	Value.	Value per Met. Ton.	Zinc; Pigs, Bars, Plates and Sheets. Lb.	Metric Tons.	Value.	Value per Met. Ton.	All Other Manufactures. Value.	Total Value.
1895	53,760	24	$1,008	$42·00	3,060,805	1,388	$153,175	$110·35	$50,051	$204,234
1896	4,648,000	2,108	47,408	22·49	20,800,169	9,388	1,013,620	110·08	51,001	1,112,029
1897	18,502,400	8,393	211,350	25·18	28,490,662	12,923	1,356,538	104·97	71,021	1,638,909
1898	23,564,800	10,689	299,870	28·05	20,998,413	9,525	1,033,959	108·55	138,165	1,471,994
1899	56,441,280	25·600	725,944	28·36	13,509,316	6,128	742,521	121·17	143,232	1,611,697

MINERAL EXPORTS OF FOREIGN PRODUCE FROM THE UNITED STATES. (*a*)

Year.	Antimony. Lb.	Metric Tons.	Value.	Value Per Metric Ton.	Antimony Ore. Lb.	Metric Tons.	Value.	Value Per Metric Ton.
1895	*Nil.*	....	....	....	*Nil.*	....	....	....
1896	6,951	3·2	$475	$148·44	(*p*)	....	....	....
1897	4,089	1·9	287	151·55	37,895	17·0	$2,348	$138·12
1898	25,275	11·4	1,729	151·67	34,321	15·6	784	50·26
1899	16,815	7·6	1,275	167·76	*Nil.*	....	....	....

Year.	Asphaltum or Bitumen (Crude.) Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Brass and Manufactures of. Value.	Cement. Lb.	Metric Tons.	Value.	Value per Met. Ton.
1894	72	73	$1,906	$26·11	$8,711	3,578,280	1,623	$13,895	$8·56
1895	113	115	2,615	22·74	3,488	4,437,261	2,013	16,558	8·22
1896	40	41	1,042	25·41	3,210	2,488,350	1,129	9,705	8·60
1897	1,514	1,538	23,666	18·64	1,080	6,574 858	2,982	24,283	8·14
1898	1,194	1,213	24,486	20·19	766	11,672,694	5,295	47,384	8·95

Year.	Chemicals. Salts of Potash. (*h*) Lb.	Kg.	Value.	Value per Kg.	Chloride of Lime. Lb.	Kg.	Value.	Value per Kg.	Nitrate of Soda. Lb.	Kg.	Value.	Value per Kg.
1895	379,767	172,260	$12,555	$0·07	10,600	4,808	$305	$0·06	2,428,160	1,101,413	$44,847	$0·04
1896	774,452	351,291	22,333	·06	5,500	2,495	105	·04	1,462,720	663,490	24,254	·04
1897	124,277	56,372	4,763	·08	40,916	18,559	718	·04	2,022,720	917,508	33,979	·04
1898	183,804	82,466	5,444	·07	28,605	12,975	381	·03	1,646,400	746,802	24,825	·03
1899	402,230	182,450	12,594	·07	*Nil.*	....	....	....	5,510,400	2,499,501	78,877	·03

Year.	Chemicals—*Continued.* Caustic Soda. Lb.	Kg.	Value.	Value per Kg.	Sal Soda and Soda Ash. Lb.	Kg.	Value.	Value per Kg.	All Other Salts of Soda. Lb.	Kg.	Value.	Value per Kg.
1895	1,592,150	722,199	$31,691	$0·04	269,765	122,365	$2,423	$0·02	68,127	30,902	$1,174	$0·04
1896	1,708,173	774,559	33,433	·04	470,102	213,238	4,098	·02	36,839	16,710	380	·02
1897	1,824,565	827,623	34,332	·04	2,246,181	1,018,868	9,128	·01	20,797	9,433	262	·03
1898	1,237,257	561,216	22,202	·04	4,247,945	1,926,855	14,255	·01	120,542	54,677	1,400	·03
1899	1,032,831	468,489	18,330	·04	1,173,781	532,424	4,193	·01	93,520	42,420	2,035	·05

Year.	Clays or Earths of All Kinds, including China Clay.				Coal, Bituminous.				Copper. Ore and Regulus.			
	Long Tons.	Metric Tons.	Value	Value per Met. Ton.	Long Tons.	Metric Tons.	Value	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Val. per Met. Ton.
1895	89	90	$740	$8·22	3,440	3,495	$7,699	$2·20	2,457,280	1,115	$111,007	$99·56
1896	82	83	109	3·30	5,204	5,287	7,990	1·51	1,075,200	488	32,715	67·04
1897	16	16	63	3·93	5,297	5,382	6,337	1·18	1,328,320	603	85,170	141·24
1898	91	92	657	7·11	2,890	2,936	2,675	0·91	7,622,720	3,458	547,960	158·46
1899	152	154	1,304	8·44	6,806	6,915	9,598	1·39	3,841,600	1,743	265,326	152·22

Year.	Copper—*Continued.* Pigs, Bars, Ingots, Old, and Other Unmanufactured.				Manufactures. Value.	Earthen, Stone, and China-ware. Value.	Fertilizers. Guano.			
	Lb.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	475,588	215	$43,193	$200·89	$11,626	$25,482	*Nil.*	.....	.....	.....
1896	*Nil.*	.....	.....	.....	7,307	28,267	26	26	$657	$25·27
1897	406,598	184	30,167	163·93	4,225	23,438	*Nil.*	.....	.....	.....
1898	23,647,968	10,727	1,437,464	134·00	4,637	30,646	*Nil.*	.....	.....	.....
1899	2,550,149	1,157	350,919	303·30	10,250	38,393	.....	.....	8	.....

Year.	Fertilizers—*Continued.* Phosphates, Crude or Native.				Other Fert'z's	Glass and Glassware.	Graphite.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Value.	Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1895	743	755	$12,775	$16·92	$2,154	$15,303	5	5	$305	$61·00
1896	*Nil.*	.....	.....	.....	1,111	13,761	6	6	296	46·00
1897	65	66	1,782	27·00	3,482	15,579	128	130	7,393	56·87
1898	10	10	306	30·12	13,499	13,805	165	168	11,645	69·32
1899	723	735	8,718	11·87	4,510	13,236	5	5	338	67·60

Year.	Iron and Steel, and Tin Plate. Pig Iron.				Scrap Iron and Steel, fit only to be remanufactured.				Bar Iron, Rolled or Hammered.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895	210	213	$2,536	$11·89	92	94	$1,024	$10·89	8,207	3	$288	$96·00
1896	599	609	12,220	20·06	240	244	2,160	8·85	19,107	9	610	67·77
1897	114	116	1,394	12·00	57	58	313	5·40	66,345	30	1,699	56·60
1898	581	590	7,098	12·03	63	64	270	4·22	48,151	22	1,648	74·91
1899	597	607	7,908	13·04	195	198	2,671	13·49	85,630	39	3,159	81·42

Year.	Iron and Steel, and Tin Plate—*Continued.* Railway Bars of Iron or Steel, or in Part of Steel.				Ingots, Blooms, Slabs, Billets and Bars of Steel, and Steel in Forms, N. E. S.				Sheet, Plate and Taggers, Wire Rod, Wire, and Wire Rope and Strand, Iron or Steel.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.	Lb.	Metric Tons.	Value.	Value per Met. Ton.
1895	204	207	$3,755	$18·14	43,143	19	$2,499	$131·52	$516,067	235	$12,026	$51·17
1896	*Nil.*	.....	.....	.....	*Nil.*	.....	.....	.....	3,102,098	1,407	46,381	32·96
1897	262	266	11,921	44·82	15,515	7	1,728	246·86	2,446,012	1,109	68,878	62·11
1898	140	142	3,479	24·50	100,417	46	3,829	83·24	824,733	374	21,231	56·77
1899	*Nil.*	.....	.....	.....	10,769	5	1,718	350·61	377,402	171	11,148	65·12

Year.	Iron and Steel, and Tin Plate—*Continued.*					Lead and Manufactures of. Value.	Marble and Stone and Manufactures of. Value. (*k*)	Metal Compositions and Manufactures. Value.	Mineral Substances. N. E. S. Value.
	Tin Plates, Terne Plates, and Taggers Tin.				Manufactures. Value.				
	Lb.	Metric Tons.	Value.	Value per Met. Ton.					
1895.....	666,458	302	$18,446	$61·08	$126,576	$713,971	$6,998	$66,389	$4,655
1896.....	1,674,187	760	40,290	53·84	114,472	1,504,967	14,297	43,757	.....
1897.....	2,951,473	1,339	79,179	59·21	165,710	2,066,923	15,156	32,791	.....
1898.....	899,757	408	22,271	54·59	279,717	2,599,734	5,452	58,810	.....
1899.....	378,866	172	11,308	65·82	346,395	2,942,497	7,525	55,250	.....

Year.	Oil, Mineral.				Paints and Colors. Value.	Precious Stones. Value.	Salt.			
	Gallons.	Liters.	Value.	Value per Liter.			Lb.	Met. Tons.	Value.	Value per M. Ton.
1895.....	23	87	$43	$0·49	$24,147	$22,967	3,116,595	1,414	$6,500	$4·59
1896.....	*Nil.*	.....	.....	.....	23,703	18,979	7,088,184	3,192	14,947	4·68
1897.....	*Nil.*	.....	.....	.....	15,473	26,696	5,074,995	2,302	10,189	4·43
1898.....	3,219	12,185	486	·04	15,854	39,974	4,687,286	2,189	4,751	2·17
1899.....	*Nil.*	.....	.....	.....	12,104	49,256	5,216,112	2,366	9,858	4·17

Year.	Sulphur or Brimstone (Crude).				Tin, in Bars, Blocks, Pigs, Grain or Granulated.				Zinc and M'f'res
	Long Tons.	Met. Tons.	Value.	Value per M. Ton.	Lb.	Met. Tons.	Value.	Value per Met. Ton.	Value.
1895.....	220	224	$3,685	$16·45	395,456	184	$63,584	$474·50	$3,873
1896.....	484	492	8,535	17·34	289,266	131	41,874	319·64	215
1897.....	194	197	3,992	20·20	869,929	395	118,882	300·97	2,391
1898.....	1,414	1,437	31,822	22·14	740,326	336	113,334	337·30	1,536
1899.....	477	485	10,804	22·29	999,315	453	265,188	585·02	1,604

(*a*) From *Summary of Commerce and Finance of the United States.*.
(*b*) Includes lime.
(*c*) Ore, so called, consisting chiefly of matte.
(*d*) Sheets are not included these years, but are reported with manufactures.
(*e*) Total exports of coin and bullion; that is, includes both domestic and foreign not elsewhere specified.
(*f*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte which is classified as copper ore and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin.
(*g*) Including nickel oxide and matte.
(*h*) Includes chlorate, muriate, and nitrate of potash, and all other salts of potash.
(*j*) Reported in bbls. and reduced to gals. at 42 gals. = 1 bbl.
(*k*) Including slate.
(*m*) Calculated at 1 bbl. = 400 lb.
(*n*) Not enumerated.
(*o*) Not reported separately. Included with crude phosphates.
(*p*) Not reported separately. Included with antimony.

NOTE.—N. E. S. signifies not elsewhere specified.

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The
Buyers'
Manual

Preface.

THE advertising pages of the annual volumes of this work will well repay the careful perusal and study of every reader who wishes to be well informed upon the present condition of the mineral industry. They give an admirable and practical insight into the present state of the mining and metallurgic arts, for in them nearly every manufacturer or dealer of note in this country advertises the machines, appliances, and processes which are now in vogue, or which it is sought to introduce; while the names and specialties of the most eminent members of the engineering professions, as indicated in their cards, show the direction of modern mining and metallurgical progress.

These advertising pages are no less important to those who desire a clear knowledge of the means by which this country has come to be far the most important producer of minerals and metals than to those who wish to know where to get that full and reliable information concerning the values of properties, machinery, processes, and products which should precede the investment of capital.

Every country in the world is wisely striving to develop its mineral resources, and to build up its mineral industry, and in all of them this volume, which gives the latest and best practice in every department of the industry, has become indispensable. It is constantly consulted for the best technical skill and the most advantageous machinery and appliances in use. All enterprising manufacturers of such wares appreciate this fact, and know that in no other way can they so effectively bring their goods before those who may need them as through the advertising pages of THE MINERAL INDUSTRY. These pages have become a veritable directory of the best in everything relating to the industry, and American, German and English manufacturers there compete for the orders of the whole world. $1,000,000,000 a year is certainly not an overestimate of the annual expenditure for technical skill, machinery and supplies used in the industry of which THE MINERAL INDUSTRY and *The Engineering and Mining Journal* are not only the chief but the only universal representatives published in any language.

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Mecklenberg Iron Works.
Rand Drill Co.
(See Machinery.)

Coal-Washing Machinery.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Coke Ovens.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Coke Works Equipments.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Compressors.
Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Krupp, Fried, Grusonwerk.
Mecklenberg Iron Works.
Rand Drill Co.
(See Machinery.)

Concentrators.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Condensers.
(See Machinery.)

Contractors' Supplies.
Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
Rand Drill Co.

Converters.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Conveying Belts.
Jeffrey Mfg. Co.
Robins Conveying Belt Co.
(See Elevating and Conveying Machinery.)

Conveying Machinery.
(See Elevating and Conveying Machinery.)

Copper.
(See Metals.)

Copper Bottoms.
(See Metals.)

Copper Dealers and Producers.
(See Metals, Copper.)

Cranes.
(See Elevating and Conveying Machinery.)

Crucibles.
Braun, F. W. & Co.
Hoskins, Wm.
(See Assayers' and Chemists' Supplies.)

Crusher Plants.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Crusher Rolls.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Crushers.
Allis, Edw. P., Co.
Bacon, Earle C.
Braun, F. W. & Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenburg Iron Works.
(See Machinery.)

Cupel Machines.
Braun, F. W. & Co.

Cyanide Mills.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried, Grusonwerk.
(See Machinery.)

Cyanide Tanks.
Allis, Edw. P., Co.

Diamond Drills.
Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
(See Drills.)

Disintegrators.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Pulverizers.)

Drawing Materials.
Devoe, F. W., & C. T. Raynolds Co.
(See Mathematical Instruments.)

Dredging Machinery.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
(See Machinery.)

Drills.
Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Leyner, J. Geo.
Mecklenberg Iron Works.
Rand Drill Co.
(See Machinery.)

Drill Steel.
(See Metals—Mine Supplies.)

Drying and Calcining Machinery.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Dumping Cars.
(See Cars.)

Dynamite.
(See Explosives.)

Dynamos.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
(See Machinery.)

Electric Coal Mining Machines.
Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried, Grusonwerk.
(See Coal Mining Machinery.)

Electric Machinery and Supplies.
(See Machinery.)

Elevating and Conveying Machinery.
Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Robins Conveying Belt Co.
(See Machinery.)

Engineering Instruments.
Devoe, F. W., & C. T. Raynolds Co.

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Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
Rand Drill Co.
Weber Gas and Gasoline Engine Co.
Witte Iron Works.
(See Machinery.)

Engravers.
Poates, L. L.
Struppmann, C. & Co.

Etching Plates.
Matthiessen & Hegeler Zinc Co.

Excavating Machinery.
(See Steam Shovels—Machinery.)

Explosives.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.

Fans.
(See Machinery and Ventilating Machinery.)

Feeders.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Ore Feeders—Quicksilver Feeders—Machinery.)

Ferro-Nickel.
Orford Copper Co.
(See Metals.)

Filter Press
Johnson, J. & Co.

Fire Brick.
Braun, F. W. & Co.
Hoskins, Wm.

Forgings.
(See Metals—Iron and Steel.)

Frankiinite Ore.
New Jersey Zinc Co.
(See Ore and Zinc Ore.)

Furnaces.
Allis, Edw. P., Co.
Braun, F. W. & Co.
Fraser & Chalmers, Ltd.
Hoskins, Wm.
Krupp, Fried., Grusonwerk.
Nichols Chemical Co.
(See Assay Furnaces and Machinery.)

Fuse.
(See Explosives and Mill, etc., Supplies.)

Gas and Gasoline Engines.
Weber Gas and Gasoline Engine Co.
Witte Iron Works Co.
(See Machinery.)

Gas Producers.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.

Gauges, Recording.
(See Machinery.)

Gearing.
(See Machinery.)

Gold.
(See Metals.)

Hauling Machinery.
(See Machinery.)

Hardware, Miscellaneous.
Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.

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Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
Rand Drill Co.
Weber Gas and Gasoline Engine Co.
Witte Iron Works Co.
(See Machinery.)

Hydraulic Machinery.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Mecklenberg Iron Works.
(See Machinery.)

Insulated Wires and Cables.
(See Wire and Wire Cables—Electrical Supplies.)

Iron.
(See Metals.)

Jigs.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Concentrators.)

Lamps, Miners', Safety, Etc.
Allis, Edw. P., Co.
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Lead.
(See Metals.)

Lead Linings (Chemical and Chlorinating.)
(See Mill, etc., Supplies.)

Link Belts.
Allis Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
(See Belting—Machinery.)

Litharge.
(See Assay Furnaces—Apparatus and Supplies—Chemicals.)

Lithographers' Plates.
Matthiessen & Hegeler Zinc Co.

Locomotives.
Allis, Edw. P., Co.
Baldwin Locomotive Works.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
(See Cars and Machinery.)

Lubricants.
(See Mill, etc., Supplies.)

Lubricating Oils.
(See Oils.)

Lubricators, Oilers and Grease Cups.
(See Mill, etc., Supplies.)

Machinery.
(Makers and Dealers in Mining, Smelting, and other Machinery and Supplies.)
Allis, Edw. P., Co.
Bacon, Earle C.
Braun, F. W. & Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Hoskins, Wm.
Jeffrey Mfg. Co.
Johnson, J. & Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
Nichols Chemical Co.
Rand Drill Co.
Robins Conveying Belt Co.
Weber Gas and Gasoline Engine Co.
Witte Iron Works Co.

Machine Tools and Supplies.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.

Machinists.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.

Mathematical Instruments
Devoe, F. W., & C. T. Raynolds Co.
(See Drawing Materials.)

Matte.
American Metal Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Mountain Copper Co.
Nichols Chemical Co.
Orford Copper Co.
Ricketts & Banks.
United Metals Selling Co.
(See Metals—Ores.)

Metal Brokers and Dealers.
(See Metals.)

Metallurgical Works & Ore Purchasers.
American Metal Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Matthiessen & Hegeler Zinc Co.
Orford Copper Co.
Ricketts & Banks.
United Metals Selling Co.

Metallurgists.

Ricketts & Banks.
Simonds & Wainwright.
(See Professional Directory.)

Metals.

Aluminum.
American Metal Co.
Bath, Henry & Son.

Antimony.
American Metal Co.
Bath, Henry & Son.

Bearing Metals.
Allis, Edw. P., Co.

Copper.
American Metal Co.
Arizona Copper Co.
Atlantic Mining Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Canadian Copper Co.
Copper Queen Consolidated Mining Co.
Detroit Copper Mining Co.
Montana Ore Purchasing Co.
Mountain Copper Co.
Nichols Chemical Co.
Orford Copper Co.
Phelps, Dodge & Co.
Stanton, John.
United Metals Selling Co.
Wolverine Copper Mining Co.

Ferromanganese (See Spiegeleisen.)
American Metal Co.
New Jersey Zinc Co.

Gold.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Montana Ore Purchasing Co.

Iron (See Iron Ore.)
Bath, Henry & Son.
Krupp, Fried., Grusonwerk.

Lead.
American Metal Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
United Metals Selling Co.

Nickel.
American Metal Co.
Bath, Henry & Son.
Canadian Copper Co.
Orford Copper Co.

Platinum.
Baker & Co.
Bath, Henry & Son.

Silver.
American Metal Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Montana Ore Purchasing Co.
Orford Copper Co.

Spelter (See Zinc.)
American Metal Co.
Bath, Henry & Son.
New Jersey Zinc Co.
United Metals Selling Co.

Spiegeleisen (See Ferromaganese.)
American Metal Co.
New Jersey Zinc Co.

Steel.
Allis, Edw. P., Co.
Bath, Henry & Son.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.

Tin.
American Metal Co.
Bath, Henry & Son
United Metals Selling Co.

Zinc (See Spelter.)
American Metal Co.
Bath, Henry & Son.
Matthiessen & Hegeler Zinc Co.
New Jersey Zinc Co.
United Meta's Selling Co

Mill, Factory, Smelter & Mine Supplies.

Allis, Edw. P., Co.
Bacon, Earle C.
Braun, F. W. & Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Hoskins, Wm.
Jeffrey, Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
Rand Drill Co.
Roebling's, J. A., Sons & Co.
Weber Gas and Gasoline Engine Co.
Witte Iron Works Co.

Mills.

(See Stamp Mills—Machinery.)

Millstones.

(See Mill, Factory, Smelter and Mine Supplies.)

Mine Cars and Car Wheels.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
(See Cars.)

Mine Supplies.

(See Mill, etc, Supplies.)

Mine Harness.

(See Mill, etc, Supplies.)

Miners' Lamps.

(See Lamps.)

Mining Companies.

Atlantic Mining Co.
Canadian Copper Co.
Copper Queen Consolidated Mining Co.
Detroit Copper Mining Co.
Montana Ore Purchasing Co.
Mountain Copper Co.
Wolverine Copper Mining Co.

Mining Machinery, General.

Allis, Edward P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Hoskins, Wm.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Leyner, J. Geo.
Lidgerwood Mfg. Co.
Mecklenberg Iron Works.
Rand Drill Co.
Robins Conveying Belt Co.

Motors.

(See Machinery.)
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.

Nails.

(See Mill, etc., Supplies.)

Nickel.

(See Metals.)

Nickel Oxide.

Orford Copper Co.

Nitrate of Soda.

Bath, Henry & Son.

Oils and Greases, Illuminating and Lubricating.

(See Mill, etc., Supplies.

Ore Breakers.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Breakers—Crushers—Machinery.)

Ore Buyers.

American Metal Co.
Balbach Smelting and Refining Co.
Bath, Henry & Son.
Mariner & Hoskins.
Montana Ore Purchasing Co.
Nichols Chemical Co.
Orford Copper Co.
Ricketts & Banks.
United Metals Selling Co.
(See Sampling Works and Smelters.)

Ore Cars.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Cars and Machinery.)

Ore Concentrators.

(See Concentrators—Machinery.)

Ore-Dressing Machinery.

(See Concentrators.)

Ore Feeders.

Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Feeders and Machinery.)

Ore-Handling Machinery.

(See Machinery.)

Ore Roasters.

(See Furnaces—Roasting Furnaces.)

Ore Samplers.

(See Machinery—Sampling Machinery.)

Ore-Testing Works.

Ricketts & Banks.
(See Sampling and Testing Works.)

Ore Washers.

(See Machinery.)

Oxides.

New Jersey Zinc Co. Orford Copper Co.
(See Copper Oxide—Lead Oxide.)

Packings.

(See Rubber Goods—Mechanical, Mill and Mine Supplies—Machinery.)

Paints.

Devoe, F. W., & C. T. Raynolds Co.
New Jersey Zinc Co.

Periodicals.

(See Publications.)

Perforated Metals.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Machinery—Screens.)

Picks.

(See Mill, etc., Supplies.)

Pile Drivers.

Lidgerwood Mfg. Co.
(See Machinery.)

Pipe.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Tubes.)

Pipe Machinery.

(See Machinery.)

Placer Machinery.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
(See Machinery.)

Platinum.

(See Metals.)

Platinum Apparatus.

Baker & Co.

Power Transmission Machinery.

(See Machinery.)

Pressure Gauges.

(See Machinery.)

Processes.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Prospecting Drills.

(See Drills.)

Publications.

Columbian Parliamentary Compend.
Cyanide Process, Practical Notes on.
Engineering and Mining Journal.
Gems and Precious Stones of North America.
Lead and Copper Smelting and Copper Converting.
Manufacture and Properties of Structural Steel.
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Mining and General Telegraphic Code.
Modern Copper Smelting.
New Basis for Chemistry.
Ore Deposits of the United States and Canada.
Phosphates of America.
Qualitative Analysis, An Outline of.
Qualitative Blowpipe Analysis, A Manual of.
Stamp Milling of Gold Ores.
Universal Bimetallism.

Publishers.

Scientific Publishing Co.

Pulsometers.

(See Pumps—Machinery.)

Pulverizers.

(See Crushers—Machinery.)

Pumps.

Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Quarrying Machinery.
Allis, Edw. P., Co.
Bacon, Earle C.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
Rand Drill Co.
(See Machinery.)

Quartz Mills.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Quicksilver Feeders, Furnaces, Etc.
(See Machinery.)

Rails.
(See Metals—Iron and Steel.)

Railways.
(See Tramways.)

Railway Equipment and Supplies.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.

Refineries.
Balbach Smelting & Refining Co.
Nichols Chemical Co.
Ricketts & Banks.
(See Smelters.)

Regulators, Dampers, etc.
(See Mill, etc., Supplies.)

Retorts.
(See Assay Supplies, Mill Supplies.)

Roasting Furnaces.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
Nichols Chemical Co.
(See Furnaces.)

Rock Breakers.
(See Breakers—Crushers—Ore Breakers—Machinery.)

Rock Drills.
(See Drills—Machinery.)

Roofing.
(See Mill, etc., Supplies.)

Rubber Goods.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.

Safety Lamps.
(See Lamps.)

Sampling and Testing Works.
Mariner & Hoskins.
Ricketts & Banks.
United Metals Selling Co.

Sampling Machinery.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Saw Mill Machinery.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
(See Machinery.)

Screens.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Perforated Metals.)

Sectional Track.
(See Machinery.)

Shafting and Pulleys.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Sheet Copper.
(See Metals.)

Sheet Lead.
(See Metals.)

Shoes and Dies.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Mill Supplies—Machinery.)

Shovels, Hand.
(See Mill, etc., Supplies.)

Shovels, Steam.
(See Steam Shovels.)

Silver.
(See Metals.)

Skips.
(See Cages and Skips—Machinery.)

Smelting Furnaces.
(See Furnaces.)

Smelting Works.
Baker & Co.
Balbach Smelting and Refining Co.
Canadian Copper Co.
Matthiessen & Hegeler Zinc Co.
Montana Ore Purchasing Co.
Nichols Chemical Co.
Orford Copper Co.
Ricketts & Banks.
United Metals Selling Co.
(See Refineries.)

Sorting Belts.
Jeffrey Mfg. Co.
Robins Conveying Belt Co.

Spelter.
(See Metals.)

Spiegeleisen.
(See Metals.)

Sprocket Wheels.
Jeffrey Mfg. Co.
(See Machinery.)

Stamp Mill Machinery.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Machinery.)

Steam Separators.
(See Machinery.)

Steam Shovels.
(See Dredges—Machinery.)

Steel.
(See Metals.)

Steel Cams.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Work.
(See Machinery.)

Stove Blanks (Zinc).
Matthiessen & Hegeler Zinc Co.

Structural Iron and Steel.
(See Corrugated Iron and Steel.)

Sulphuric Acid.
Matthiessen & Hegeler Zinc Co.
(See Chemicals.)

Tanks.
(See Mill, etc., Supplies.)

Tappets.
(See Stamp Mills—Machinery.)

Testing Works.
(See Sampling and Testing Works.)

Tin.
(See Metals.)

Tipples.
(See Machinery.)

Tool Steel.
(See Metals—Steel.)

Tramways.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Krupp, Fried., Grusonwerk.
Lidgerwood Mfg. Co.
(See Cableways—Machinery.)

Turbines and Water Wheels.
(See Machinery.)

Valves.
(See Machinery.)

Valves, Rubber.
(See Rubber Goods.)

Vanners.
(See Concentrators—Machinery.)

Varnishes.
Devoe, F. W., & C. T. Raynolds Co.

Ventilators.
Allis, Edw. P., Co.
Bullock, M. C., Mfg. Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
Mecklenberg Iron Works.
(See Blowers—Fans.)

Washboard Blanks (Zinc).
Matthiessen & Hegeler Zinc Co.

Waterwheels and Turbines.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Krupp, Fried., Grusonwerk.
(See Machinery.)

Well Drilling Machinery.
(See Machinery.)

Wheelbarrows.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
(See Mine Supplies.)

Wheels, Car.
(See Cars.)

Whims, Hoists.
Allis, Edw. P., Co.
Bacon, Earle C.
Fraser & Chalmers, Ltd.
Mecklenberg Iron Works.

White Lead.
(See Paints.)

Wire and Wire Rope Cables.
Allis, Edw. P., Co.
Fraser & Chalmers, Ltd.
Jeffrey Mfg. Co.
Lidgerwood Mfg. Co.
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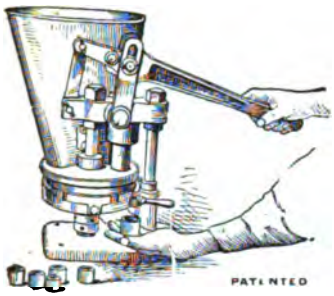

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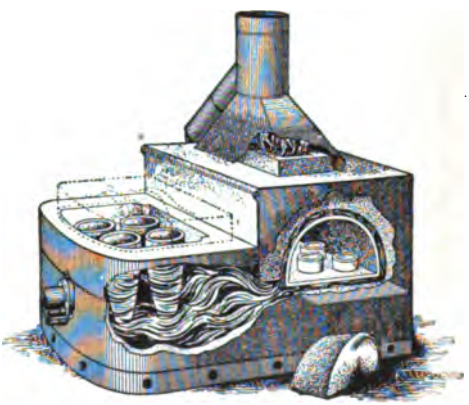

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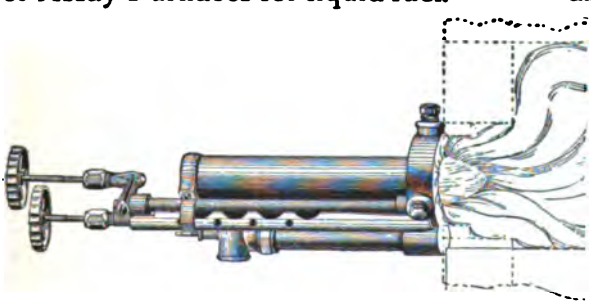

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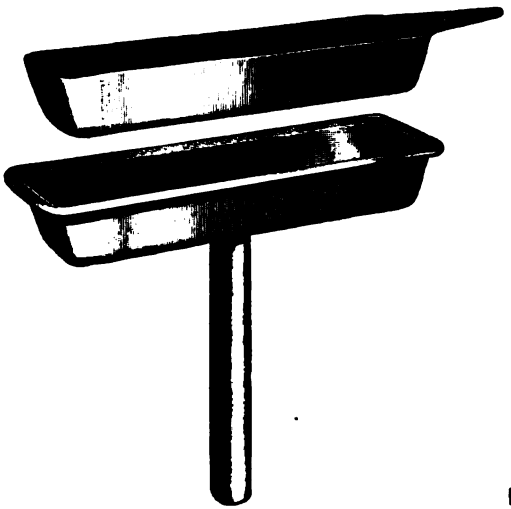

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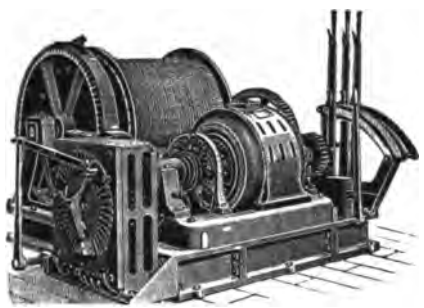

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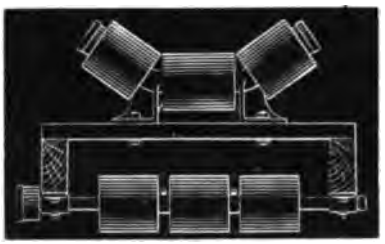

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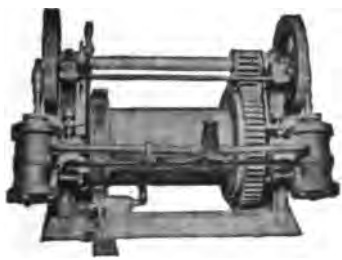

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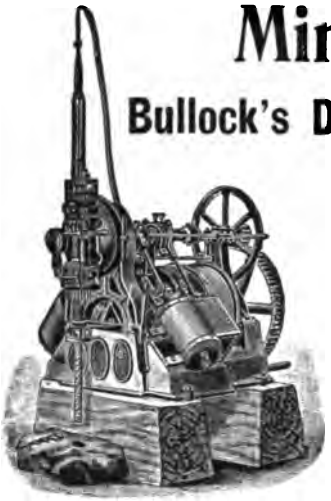

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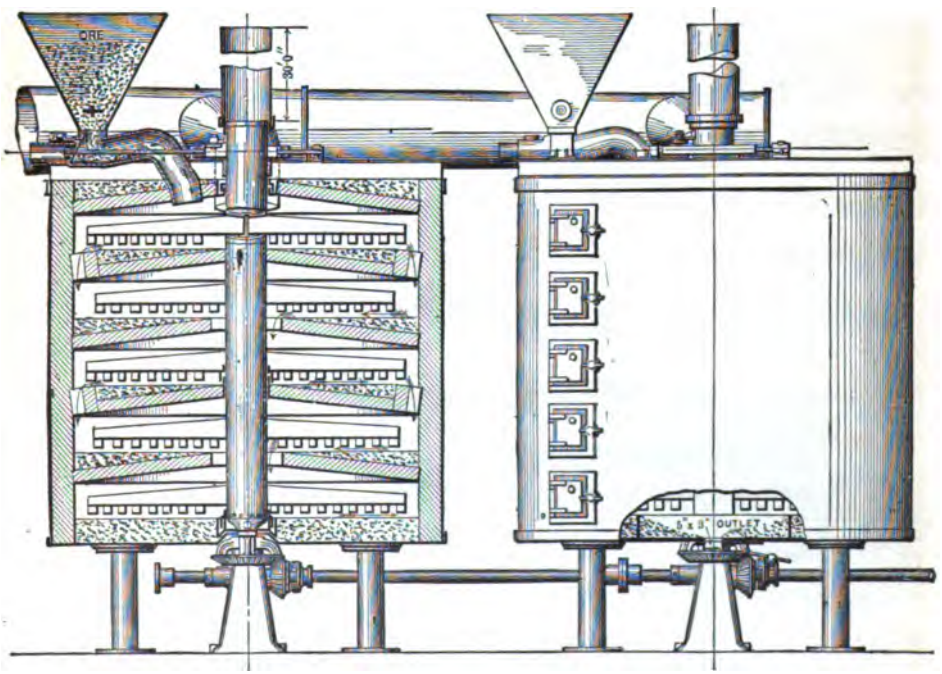

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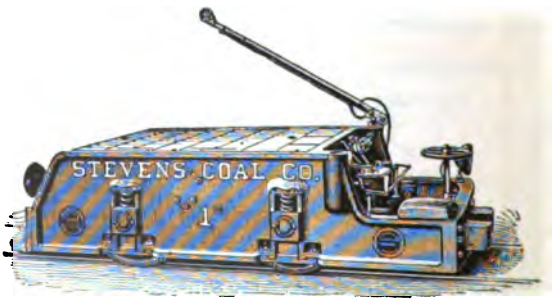

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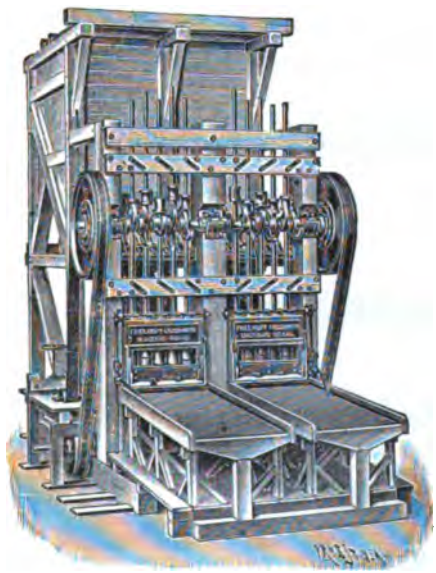

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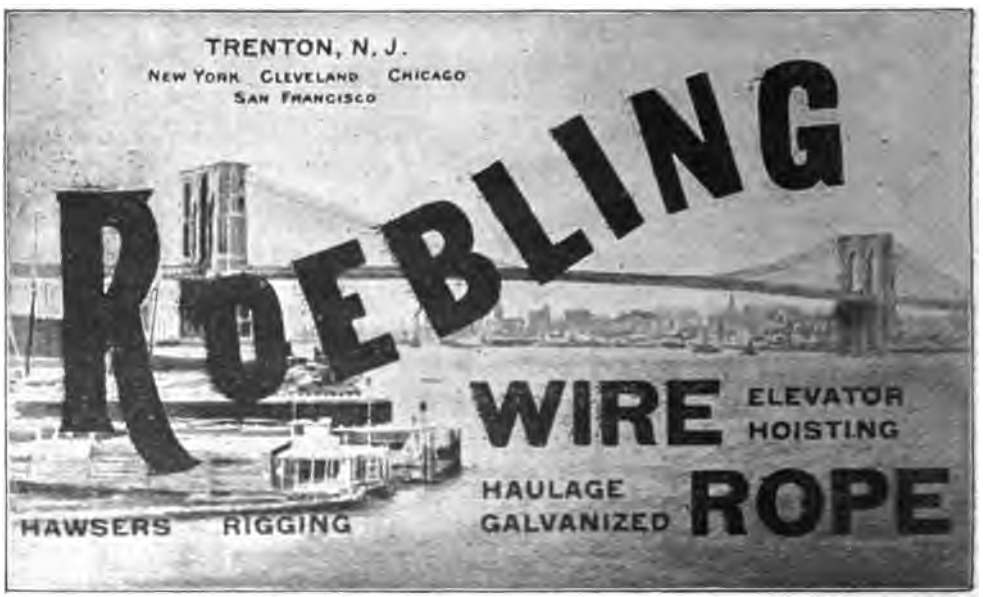

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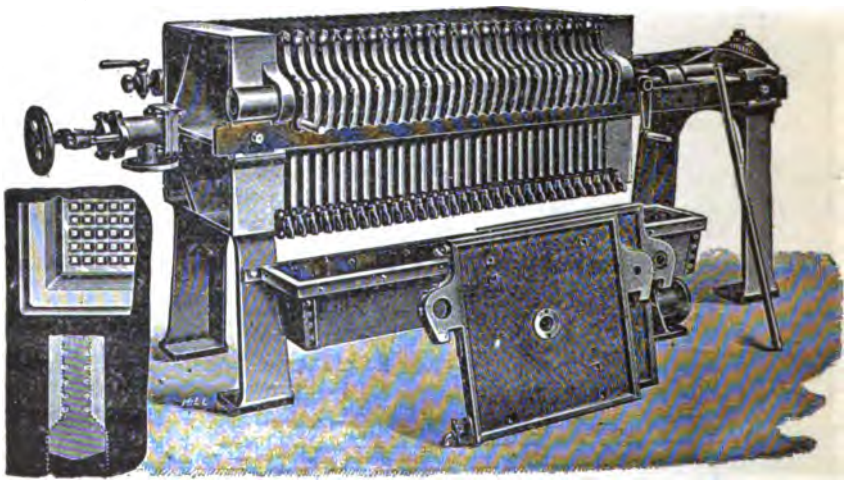

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